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Bulletin 127

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY
BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

GOLD DREDGING IN THE UNITED STATES

BY

CHARLES JANIN

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CONTENTS.

Page.

Introduction........ 3

Acknowledgments........ 4

History........ 4

  Early dredges........ 5

    Spoon dredge........ 5

    Steam dredges........ 6

    Dredging in California and Montana........ 6

    Bucket-elevator dredge........ 8

    Progress in design and equipment........ 9

    Gold produced by dredging........ 10

    Dredging in cold regions........ 13

    Evolution of gold dredges........ 13

Gold-dredging districts in the United States........ 13

  California........ 14

    Oroville and Hammonton districts........ 14

    Other districts........ 14

    Present status of dredging........ 15

  Montana........ 15

  Colorado........ 15

  Idaho........ 15

  Oregon........ 16

  Alaska........ 16

  Philippine Islands........ 16

Topography and geology of dredging areas........ 17

  California dredging districts........ 17

    Chief districts........ 18

      Oroville field........ 19

      Yuba field........ 20

      Folsom field........ 20

  Colorado dredging fields........ 21

  Montana dredging fields........ 22

  Idaho and Oregon placer deposits........ 23

  Placer areas in Philippine Islands........ 23

  Yukon dredging fields........ 24

Prospecting dredging ground........ 26

  Features that must be determined........ 27

  Size of prospecting shafts........ 29

  Prospecting by shaft and drill holes........ 30

  Handling of bits........ 32

  Use of sand pump........ 32

  Procedure in drilling........ 32

  Handling of drill core........ 33

  Panning........ 35

  Prospecting records........ 38

  Speed of drilling........ 41

  Cost of drilling........ 41

  Placing of prospect holes........ 42

  Volume of drillings........ 43

  Determining gold content of ground........ 44

    Factors used........ 44

Prospecting dredging ground—Continued. Page.
- Types of hand drills ... 49
  - Empire drill ... 49
- New type of light power drill ... 50
- Determining period of dredging project ... 50
- Selection of dredge for a given project ... 51
- Use of unusual type of drill in Alaska ... 52

Details of construction of gold dredges ... 54
- Determining suitable size of dredge ... 54
- Bucket line and tumblers ... 54
- Pins ... 56
- Lower tumbler ... 57
- Upper tumbler ... 59
- Main drive ... 60
- Winches ... 61
  - Ladder-hoist winch ... 62
  - Swing winch ... 63
- Spuds and spud supports ... 65
- Hulls and superstructures ... 68
- Bow gantry ... 70
- Ladder rollers and bearings ... 71
- Main hopper, save-all, and bucket idlers ... 71
- Screen and screen drive ... 73
- Distributor ... 77
- Stacker ... 77
- Belts ... 78
- Guide rollers ... 82
- Wire ropes ... 83
- General details of dredge construction ... 85

Special steels for dredge parts ... 90
- Properties of alloys of steel ... 90
- Manganese steel ... 90
- Nickel steel ... 91
- Nickel-chrome steel ... 92
- High-carbon steel ... 92

Electrical equipment on dredges ... 93
- Bucket-drive motor ... 93
- Power consumption by electrical equipment ... 94
- Recent type of direct-connected motor drive ... 97
- Kind of electric current used ... 98
- Voltage of current used on dredges ... 99

Rebuilt dredges ... 107
- Feasibility of moving dredges or machinery ... 107
- Examples of second use of dredges or machinery ... 107
  - Tin Cup dredge ... 107
  - Scott River dredge ... 108
  - Butte dredge ... 108
  - El Oro No. 1 dredge ... 109
  - Dredge near Leadville, Colo ... 109
  - Castle Creek dredge ... 109
  - Features to be considered in removal of dredging equipment ... 110
  - Other examples of moving dredges ... 110

Operation of gold dredges ... 113
- Essentials of dredge operation ... 113
- Ample spare parts should be available ... 113

Operation of gold dredges—Continued.

Page.
Digging on spuds........ 114
Duties and requirements of winchmen........ 116
Regulating speed of bucket swing........ 116
Practice at Ruby, Mont........ 118
Bucket-cleaning methods........ 119
Use of monitors and of blasting in cemented gravel........ 120
Life of hull........ 121
Repairing and maintaining hulls........ 122
Handling of large bowlders........ 123
Night lights........ 124
Method of making repairs........ 124
Use of depth indicator chart........ 125
Extent and causes of lost time........ 127
Water consumption in gold dredging........ 130
Accident prevention on gold dredges........ 133
Gold-saving appliances........ 138
Use of jigs........ 139
Gold-saving equipment on two Natoma dredges........ 139
Character of gold-saving tables........ 141
Consumption of quicksilver........ 141
Factors affecting efficiency of tables........ 143
Loss of gold in suspension........ 146
Cleaning of tables........ 147
Gold saving in Yuba field........ 148
Working costs........ 150
Requirements in cost accounting........ 150
Costs for same dredge may vary by years........ 153
How operating costs may indicate inefficiency........ 154
Cost figures for certain dredges........ 154
Accounting methods of Natomas Consolidated........ 163
Labor costs........ 162
Dredge operating costs, Natoma division........ 164
Miscellaneous cost tables and forms........ 165
Recovery of gold compared to content indicated by prospecting........ 180
Methods of prospecting........ 181
Factors used in calculations........ 181
Recovery from different parts of table........ 182
Gold in tailings........ 184
Examples of recoveries by different dredges........ 185
Faulty prospecting a cause of seemingly poor recovery........ 188
Recoveries from redredged ground........ 189
Recovery of gold at Oroville, Cal........ 190
Recovery at Snelling, Cal........ 192
Power plants for dredge installations in isolated districts........ 193
Example of fuel saving by redesigning power plant........ 194
Power cost on two dredges on Seward Peninsula........ 195
Novel power plant of Philippine dredge........ 196
Reclamation of land after dredging in California........ 198
Hydraulic mining ended by legislation........ 198
Dredging in California not inimical to agriculture........ 198
Size of California dredging areas........ 199
Comparative income from dredging........ 200
Resoiling methods tried........ 201
Land reclamation demands dredge changes........ 203

Page.
Future of dredging 204
  Feasibility of redredging certain areas 204
  Alaska fields promising 206
  Dredging outlook in other countries 206
  Successful dredging operations demand competent experts 208
List of dredging operations 209
Selected bibliography on gold dredging 212
Publications on metal mining 217
  Publications available for free distribution 217
  Publications that may be obtained only through the Superintendent of Documents 219

ILLUSTRATIONS.

Page.
PLATE I. *A*, New Zealand spoon dredge, 1865; *B*, Current-wheel dredge, New Zealand, 1882 6
II. *A*, Model of dredge used by natives in Philippine Islands; *B*, Face of cut in ground along Mokelumne River, Cal 7
III. *A*, New Zealand dredge using two screens; *B*, Earnsclengh No. 2 dredge, Clutha River, New Zealand 8
IV. *A*, The Fielding L. Graves, first successful bucket-lift dredge in the United States, 1894; *B*, Side view of A. F. Graeter dredge used at Bannack, Mont., started 1895; *C*, Bucket line on A. F. Graeter dredge 10
V. *A*, Dredge built at Breckenridge, Colo., in 1898; *B*, Old method of working placer ground below Oroville, Cal 11
VI. *A*, General view of the Yuba field, Cal.; *B*, Dredging ground in the Folsom field, Cal 20
VII. *A*, Dredge pit near Ruby, Mont.; *B*, Boston and Idaho dredge in Boise Basin, Idaho 22
VIII. *A*, Overburden on placer ground, Nome district, Alaska.; *B*, Driller, panman, and engineer at work near Ruby, Mont; Keystone drill 24
IX. *A*, Thawing with steam points on Bonanza Creek, Yukon Territory; *B*, Prospecting with a Keystone drill 25
X. Section showing logs of drill holes in dredging ground 26
XI. Working parts of Empire prospecting drill 48
XII. *A*, Keystone traction drill descending a river bank; *B*, Keystone nontraction drill fording a river 50
XIII. *A*, Operating light power prospecting drill; *B*, Recent type of light power drill being moved 51
XIV. *A*, Low-back bucket for digging bowlders, also showing ladder roller; *B*, Base of new type bucket for use with round lower tumbler; *C*, Manganese steel bucket of 16 cubic feet capacity, Conrey No. 4 dredge, shows full circular bushing in back eye 54
XV. Detail drawing of a 16-foot bucket 56
XVI. *A*, One-piece manganese steel bucket with special type of lip; *B*, Bucket with eyes elongated by wear; *C*, Bucket with inserted plate of manganese steel to resist wear on tumbler. 56
XVII. Original California type of buckets 56

Page.

PLATE XVIII. *A*, Recent type of 16-foot bucket with bottom and hood cast as one piece; *B*, Bucket used on a Panama Canal dredge.......... 57
XIX. *A*, Six-foot bucket of Guaymas dredge; *B*, Round lower tumbler; *C*, Hexagonal tumbler made round by adding cast steel segments.......... 58
XX. *A*, Bearing of lower tumbler air dredge at Ruby, Mont.; *B*, Hexagonal lower tumbler with device to prevent ladder from spreading.......... 58
XXI. Cushion-plate type of upper tumbler.......... 59
XXII. *A*, Hexagonal upper tumbler with three-piece wearing plates, Conrey No. 4 dredge; *B*, One-piece upper tumbler, Yuba No. 14 dredge.......... 60
XXIII. *A*, Hauling a large tumbler; *B*, "Bull gear" ready for shipment; *C*, Intermediate gears, herring-bone type.......... 60
XXIV. Milling the teeth of a bull wheel.......... 60
XXV. *A*, Winch room on Conrey No. 4 dredge; *B*, Winch room and levers on Yuba No. 4 dredge.......... 61
XXVI. Starboard winch for 9-foot dredge.......... 62
XXVII. General arrangement of winches on a 9-foot dredge.......... 62
XXVIII. Ladder-hoist winch for 9-foot dredge.......... 63
XXIX. *A*, Ladder-hoist winch for large dredge, direct connected; *B*, Standard type of swing winch.......... 64
XXX. *A*, Cast-steel round point for spud; *B*, Steel spud, with short square head.......... 65
XXXI. *A*, Hauling a spud; *B*, Equalizing support for spuds.......... 66
XXXII. Elevation of wooden hull of a 6-foot dredge.......... 68
XXXIII. Hull timbers of a 6-foot dredge.......... 68
XXXIV. *A*, Steel hull with bevelled stern, Yuba No. 15 dredge; *B*, Steel ribs, bottom and deck beams, 8-foot dredge.......... 70
XXXV. Steel hull under construction.......... 70
XXXVI. *A*, Yukon No. 9 dredge under construction; *B*, Superstructure of a steel dredge.......... 70
XXXVII. *A*, Plate-girder digging ladder; *B*, Save-all sluice, Conrey No. 2 dredge.......... 71
XXXVIII. *A*, Ladder-roller bearing suspended on lugs,; *B*, Ladder-roller bearing, pivoted type; *C*, Shears for bow line.......... 72
XXXIX. *A*, Grizzly over save-all sluice; *B*, Screen at Union dredge showing drive at upper end.......... 72
XL. One-piece bucket idler.......... 72
XLI. *A*, Revolving screen, segmental step type; *B*, Cylindrical screen, 50½ feet long and 9 feet in diameter, for 15-foot dredge; *C*, Screen for Conrey No. 4 dredge.......... 73
XLII. *A*, Housing and distributor from above, Yuba No. 13 dredge; *B*, Manganese-steel screen plates.......... 74
XLIII. *A*, Screen drive, single-roller type; *B*, Screen housing and distributor.......... 75
XLIV. *A*, Top of distributor; *B*, Flat type of stacker belt.......... 76
XLV. *A*, Side view of distributor, Yuba dredge; *B*, Side view of distributor.......... 77
XLVI. *A*, Stacker winch, double geared for 9-foot dredge; *B*, Idlers, three-roller type, for stacker belt.......... 78
XLVII. Yuba No. 16 dredge, with double tailing stacker.......... 79
XLVIII. Deck plan of a 7½-foot dredge.......... 80

Page.
PLATE XLIX. Elevation of a 7½-foot dredge.............................. 80
L. Plan of equipment on 8-foot Nambuloa dredge........... 80
LI. Elevation of equipment on 8-foot Nambuloa dredge....... 82
LII. Flotation diagram of a 5½-foot dredge........................ 84
LIII. *A*, Yuba No. 15 dredge, 17 foot buckets; *B*, Gold-saving tables and sluices, No. 4 dredge.............................. 98
LIV. *A*, Digging motor of Conrey No. 3 dredge, with cut steel gear; *B*, Power-cable floats, El Oro dredge............. 99
LV. Sections of Conrey No. 4 dredge........................... 104
LVI. Sketch of Conrey No. 3 sluice dredge..................... 105
LVII. *A*, Partly filled buckets, hard ground; *B*, Full buckets, loose ground.. 118
LVIII. *A*, Digging hard cemented gravel; *B*, Large bowlders dug at Natoma... 119
LIX. *A*, Trouble with the bucket line; *B*, Water nozzles in pipe in interior of revolving screen............................ 124
LX. *A*, Changing buckets; *B*, Interior of a revolving screen, showing end placing of water nozzles................. 124
LXI. *A*, Wreck of Viloro dredge; *B*, Shaking screen and gold tables, Union dredge, Folsom field.................... 136
LXII. Conrey No. 3 dredge in winter............................ 158
LXIII. *A*, Sluice box and soil chute of Hinnomunje dredge, Omeco, N. Z., *B*, Tailing distributor on Natoma No. 4 dredge.. 202
FIGURE 1. Patented device for pulling drill-hole pipe...................... 35
2. Details of rocker.. 36
3. Diagram showing weight to be given results from different holes.. 46
4. Map showing arrangement of drill-holes in a certain area.......... 47
5. Map showing arrangement of drill holes in another area............ 48
6. Cross section of box type of spud................................ 66
7. Cross section of spud with center web............................ 66
8. Cross section of spud made of four 30-inch I beams bolted together. 66
9. Cross section of new type of distributor.......................... 76
10. Cross sections of four types of conveyor belts in general use...... 79
11. Sections of washing screen of Conrey No. 4 dredge, showing placing of 100-horsepower driving motor........................... 103
12. Plan and elevation of electrically driven gold dredge............ 104
13. Plan and elevation of 17 cubic foot Yuba dredge, electrically operated, digging 80 feet below water level.................... 105
14. Plan and elevation of flume-type Alaskan dredge with revolving screen and distillate engines................................ 106
15. Method of operating gold dredge to leave a channel............ 115
16. Clay cut-out device for gold dredge............................. 120
17. Specimen chart showing record of depth dug by gold dredge.... 126
18. Plan and elevation of 9-foot dredge, showing swing winch and swing lines; also method of calculating yardage handled...... 127
19. Neill jig and eccentric drive..................................... 138
20. Cross section showing arrangement of jigs, Hardinge mill, and amalgamator, Yosemite dredge................................ 139
21. Plan and elevation of tables on Yosemite dredge, showing position and drive of jigs and mill............................. 140
22. Plan and elevation of undercurrent system used on No. 3 dredge at Breckenridge, Colo.. 145
23. Table plan of gold-saving area on dredge equipped with revolving screen, showing distribution of flow and recovery from segregated areas... 183

GOLD DREDGING IN THE UNITED STATES.

By Charles Janin.

INTRODUCTION.

The recovery of gold from sands and gravels is one of the oldest forms of mining; it antedates history and has been practiced by savage peoples. In North America the search for placer gold has been a powerful agency in the exploration and development of unknown regions and has profoundly influenced the course of our civilization. With the progress of invention, the improvements in machinery, and the availability of large capital for the conduct of great operations, the tendency in placer, as in many other branches of mining, has been toward the working of extensive but comparatively low-grade deposits. The early miner washed the rich spots in stream beds or terraces with his pan, his long tom, and his primitive sluice. Then followed the growth of hydraulic mining and the use of giants supplied with water under pressures of hundreds of pounds to the square inch that could work at a profit gravels whose gold tenor was less than 3 cents to the cubic yard. Finally came the dredge that could handle deposits lying so far below water level as to be beyond attack by other methods.

Although the gold dredge was first successful on a commercial scale in New Zealand, it has reached its present strength and efficiency in this country and its development is a monument to the daring, perseverance and technical skill of the many men, mine owners and engineers who have each contributed something to the final result. The Bureau of Mines, authorized to conduct investigations for the purpose of increasing efficiency in the mineral industries, had its attention called some years ago to the need of a comprehensive report on gold dredging in the United States. The director of the bureau recognized that such a report should summarize the development of the gold dredge, should describe in detail the essential features of present-day dredges, should discuss the facts that determine whether a placer deposit can be profitably dredged, and should point out the approved methods of prospecting placer ground and of operating dredges.

At the request of the director, the author undertook the preparation of such a report and has been intermittently engaged on the task for several years. This bulletin is the outcome of his labors. He hopes that it will prove of interest to designers and builders of dredges, to mining engineers, and to persons who may contemplate investing capital in placer-mining ventures.

ACKNOWLEDGMENTS.

In the preparation of this report the writer has obtained information from many sources. He takes pleasure in saying that nearly every request for information was met with courteous response and hearty cooperation. Although it would be difficult to mention the names of all who have given information at different times, the writer feels especially indebted to the following men: W. P. Hammon, Emery Oliver, Charles Gardiner, George Carr, M. L. Summers, E. G. Thunen, Charles Coil, Louis Hopfield, Frank Griffin, Hennen Jennings, Charles Kammerer, W. H. James, C. G. Leeson, E. J. Hopkins, and Charles Haley.

Of the construction companies, the Bucyrus Co., through its engineer, Mr. Rosenblatt; the Marion Steam Shovel Co., through Mr. Felsmuth; the New York Engineering Co., through Mr. Ludlum and Mr. Lowerie; the Union Construction Co., through Messrs. Johnson and Peake, the Union Iron Works, through Mr. Hearst, and the Yuba Construction Co., have supplied considerable information. Many of the photographs used have been furnished by these companies.

The Edgar Allen American Manganese Co. and the Taylor Wharton Iron & Steel Co. have also been generous in giving information.

The chapter on dredge construction was written almost entirely by George E. Sibbett. This chapter is especially mentioned, as the author of this bulletin considers it by far the best presentation of the subject ever published.

HISTORY.

Little is to be gained by narrating in detail the history of gold dredging, but a report of this kind should contain some mention of the progress and development of the industry. For this reason a brief historical summary is given.

Successful gold dredging in the United States is a matter of not more than 20 years' standing. Efforts were made even before 1890 to dredge the beds of rivers, but it was not until 1895 that success began to crown the efforts of the pioneers in this country. The first dredges to work in the United States were erected in California; and a machine was constructed even as early as 1849 for the purpose of scooping the gravel from the inaccessible beds of the Sacramento River. This machine was sent around the Horn from New York, but soon after its arrival it sank to the bottom of the river, where no doubt it still remains. During succeeding years many attempts at gold dredging were made in California and elsewhere in the United States.

EARLY DREDGES.

It is curious that, in spite of knowledge that might have been obtained from New Zealand, many of the "dredges" first tried were of the suction type which, so far as the writer has been able to discover, has never proved a success in gold dredging.

SPOON DREDGE.

An early, if not the earliest, type of dredge was the so-called "spoon dredge." In 1865 spoon dredges worked by hand had been tried with some success in New Zealand on rich spots on the Clutha River. A photograph of the first dredge of this kind is reproduced in Plate I, *A*. Plate II, *A*, shows a model of a dredge used by natives in the Philippine Islands. In the California dredges steam was used for power and the spoons were made of iron and weighed upward of a thousand pounds, whereas the spoon of the early New Zealand dredge consisted of an iron ring or cutting edge to which was fastened a rawhide bag. These spoons were placed at the end of a long pole and dragged along the river bottom by means of a rope attached to a windlass. When the bags were filled they were lifted to the barge by means of the windlass and their contents dumped into sluices or washed in ordinary rockers by hand.

One California "spoon" dredge in 1893 is reported to have recovered $40 a day for a short period. The New Zealand boats, hand operated, recovered as much as 80 ounces a week of six working days; but those with steam power were reported a failure.

The orange-peel bucket excavators, such as are used on harbor work, were tried in various places and were used in California in 1886;[a] two were also tried at Ruby, Mont. However, excavations of this type were uniformly unsuccessful. The steam shovel has been a partial success in gold dredging. With modifications and an auxilary washing plant, and placed either on a floating pontoon or used as a dry land excavator, with a long or short boom, the steam shovel has been tried in many fields. The first one recorded in gold dredging was used in New Zealand in 1870.

The gold dredge with bucket elevators was designed by a man named Scott and put in operation in New Zealand in 1882.[b] This dredge at first used steam power, but as the boilers were worn out and the engines were too large and heavy for the pontoons, the consumption of fuel proved excessive for the capacity of the dredge. Consequently the engines and boilers were discarded. Side paddle wheels placed on the pontoon and driven by the current of the river were used to elevate the dredging buckets. Sheet-iron buckets attached to the paddle wheels lifted the water for washing the material dug. This dredge (Pl. I, *B*), after a varied career, finally proved a success, and at one time more than 12 dredges of this type were busy on the river. One current-wheel dredge, according to the report of the Minister of the Mines, New Zealand, recovered £3,750 during two months operation.

---

[a] California State Mining Bureau, Eighth ann. rept., State Mineralogist, 1887-88, pp. 100-102.

[b] Cutten, W. H., New Zealand Inst. Min. Eng., Aug. 23, 1898.

The advantages of the current-driven bucket dredge were: Small cost, no fuel, few hands required. The disadvantages were: Dependence on the current for power, and hence necessity of keeping well out in the stream where the efficiency was regulated by the depth of the water; during a considerable part of the year the water may have been too deep or too shallow to permit work.

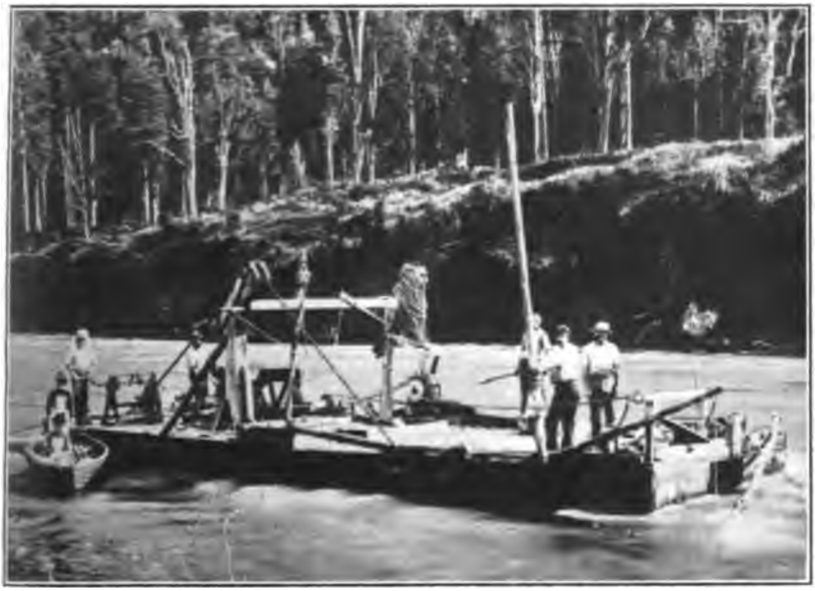
STEAM DREDGES.

The first steam gold dredge was designed by Charles McQueen and was in operation on the Clutha River in 1882. This dredge was equipped with a double line of elevator buckets, one on each side of the hull, which was of iron, 70 feet long, 15 feet beam, and 6 feet deep. The buckets discharged into two revolving screens. The fine dirt went into the sluice boxes, which at first were equipped with revolving copper pans to amalgamate the gold. The pans were soon discarded as useless and the gold was caught on cocoa matting. The two ladders were not wholly a success, as dredging first on one side and then the other occasionally rolled the decks under water; then the strong current of the river at times made the boat almost unmanageable. During 16 years of operation, first with two ladders and afterwards with one, the dredge is credited with having recovered nearly £60,000 ($300,000). The dredge is shown in Plate III.

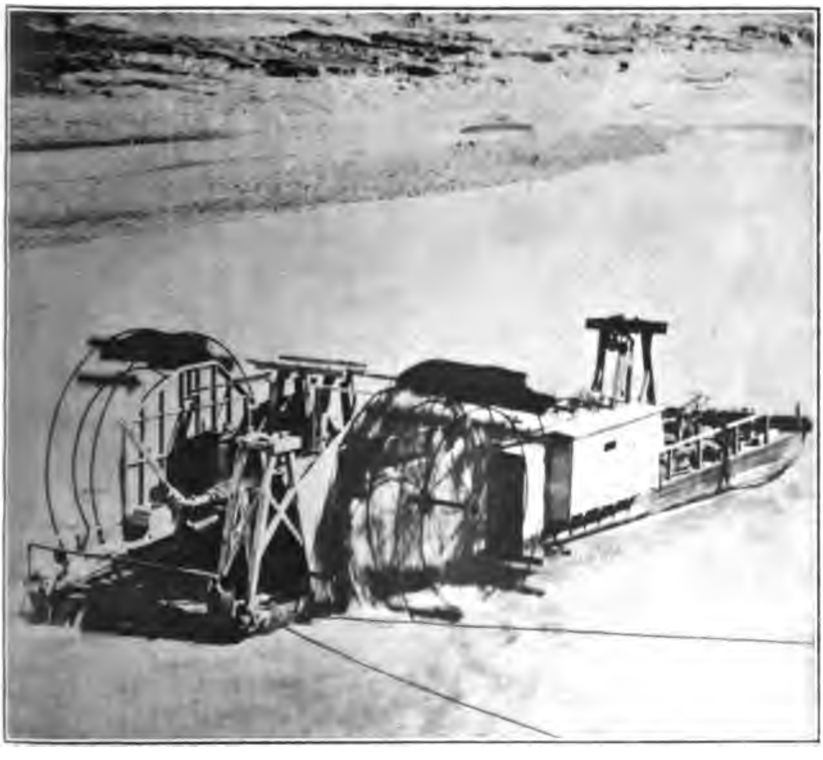
Although dredging for gold had its inception in New Zealand and the number of dredges at work in that country has been in excess previous to 1914 of the total number annually working in the United States, excluding Alaska, progress in dredge construction and operation was far more rapid in the United States than in New Zealand. The improvement in this country no doubt has been due largely to the progressive spirit of operators and the keen competition that early developed among dredge-construction firms. The latter strove to build machines of increasing efficiency for a rapidly growing demand, and the operators desired machines of greater capacity and strength.

DREDGING IN CALIFORNIA AND MONTANA.

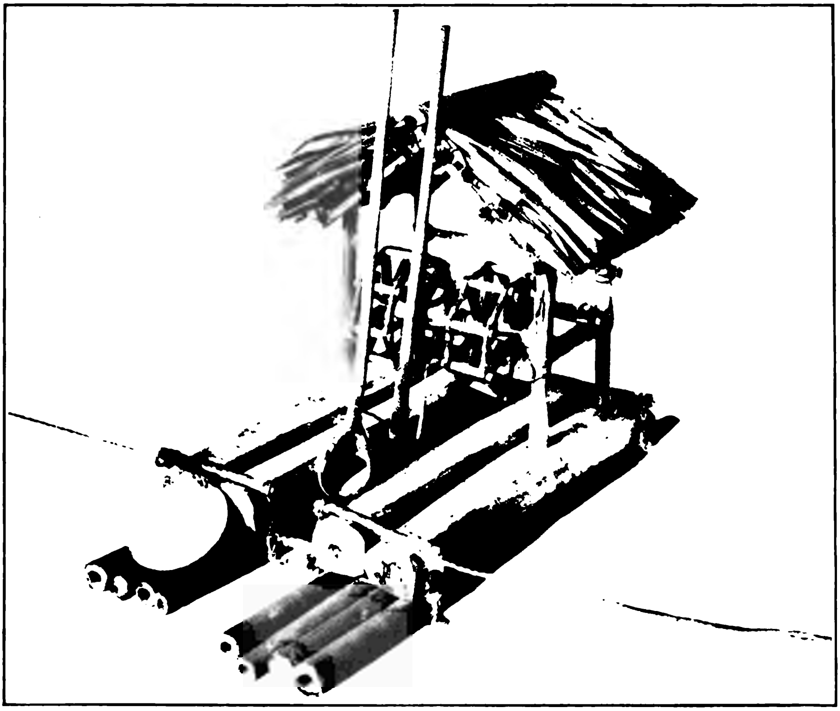
The first successful bucket-lift gold dredge in the United States began work at Bannack, Mont., in 1894, and in the total production of gold won by dredging in this country, Montana is second only to



A. SPOON DREDGE, NEW ZEALAND, 1865.



B. CURRENT-WHEEL DREDGE, NEW ZEALAND, 1882.



A. MODEL OF DREDGE USED BY NATIVES IN PHILIPPINE ISLANDS.

B. FACE OF CUT IN GROUND ALONG MOKELUMNE RIVER, CAL.

California. The output from gold dredges in California has far exceeded the combined output in all other States and, for that matter, in any foreign country. Also, it is true that most of the progress made in dredge construction has been due to dredging operations in California, although Montana has contributed materially to the development of the modern gold dredge. A number of the features that originated or were first tried out in Montana are discussed in a paper by Jennings.[a]

The history of successful gold dredging in California practically begins with the floating of the first bucket dredge that proved a success in that State at Oroville, March 1, 1898. A dredge designed after the type used in New Zealand was built by R. H. Postlethwaite on the Yuba River in 1897, and had the stream been less turbulent that dredge also would have proved a success, as similar boats operated later in other fields. However, because of difficulties not at first realized the dredge sank to the bottom soon after starting. A number of other dredges were subsequently built by the Risdon Iron Works after the design of Postlethwaite and were more or less successfully operated for many years in California. These early Risdon dredges were all modeled after the New Zealand dredges of that period.

Figures showing the number of dredges operating and the value of the gold recovered by dredging in California from 1896 to 1915 follow:

Value of gold won by dredging in California, 1896–1915.[b]

Year.	Number of dredges.	Value of gold.	Increase (+) or decrease (−).
1896.......	1	[c] $2,000	....
1897.......	1	[c] 2,000	....
1898.......	3	18,887	....
1899.......	8	206,302	+ $187,415
1900.......	16	200,929	− 5,373
1901.......	22	471,762	+ 270,833
1902.......	29	867,665	+ 395,903
1903.......	31	1,475,749	+ 608,084
1904.......	42	2,187,038	+ 711,289
1905.......	50	3,276,141	+1,089,103
1906.......	59	5,098,359	+1,822,218
1907.......	57	5,065,437	− 32,922
1908.......	69	6,536,189	+1,470,752
1909.......	63	7,382,950	+ 846,761
1910.......	72	7,550,254	+ 167,304
1911.......	65	7,666,461	+ 116,207
1912.......	65	7,429,955	− 236,506
1913.......	63	8,090,294	+ 660,339
1914.......	60	7,783,394	− 306,900
1915.......	58	7,796,465	+ 13,071
Total...	....	79,104,231	....

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[a] Jennings, Hennen, Gold dredging in Montana, with a chapter on placer mining methods and operating costs, by Charles Janin, Bull. 121, Bureau of Mines, 1916, 63 pp.

[b] Figures for 1896–1914 from Mineral Resources U. S. for 1914, U. S. Geol. Survey, pt. 1, 1915, p. 357; figures for 1915 from Mineral Resources U. S. for 1915, U. S. Geol. Survey, 1916, pt. 1, pp. 211, 213.

[c] Estimated.

A short description[a] of the first Yuba River dredge is inserted here as of interest in comparison with the modern dredges now working on that river and described in other chapters of this bulletin:

> The dredge consists of two long pontoons, each 96 feet long by 9 feet beam. These are connected at the stern by a small pontoon 17 feet long and 5 feet wide, the bow being connected by heavy overhung beam. This practically makes one boat 96 feet long, 23 feet wide, with a well-hole 5 feet wide, running through the center for some 75 feet.
>
> The ladder, 67 feet long, built up as a heavy lattice girder, is hung at the stern end by a bar fixed across a heavy wooden framing. The lower end of the ladder carries a five-sided tumbler and is suspended by blocks and tackle to a cross beam. By means of wire rope and blocks the winch can raise or lower the bottom end as required. The top tumbler is carried by the timber framing some 3 feet above the top end of the ladder. The continuous bucket chain comes up the top side of the ladder on rollers around the top tumbler and back in a catenary curve to the lower tumbler. The top tumbler is driven through a rope transmission and heavy gears by the engine, a vertical compound condensing one, which also drives the pump and indicates about 35 horsepower.

This dredge was designed to dig 45 feet deep, whereas the modern dredges on the Yuba dig 70 to 85 feet below water level, a feat that was considered by come engineers mechanically impossible, even as late as 1905.

Up to 1898 gold dredging in the United States had not been financially successful, though dredges embodying the ideas of various manufacturers were tried in California, British Columbia, Idaho, Montana, and Colorado. From crude beginnings, however, the machines were rapidly improved until in some localities, as at Bannack, Mont., the efforts of the operators began to meet with greater financial encouragement. Most of these early dredges were light, relatively inexpensive affairs, costing $20,000 to $30,000 and were equipped with buckets having a capacity of 2½ to 5 cubic feet. The first Yuba River dredge is stated to have cost $25,000; a boat now being built in that district will cost about $450,000.

BUCKET-ELEVATOR DREDGE.

The first successful bucket-elevator gold dredge to operate in the United States was built in Montana. It was known as the *Fielding L. Graves*, and was constructed on Grasshopper Creek near Bannack. G. T. Reiling, now interested in gold dredging at Breckenridge, Colo., was connected with this early enterprise in the spring of 1895. The dredge was of the double-lift type. The material, excavated and elevated by the buckets as in the present single-lift type, was discharged into a revolving screen with large perforations, so that all gravel over 4 inches in size was discharged overboard at the side of

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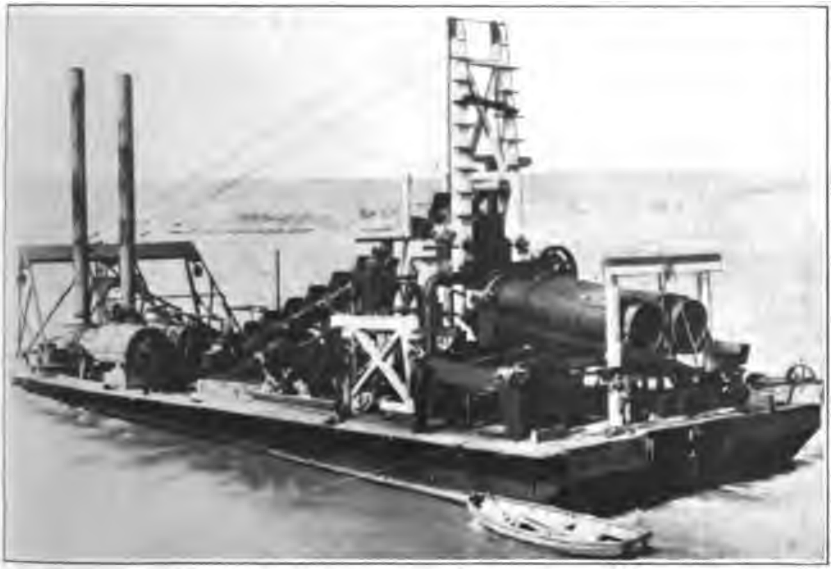
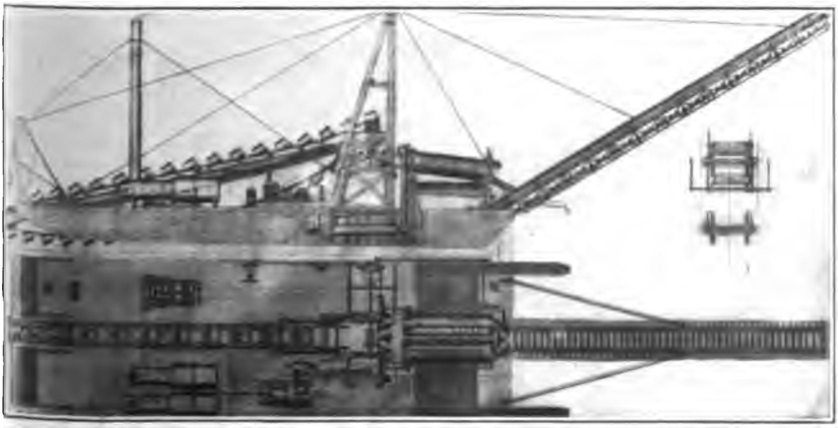
[a] Postlethwaite, R. H., River dredging for gold, Min. and Sci. Press, vol. 75, Sept. 4, 1897, p. 217.



A. NEW ZEALAND DREDGE USING TWO SCREENS.



B. EARNSCLEUGH NO. 2 DREDGE, CLUTHA RIVER, NEW ZEALAND.

the dredge near the center while the fines passed to a sump. A second line of buckets lifted the material from this sump to a sluice fitted with riffles. The gold was caught on these riffles, which at times were charged with quicksilver. The tailings were discharged about 100 feet astern of the boat.

Several points of interest about this dredge are not generally known. The second lift was by elevator buckets, electricity was the motive power, and the boat was equipped with two spuds. The washing sluice or flume was at first suspended from cables anchored to towers on either side of the gulch; but as this method proved impracticable, it was abandoned and the sluice placed on an auxiliary scow. Plate IV, *A*, shows this dredge as it now appears.

The *A. E. Graeter*, the second dredge to work in the Bannack district, was built in 1897. This dredge, also illustrated here (Pl. IV, B and *C*), although it had buckets of the same size as those on the *Graves*, differed from that boat in having steam power and using a centrifugal pump to elevate the fines from the sump to the sluices. A point of interest in the *Graeter* is that the lower end of the tailing sluice was arranged so that it could be swung laterally by means of cables, in order to deposit the fines over a wider area. To preserve the balance of the boat, which might be affected by this change of dumping point, two tanks each 42 feet long, 12 feet wide, and 42 inches deep, were contained in either side of the hull. When the sluice was to be swayed to one side water was pumped into the tank on the opposite side.

Plate V, *A*, shows one of the early Colorado dredges that worked in the Breckenridge district in 1898, the forerunners of the dredges now successfully operating there. It is interesting to study the different forms of construction used in these early boats and to note the gradual development of mechanical and structural improvements.

PROGRESS IN DESIGN AND EQUIPMENT.

The work of the dredges built prior to 1900, or even of some built as late as 1910, was by no means completely satisfactory from the start, and large sums of money were expended in changes and experiments before some of the ventures proved a financial success. Troubles with the later dredges were caused partly by the operators becoming more ambitious, as early difficulties were overcome, and attempting to work deposits that at first were considered entirely outside the realm of possible dredging. One can picture the astonishment of the pioneers in this industry when expressing anxiety over the work of their early boats, capable of handling only 50 cubic yards an hour, if they had been asked to design and finance a dredge that would dig 80 feet below water level and would handle

500 to 600 cubic yards an hour or 400,000 cubic yards a month for months at a time. Yet this is what the mammoth gold dredges in California, Idaho, and Montana are doing now. Too much credit can not be given the pioneers for their persistent and progressive efforts, and it seems eminently fitting that W. P. Hammon, who has been the leading operator since gold dredging began in California, is still actively interested in the operations of the largest gold-dredging company in America. Among other names prominent in California dredging are those of R. G. Hanford, George Carr, Frank Griffin, Charles Gardiner, and Louis Hopfield; in Montana the names of N. S. Schafer, Gordon McKay, and Charles Kammerer are well known. In addition to these men many others have contributed greatly to the development and success of the modern gold dredge.

GOLD PRODUCED BY DREDGING.

The total production from gold by dredging in the United States, excluding Alaska, from the beginning of operations up to the end of 1914, has been approximately $85,000,000. Of this sum California has contributed more than $71,000,000. Montana comes next, with $6,500,000, and is followed by Idaho and Colorado, with about $3,000,000 each. During 1914 there were nearly 80 dredges busy in the United States outside of Alaska. The following table, showing the production of gold by the various States for different years, has been compiled by the United States Geological Survey:[a]

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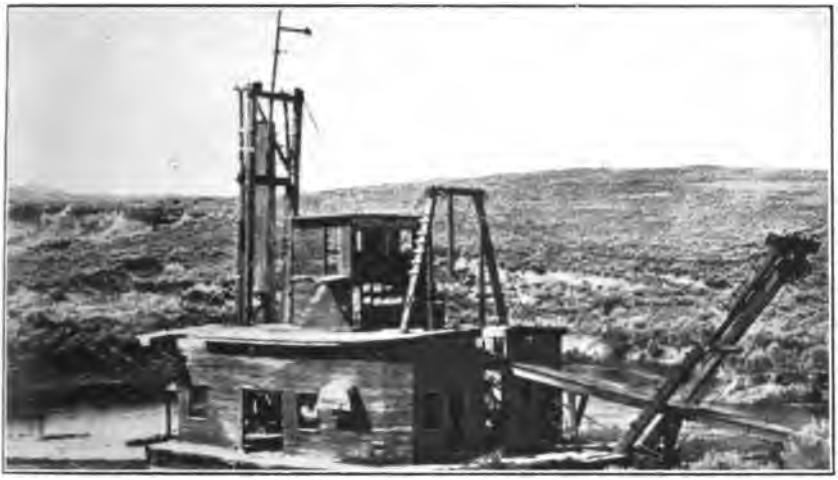
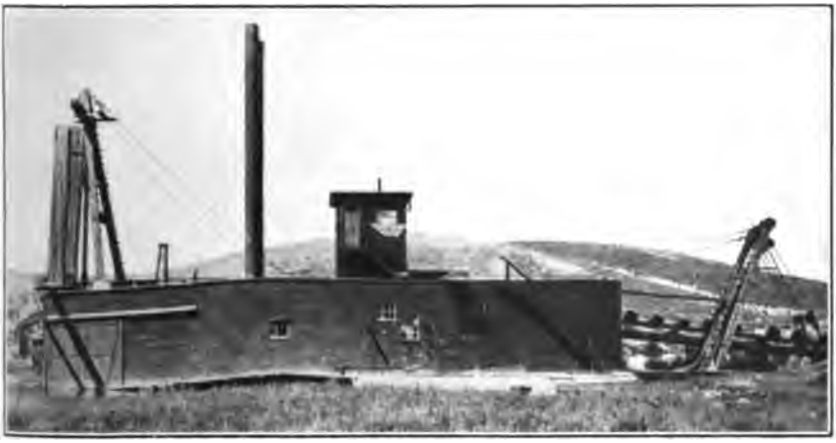
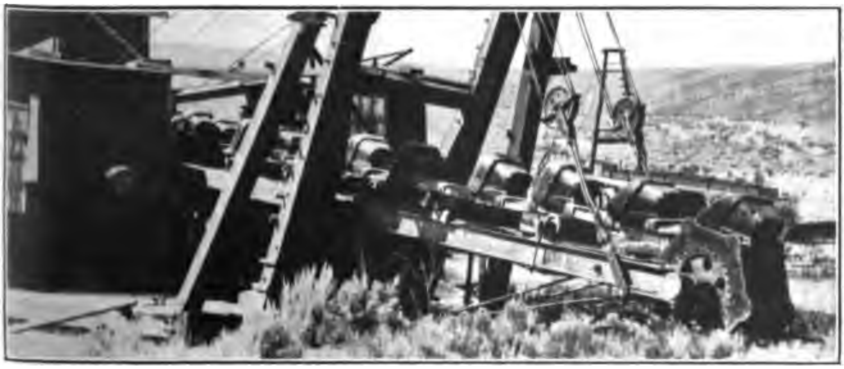
[a] McCaskey, H. D., Gold and silver (general report): Mineral Resources U. S. for 1914, U. S. Geol. Survey, 1915, p. 855.



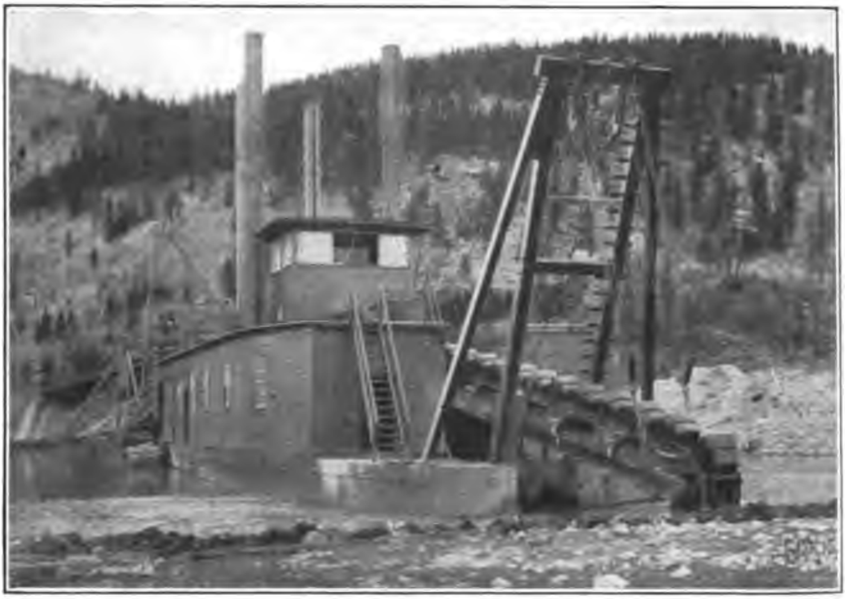
A. THE FIELDING L. GRAVES, FIRST SUCCESSFUL BUCKET-LIFT DREDGE IN THE UNITED STATES, 1894.



B. SIDE VIEW OF A. F. GRAETER DREDGE USED AT BANNACK, MONT.; STARTED 1895.



C. BUCKET LINE ON A. F. GRAETER DREDGE.



A. DREDGE BUILT AT BRECKENRIDGE, COLO., IN 1898.

B. OLD METHOD OF WORKING PLACER GROUND BELOW OROVILLE, CAL. FROM AN OIL PAINTING.

Production of gold in the United States by dredges and number of dredges producing, by States, 1896–1914.

State or Territory.	1914		1913		1912		1911		1910		1909		1908		1907		1906		1905	
	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.
Alaska	$2,350,000	42	$2,200,000	36	$2,200,000	38	$1,500,000	27	$800,000	18	$424,998	14	$170,901	4	$250,000	4	$120,000	3	$40,000	3
California	7,783,394	60	8,090,294	63	7,429,955	65	7,666,461	65	7,550,254	72	7,382,950	63	6,536,189	69	5,065,437	57	5,098,359	59	3,276,141	50
Colorado	602,655	5	372,288	4	384,748	3	272,173	4	344,211	6	404,636	4	141,773	4	35,235	3	48,343	3	33,342	3
Idaho	568,989	4	561,876	6	481,077	8	258,791	7	91,247	6	101,704	8	77,189	5	74,438	6	38,340	3	34,336	3
Montana	835,615	5	685,210	5	710,387	6	597,778	8	473,318	7	426,649	5	402,667	4	197,141	4	397,030	4	275,542	5
Oregon							14,575	3	34,010	6	42,667	2			23,191	2				
Other States	[a] 372,130	4	[b] 317,268	2	[c] 12,744	4	[d] 16,591	5					[e] 24,852	2	[f] 10,260	3	[g] 19,322	4	[h] 28,015	4
Total	12,512,783	120	12,226,936	116	11,218,911	124	10,326,369	119	9,293,040	115	8,783,599	96	7,353,571	88	5,655,702	79	5,721,394	76	3,687,376	68

State or Territory.	1904		1903		1902		1901		1900		1899		1898		1897		1896		Total value.
	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	
Alaska	$25,000	3	$20,000	2															$10,100,894
California	2,187.038	42	1,475,749	31	$867,665	29	$471,762	22	$200,929	16	$206,302	8	$18,887	3	$5,000	1	$2,000	1	71,314,766
Colorado	65,594	3	15,000	1	10,000	1	6,000	1											2,735,998
Idaho	99,110	7	86,113	6	101,257	10	116,117	6	129,443	6	62,436	5	13,920	1	11,436	1			2,907,819
Montana	245,700	2	229,332	4	318,914	7	146,134	5	185,665	5	165,440	5	154,893	4	192,120	4	42,000	1	6,595,535
Oregon																			[k] 114,443
Other States	[i] 101,275	4	[j] 89,870	1	[j] 71,686	1													[l] 1,064,013
Total	2,723,717	61	1,916,064	45	1,360,522	48	740,013	34	520,037	27	434,178	18	187,700	8	118,556	6	44,000	2	94,833,468

[a] Oregon, Nevada, Alabama, North Carolina.
[b] Oregon, Nevada.
[c] South Dakota, North Carolina, Georgia, Wyoming.
[d] Nevada (2), South Dakota, Georgia, North Carolina.
[e] Georgia, Oregon.
[f] Georgia (2), New Mexico.
[g] Georgia, Oregon.
[h] Georgia, North Carolina, Oregon, New Mexico.
[i] Oregon (3) New Mexico.
[j] New Mexico.
[k] Part of output only; see other States.
[l] Largely from Oregon.

The significance of these figures is emphasized by comparing them with returns from foreign countries as shown by the report on dredge mining in Victoria for 1914.[a] In Victoria the total production of gold by dredges from the beginning of operations to the end of 1914 was 1,094,562 ounces or approximately $22,000,000. The largest number of dredges busy in a single year was 133 in 1897, when 20,000,000 cubic yards of material was handled with a reported average recovery of 2.4 grains or approximately 9.6 cents a yard. Compare these figures of yardage with that of one company, the Natomas Consolidated, of California, which in 1914 operated 13 dredges that handled in all 25,800,000 cubic yards with an average gold recovery of 10.25 cents a yard. In New South Wales according to the reports of the Department of Mines, during the years 1900 to 1914, inclusive, the total yield for gold dredging bucket and pump dredges was £1,682,540, or about $8,400,000. During 1914 there were 18 bucket dredges and 8 pumping plants employed, and the 18 bucket dredges treated 4,315,962 cubic yards of material during the year. These figures of dredge capacity might be compared with those of the Boston & Idaho Co. at Idaho City, Idaho, where an 18-foot bucket dredge handled a total of 4,670,124 cubic yards in 1914.

According to the minister of mines of New Zealand,[b] 64 gold dredges in 1914 produced gold to the value of £191,112. During the past decade the number of gold dredges decreased from 167 in 1906 to 64 in 1914, and the value of the output from £500,000 to that given for 1914. From 1906 to 1914, inclusive, the production of gold by dredging in New Zealand was £2,883,570, the average annual recovery being less than £5,000 per dredge.

Such figures show how the first efforts in other countries have become the basis of an industry that has been developed in the United States beyond all dreams of the pioneers.

As the modern gold dredge is the result of gradual development, the attaining of the present mechanical efficiency and economic success has required many years. After the spoon dredge operated by hand came the steam shovel, then came the endless bucket-chain, first with buckets and connecting links and then, years later, with close-connected buckets. At first the power was steam, then current wheels, then steam again. Some dredges obtained power from water wheels on the boat, the water under pressure being brought through pipes with flexible or swivel joints. The application of electric power generated by central steam plants or by hydroelectric plants was a great advance in dredge operation. Gas and oil engines of various types and makes for dredge purposes were tried and successfully developed from experience in Alaska. In some foreign fields producer gas has been used to a small extent but has not proved a great success. One of the most recent innovations in power installation is on a dredge for the Philippines, which carries a steam turbo-generator for electric power. From the spoon dredge in 1865 to the first elevator-bucket dredge in 1882 was a long step, but the advance from the early bucket dredges costing $20,000 to the present mammoth dredges costing $400,000 is fully as great.

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[a] Victoria, Australia, Department of Mines, annual report for 1914, p. 7.

[b] New Zealand, Mines statement for 1914, 1915, p. 23.

DREDGING IN COLD REGIONS.

The pioneers of the industry in the far north, in addition to the ordinary problems encountered in gold dredging, have been obliged to overcome the seemingly insurmountable difficulty of dredging frozen ground. The successful use of steam for thawing such ground has been perhaps the most important development of dredging in the north and has greatly extended the dredgeable areas in Alaska and Yukon Territory.

The first dredge to work in Yukon Territory was built on Lewis River in 1899. In Alaska the first dredge on the Seward Peninsula was built on Snake River near Nome in the same year. Since then a number of dredges with 16-foot buckets have been built on the Yukon and dredges with 10-foot buckets on Seward Peninsula, the total number of dredges operating in both fields being about 54 in 1914.

The production of gold by dredges in Alaska has exceeded $10,000,000. In Yukon Territory to the end of 1914 the production approximated $20,000,000, one American company, the Yukon Gold, contributing more than $16,000,000.

EVOLUTION OF GOLD DREDGES.

The modern dredge can not be claimed as the child of any one brain. Rather it is a composite product, representing the progressive thought and earnest cooperation of many individual operators, engineers, and manufacturers. Even some of the many utterly unsuitable and freak machines tried in the early days may be said to have suggested an improvement made in the practical dredges of that time, nor can one say that no further improvements are coming and that perfection has been reached. Some comment on the direction in which improvement is possible will be found in the chapter on dredge construction.

GOLD-DREDGING DISTRICTS IN THE UNITED STATES.

The principal gold-dredging districts of the United States are in California. Montana comes second in importance and production; Colorado, with two proven districts, is third; and Idaho and Oregon practically complete the list of the States. Some dredging has been done in Nevada, New Mexico, and Georgia, but the operations have been relatively of small economic importance.

In Alaska the principal dredging operations have been in the Nome and Council districts on the Seward Peninsula. Others are scattered in the Forty-mile, Iditarod, Ruby, Circle, and Fairbanks districts In the Philippines, the best known and most important district is the Paracale. The new dredge at Mambulao may open another field of importance.

CALIFORNIA.

OROVILLE AND HAMMONTON DISTRICTS.

In California the oldest and best known district is Oroville, but it is being rapidly worked out and the number of dredges is decreasing from year to year. In 1910 there were 25 dredges in the Oroville district, as compared with 8 at present.

As regards production, the most important district is at Hammonton on Yuba River. In 1915 15 dredges were busy there. The value of the gold recovered in 1910 was $3,172,476; in 1915 the value was $2,675,090. The decline in production is due to the working of poorer ground, for the total yardage handled by the dredges has increased. This fact may be made plainer by comparing the figures for one company, the Yuba Consolidated. In 1911 the 12 dredges of this company handled approximately 14,000,000 cubic yards, whereas for the year ended February 28, 1916, the same number of dredges handled about 17,750,000 cubic yards; yet the gross production for 1915 was about $800,000 less than that for 1911. New dredges for Yuba County, which will start in 1917, include four 16-foot bucket, all-steel boats for the two companies now operating and a 9-foot boat for a new part of the district. Yuba County will lead in 1917 in gold production and number of dredges operating.

OTHER DISTRICTS.

In Calaveras County, at Comanche, and Jenny Lind there were 5 dredges busy in 1915. In Placer County there were 3 dredges operating in 1915. The principal district is on the American River, where the extent of the available ground is limited. In 1910 dredging was confined to the Bear River, where it proved unprofitable.

In Sacramento County the Folsom district is still a heavy producer. In 1915 there were 11 dredges busy there as compared with 8 at the beginning of 1910. This district produced from dredging in 1915 $2,129,781 in gold. The reconstruction of several old boats to embody some important changes was undertaken in 1916.

In Stanislaus County one dredge has been in operation since 1908 and in Merced County one since 1907. In Siskiyou County there are two dredges busy, which in 1915 produced about $211,000 in gold. In Trinity County there are two dredges, one at Trinity Center and another at Lewiston. In 1916 considerable prospecting was done near Carrville and Junction City, and several new dredges are under construction for work in this county.

In Shasta County successful gold dredging is comparatively recent, although a number of attempts in different areas were made some years ago. Dredges are working on Cottonwood Creek and on the Salzer ranch, near Redding. The Igo district has been recently prospected.

PRESENT STATUS OF DREDGING.

As a whole, the output of gold from dredges in California has reached its maximum. The Hammonton and Folsom districts seem to have passed their highest mark, the Oroville district is nearly exhausted, and the possible area of more recent development in the northern counties is very limited. To offset the exhaustion of virgin territory, several attempts have been made to handle the tailings from early dredges, and it is possible that some plan will be evolved whereby the reworking of dredged ground will become profitable. The recent construction of a number of large dredges has greatly increased the yardage handled in California, though the gold content of the gravel is far less than of that worked in past years. With the operation of dredges now being built and the addition of others, the gold won by dredging in California will be of considerable importance for years to come.

MONTANA.

The four dredges of the Conrey Mining Co. at Alder Gulch, in Madison County, comprise all those in the State. These dredges have been working successfully a number of years, and, with the exception of the California districts named above, the district is the largest producer in the United States.

This district is being rapidly worked out, and one dredge has been shut down since the preceding statement was written.

COLORADO.

In the Breckenridge district, in Summit County, four dredges are at work. The Derry Ranch dredge, near Leadville began work in 1915. In addition to these, the possible operation of dredges in other districts is indicated by considerable prospecting done at different times.

IDAHO.

Dredging still continues in the Idaho City and Kirtley Creek districts, as well as minor operations in less important districts; and the production has increased steadily until it is now about six times what it was in 1910. At present four dredges are at work in this State. These fields are rapidly nearing exhaustion, but some new areas are now being equipped with dredges. One of the dredges of the Yukon Gold Co., having worked out the ground for which it was built, has been moved from the Yukon territory to Idaho.

OREGON.

As dredging on the Powder River had proved successful, another dredge was installed during 1916, making two for this district. The Powder River is the only district where active dredging is in progress in the State, though a dredge has recently started work in the John Day Valley.

ALASKA.

In 1915 there were 42 dredges reported operating on the Seward Peninsula. Most of these have small capacity and shallow digging depth. Two of the dredges are recovering tin ore. In 1915, a dredge was built in the Ruby district, on the lower Yukon, and one near Circle City. The small dredge on Fairbanks Creek, brought from the upper Yukon to the Tenana, is still in operation.

Two dredges are at work in the Iditarod district; and others are in prospect. The total production of gold by dredges in 1915 was about $2,330,000.

PHILIPPINE ISLANDS.

In 1915 six dredges were busy in the Philippines. Four more had been ordered or were being constructed. Most of the active dredges, all but one in fact, are working in the Paracale district, Luzon. The other one, at Mambulao, is a new dredge recently constructed, but no reports on its operations are as yet available. The annual production of gold from dredging in the islands is reported to be upwards of $1,200,000.

TOPOGRAPHY AND GEOLOGY OF DREDGING AREAS.

The largest dredging fields in California, and, so far as production is concerned, in the world lie along three of the largest rivers draining the principal mineral region of the State. The mineral area drained by the different branches of these rivers is about 350 miles long and 30 to 70 miles wide, and includes many of the ancient river channels that were mined for gold with considerable profit in early days. The present river systems have somewhat different courses and frequently cut across these ancient channels (Pl. II, *B*). River bars and flood plains constitute for the most part the placer deposits of the present streams. The placer gold is from the disintegration of the old auriferous slates and other rocks and the lodes they contained and also from ancient channels eroded by the present river.[a]

CALIFORNIA DREDGING DISTRICTS.

The predominating conditions in California are probably typical to a large extent of those in other fields.

Undoubtedly the placers derive their gold from eroded vein and other deposits in the bedrock. One should remember that a good placer should contain fairly coarse gold, because fine gold does not concentrate as readily and is borne away with the sands. Quartz veins that carry mainly coarse gold are apt to be "spotty" and uncertain. In the really valuable quartz lodes the gold mainly is fine, either free or in pyrites, and the erosion of such lodes does not result in rich placers. For this reason there is not likely to be any close association between the best quartz mining districts and the most productive placers. The American and Feather Rivers in Placer and Plumas Counties, Cal., have furnished rich placers but are notably poor in quartz mines. The Caribou district of British Columbia and the Klondike have similar characteristics. The Mother Lode region of California has yielded very little placer gold. The Columbia placer district of Tuolumne County, though limited, was one of the richest in California. It adjoins a district that has pockety quartz mines that produced considerable coarse gold—notably the Bonanza Mine, which was profitable at times—but has shown no lodes that would maintain profitable mining on an extensive scale.

All the gold of the valley placers is fine as compared with the coarse gold found along the higher courses of the present rivers. Although the finest particles have undoubtedly been carried some distance, the coarse gold has probably not traveled far from where it first lodged.

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[a] Lindgren, Waldemar, Tertiary gravels of the Sierra Nevada of California, U. S. Geol. Survey Prof. Paper 73, 1911, 226 pp., 28 pls.

Most of the gold is in fine flat scales that are comparatively easy to recover; but some is as an almost impalpable powder and is carried off in suspension and some is associated with sulphides, or is coated with iron or other oxides, and is lost through failure to amalgamate. Occasionally fair-sized nuggets are found in the dredge sluices and at Breckenridge, Colo., 2½-inch perforations in the lower sections of the revolving screen were necessary to prevent a loss of gold nuggets. At the Alta Bert property in Trinity County, Cal., a large part of the gold is relatively coarse.

CHIEF DISTRICTS.

The principal dredging districts in California are at Oroville on Feather River; at Daguerre Point, about 12 miles above Marysville, on the Yuba; and at Folsom, on the American River. A smaller district has been worked for some time near Chico in Butte County, and some years ago dredging was tried about Wheatland on the Bear River, but the gravels proved to be poor and operations were discontinued.

Smaller quantities of gold are dredged in Siskiyou, Trinity and Shasta Counties. Dredges are also busy in Calaveras County, Merced County and in Stanislaus County, and some dredging is being done on the American River above Auburn.

The depth of the gravels ranges from 15 to 80 feet. In part the gravels are coarse, cobbles up to a foot in diameter being by no means uncommon. In Trinity County, Cal., many of the bowlders met weigh more than a ton. The gravels contain much sand and clay in places and are ill sorted. The Oroville gravels almost universally rest on a false bedrock of volcanic tuff or hardpan; some of them lie below the present river level, and others rest on benches up to 100 feet above the channels of to-day. The tuff forms a sheet 12 to 60 feet thick. At Oroville, and along Yuba River, there is auriferous gravel below it, but under present conditions this lower gravel can not be worked.

While the greater proportion of the gold is in contact with or directly above bedrock, gold is by no means everywhere confined to the lower layers. Gold is often distributed through the gravels or contained in one or more upper beds. Where occasional seams of clay or other material from an impervious layer in otherwise porous gravel, the downward movement of heavy minerals is arrested, and above these layers "pay streaks" are formed, a cross section of a deposit sometimes showing several of them. Where the material is clayey, the gold is likely to be distributed throughout it. As the distance from the mouth of a canyon increases the gold becomes finer and the gravels poorer.[a]

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[a] For a discussion of the geology of the California placers, see Lindgren, Waldemar, The Tertiary gravels of the Sierra Nevada of California: U. S. Geol. Survey Prof. Paper 73, 1911, 220 pp.

OROVILLE FIELD.

The **Oroville** dredging ground (Pl. V, *B*), now practically exhausted, comprised about 7,000 acres of the plain of Feather River, beginning at the town and continuing to about 5 miles below, where the ground gradually became poorer. The width varied from less than a mile to about 2 miles. The average depth of gravel is about 30 feet, increasing to 40 farther down the river. The gravel rests on a false bedrock consisting of a compact tuff containing no large lava pebbles or fragments.

The Feather River, along which the placers lie, flows through a broad valley whose surface is covered with gold-bearing gravels and sands of varying richness and depth. The gold originally came from the gold-bearing quartz veins and old gold-bearing gravels in the mountainous region of the river's upper course, and has been redeposited with the gravels and sands in the neighborhood of Oroville.

As a whole the placer ground is chiefly heavy gravel, with a top layer of sand or sandy loam. This loam is underlain by a stratum of mixed gravel and sand, and in places by well-defined strata of sand, below which are other strata of coarse or fine gravel and sand, the size of the rock fragments varying from that of a pea to occasional bowlders 18 inches in diameter. This mixture of gravel and sand is 20 to 30 feet, and in places 40 feet thick. Beneath it lies the false bedrock, a tuff or volcanic ash, fairly smooth and regular in contour, and soft enough to dig with the dredge buckets. In places the placers contain cemented gravel, and in places considerable clay makes washing difficult. Most of the gold is in coarse gravel; here and there some is contained in sand. The gold is fine, and some of it may be called flour gold. The gold content of the gravel worked ranges from 8 to 35 cents to the cubic yard.

The fineness of grain of the gold in the Oroville field is given in the following statement, which represents averages from a number of tests made by screening through different sized screens:

Size of gold grains from the Oroville field.

Mesh of screen.	Per cent.
60 and coarser	58.1
60 to 100	15.3
100 to 120	2.9
120 to 150	2.4
150 and finer	21.1

In the other districts the gold has approximately the same size of grain. Occasionally, as in Trinity and Siskiyou Counties, small nuggets having a value of 50 cents to several dollars are found, but in California such nuggets are confined almost entirely to the gravels of those counties. The gold is generally of high grade, its fineness averaging 915 to 930. Platinum, iridosmine, etc., are associated with the gold, but their quantity is very small. The platinum does not amalgamate and is separated by panning the black sand of the weekly clean-up of each dredge. The production of platinum from all districts amounts to only a few hundred ounces a year.

YUBA FIELD.

The Yuba field (Pl. VI, *A*) 12 miles from Marysville, at Daguerre Point, is situated where the first "greenstone" hills of Yuba County emerge from the Quaternary covering on the south side of the river. The available dredging land on Yuba River has a total area of about 6,000 acres; it extends along the river for a distance of about 7 miles and has a width of 1 to 2 miles.

The field lies only a little above river level and is covered to a depth of 40 feet or less with the old tailings the river has brought from the hydraulic mines in the mountains. Before these tailings were deposited the low bars rose to a height of 15 to 20 feet above normal water level. Above Daguerre Point, on the north side of the river, the gravels rest on the greenstone bedrock, but elsewhere in the area they lie on a stratum of hardpan or compact clay, which on the north side of the river rises to form low rolling hills. Below the clay in places is volcanic tuff, similar to that of Oroville, and probably derived from the same source, namely, the volcano of the Marysville Buttes. The bedrock brought up by the dredges does not usually show the presence of volcanic material.

FOLSOM FIELD.

The Folsom dredging field (Pl. VI, *B*) at the mouth of the canyon of American River is the largest in the State, comprising about 13,000 acres. From Folsom, in Sacramento County, the field extends mainly along the south side of the river for about 7 miles, with a width of 1 to 2 miles, forming a wide belt of dredging ground of sandy gravel 20 to 35 feet deep. In places some clay is encountered, but the conditions at Sacramento and Mississippi and on the Kendall and Nuttall tracts are generally satisfactory for economic dredging. Like the gravel of the bars, this gravel rests on a false bedrock of white volcanic ash, which outcrops in the Orangevale Bluff, north of the river, and at other points on higher ground south of the river. Only at two places along Willow Creek, within the area actually exploited, does the real bedrock (granite and slate) appear under the gravel.

Farther south and southwest of Folsom is a higher belt of terrace gravel on which the early miners did much shallow hydraulic and drift mining. This belt appears to represent an older channel, which started at Mormon Island and Blue Wing Ravine and continued along the upper course of Willow Creek. Here the channel is 400 feet wide

A. GENERAL VIEW OF THE YUBA FIELD, CAL.

B. DREDGING GROUND IN THE FOLSOM FIELD, CAL.

and formerly was worked at several places through shafts 60 to 80 feet deep. At Rebel Hill this gravel spreads out considerably and is 50 to 70 feet deep, the volcanic ash bedrock lying about 50 feet above the river. In this area the formation consists of gravel held tightly together with clay. In places the gravel is cemented and is covered with a fine clayey soil, and for years profitable dredging was thought to be impossible there. The conditions for work are unfavorable, as the digging is hard, and the clay makes washing difficult. There are two principal pay streaks; one is 34 feet below the surface, the second, at 50 feet or more, rests on the false bedrock. At Sulky Flat and Blue Ravine the gravel is similar to that on Rebel Hill and equally hard to dredge.

In California the mining season is continuous and climatic conditions are generally favorable.

COLORADO DREDGING FIELDS.

The principal dredging fields in Colorado are near Breckenridge, Summit County, where the gravels occupy the bottoms of existing valleys. The greatest known thickness, about 90 feet, is near the Gold Pan pit at Breckenridge.[a] Along French Creek the depth to bedrock in the main channel, from Nigger Gulch down, is 45 to 50 feet. Along the Blue, between Braddocks and the mouth of the Swan, the depth in the old channel is 55 to 60 feet, and along the Swan, from Galena Gulch down, the maximum thickness is 40 to 50 feet.

The width of the gravel-filled valley bottoms is 600 to 3,000 feet along the Blue, 500 to 1,200 feet along the Swan, and 700 to 1,500 feet in French Gulch.

In general, the gravels are loose and the readiness with which they crumble when undercut by the buckets facilitates dredging. Those on the lower Swan cave more easily than those on French Creek. The gravels are generally coarse and contain hard, well-rounded bowlders. Along the Blue these bowlders range in diameter from a maximum of about 6 feet near Breckenridge to 4 feet near Valdoro. In French Gulch the large bowlders rarely exceed 3 feet in greatest diameter and are not so well rounded as those on the Blue. On the Swan, below Galena Gulch, the gravel is more uniform and contains fewer large bowlders than that along the Blue or in French Gulch. Drilling and dredging show that the rock bottoms of the channels are smooth as a whole, but at a few places where the streams have cut into quartzite the auriferous surface of the bedrock is rough and hard enough to tax the dredge machinery severely. Shales generally are so soft as to be easily excavated by the buckets; and the porphyries, owing to more or less decomposition, also constitute a tractable bottom material.

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[a] Ransome, F. L., Geology and ore deposits of the Breckenridge district, Colorado, U. S. Geol. Survey Prof. Paper 75, 1911, 187 pp.

By no means all of the gravel is commercially auriferous. The workable strip, or "channel" in dredging parlance, is generally 180 to 400 feet wide. It follows a sinuous course along the valley, has no regular relation to the channel of the present stream, and does not invariably correspond to the deepest part of the bedrock trough. The lateral limits are indefinite and irregular; like the tenor of the deposit they are clearly affected in many places by the proximity of lateral gulches. Thus the gravels of French Gulch are especially rich below Nigger Gulch, and some of the best ground known on the Swan is at the mouth of Galena Gulch.

MONTANA DREDGING FIELDS.

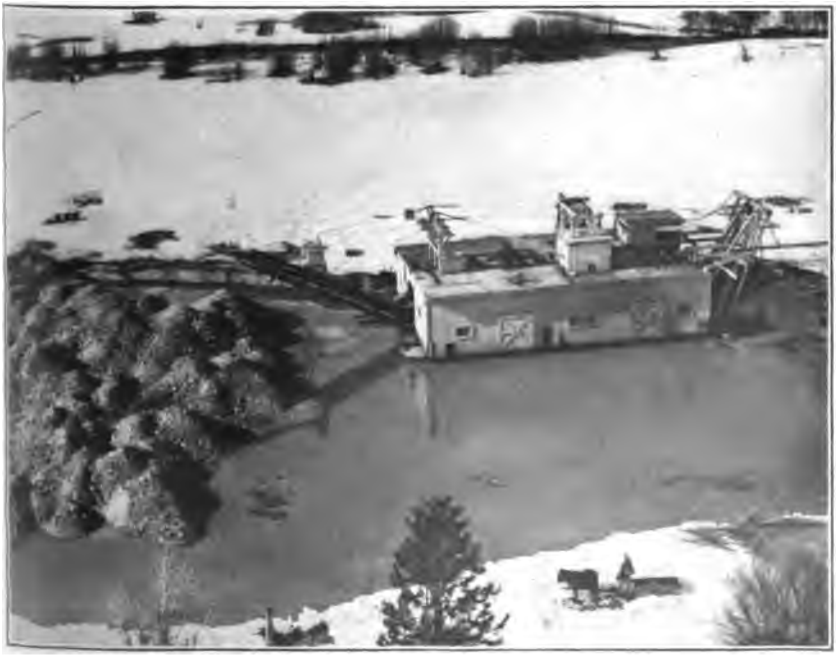
The most important dredging operations in Montana are on Alder Creek, Madison County, at Ruby. (Pl. VII, *A*.) In fact, this is the only field, except for early work at Bannack, where gold dredging has been successful in Montana.

Alder Creek rises on the northern slopes of Old Baldy Mountain, at the south end of the Tobacco Root Mountains, and flows northwest in an irregular curve to the Ruby River at Laurin. The prevailing rocks in the Tobacco Root Range are schists and gneiss, with limestone and quartzite in places. The auriferous gravels at Ruby lie on volcanic ash beds that form a smooth "false bedrock" that shows only gentle undulations and slopes regularly to the west. In the gulch itself occasional spurs of the true or original bedrock are struck in dredging and are in part responsible for some of the ensuing repairs.

Auriferous gravels extend at least 16 miles along Alder Gulch, but the area known to be profitable for dredging is probably only about six or seven miles long, and contains about 1,750 acres of proved dredging ground. Most of this ground lies at the mouth of the gulch and below, where the waters of Alder Creek have formed a low alluvial fan. The depth of the deposit increases gradually toward the west and in places at a distance of 2½ miles from the mouth of Alder Gulch is nearly 60 feet deep. In places are bowlders larger than those usually found in California dredging areas and at these places the gold content is generally higher than in the rest of the gravel.

Although not as cemented nor as hard as the hardest ground in the Oroville district or the deep gravels at Folsom, the gravel has a clay matrix that makes it rather tenacious, especially near the false bedrock. The gold grains vary in coarseness, the average size increasing up the gulch. In the dredging ground 40 to 50 per cent of the gold passes a 60-mesh screen, and 15 to 30 per cent passes a 100-mesh screen. The gold varies in fineness also, that from the dredge farthest up the gulch running about 0.836 fine and that from the lowest dredge, about 2 miles below, being about 0.873 fine.

A. DREDGE PIT NEAR RUBY, MONT.



B. BOSTON AND IDAHO DREDGE IN BOISE BASIN, IDAHO.

IDAHO AND OREGON PLACER DEPOSITS.

In Idaho the placer deposits are largely confined to the large area of deeply eroded granite in the central part of the State. Most of the placer gold has been derived from the gold-bearing quartz veins with which the districts are invariably ribbed.[a] The most important placer district in Idaho is Boise Basin (Pl. VII, *B*), where the Boston & Idaho Dredging Co. has been at work a number of years.

In this basin, the ground is light and easily handled; the gravel is about 30 feet deep; and at times a dredging face 1,300 feet wide has been carried. Before dredging started there had been extensive hydraulic mining in the vicinity and much of the material handled by the dredges is tailings from the old workings. The dredges usually work through the year, but on account of severe weather they shut down in January and February, 1915. There is a great deal of sand in the ground and it is often necessary to run the top material dry through the screens.

Other dredging operations in Idaho have been carried on at Kirtley Creek and Bohannan Bar near Salmon City, Lemhi County, and minor operations at Moose Creek, Lemhi County; Pierce City, Clearwater County; and Elk City.

In Oregon the only profitable dredging enterprise as yet developed is that of the Powder River Dredging Co., near Sumpter. The Powder River drains the richest mineral district in eastern Oregon. The gravel may be characterized as a medium-size dredging gravel, being somewhat larger than the average California gravel and containing some large bowlders. It is 12 to 25 feet deep, the average depth being about 16 feet and 300 to 1,500 feet wide. A few ledges of hard country rock are occasionally encountered. The gold content is mostly in the lower beds and on or near the soft decomposed bed rock, which is known to the dredgemen as "clay webfoot."

PLACER AREAS IN PHILIPPINE ISLANDS.

In the Philippine Islands the placer areas[b] are small as compared with those of other famous fields, but are large enough to offer very profitable working. Except for a few feet at the surface, all the ground dredged in the Philippines has been below sea level and the tide flows nearly to the head of the rivers on which the dredges are.

The gold in general is much more angular than that in placers in other countries. Usually the grains are fine and occur as small well-shaped crystals. An unusual feature along Paracale and Malaguit Rivers is the gold-bearing quartz picked up by the dredge in pieces that range in size from a pea to bowlders weighing 100 kilograms.

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[a] Bell, R. N., State Inspector of Mines, Boise, Idaho.

[b] Kane, William, Gold dredging in the Philippines: Mineral Resources of the Philippine Islands, 1913, p. 30.

The gold in the Paracale district is usually found in the layer of gravel next to the bedrock and filling the crevices and cracks of the bedrock. The gold-bearing gravel is 0.3 to 3 meters (1 to 10 feet) thick. Little or no gold is found in the sands and clays that overlie the gravel.

At Gumaos little or no black sand or pyrite is found. At Paracale and Malaguit, however, large amounts of magnetite, ilmenite, pyrite, and some metallic copper, with other accompanying minerals, are found in places. In Nueva Ecija the gold is in even finer grains than at Paracale, and exceedingly large amounts of black sand make the saving of gold difficult.

On Paracale and Malaguit the mangrove roots and soil are about 2.4 meters (8 feet) in depth. Below is a layer 3 to 6 meters (10 to 20 feet) of black sticky clay or "pug," then a thin layer of sand, and then the pay gravel, which is 0.3 to 1.5 meters (1 to 5 feet) thick. The bedrock, a decomposed granite or gneiss, is well adapted for dredging.

At Gumaos there is about 1 meter (3 feet) of soil, then 3 meters (10 feet) of yellow clay and 6 to 9 meters (20 to 30 feet) of black clay or "pug," below which is the layer of pay gravel which is 0.3 to 4.5 meters (1 to 15 feet) thick. There is very little quartz in this gravel, and none of the quartz is gold bearing like that in Paracale. The recovery of gold from ground so far worked varies from 30 to 90 centavos per cubic yard. Some patches yielded as high as 2 pesos per yard. The greatest depth dredged is 15 meters (51 feet), but the average depth is about 11 meters (35 feet).

Philippine placer ground, so far as shown by dredging, differs from that in other countries in the extremely large proportion of fine material and clay. In California only from 30 to 40 per cent of the material dredged passes through the screens, and this is generally clean sand with little clay. In the Philippines a tropical climate has caused much weathering and decomposition with resulting production of clay and soil, so that 70 to 80 per cent of the material dredged passes through the screens. Also, the material, from its being largely clay, is sticky and hard to disintegrate, resulting in loss of gold. Even the gravel bed in which the gold is found has undergone considerable decomposition to clay.

YUKON DREDGING FIELDS.

Dredging in the Yukon field, which is typical of other fields in the far north, is now being conducted on two classes of deposits, river valley and creek beds. Where thawed the gravel in the valley of the Klondike River is well adapted for dredging. The gravel is free, rounded, and clean, and averages 30 to 40 feet in depth. The bedrock is sufficiently decomposed to be dug with ease. The top burden in the thawed areas of gravel has been largely removed by stream action and presents little or no difficulty to dredging.[a]

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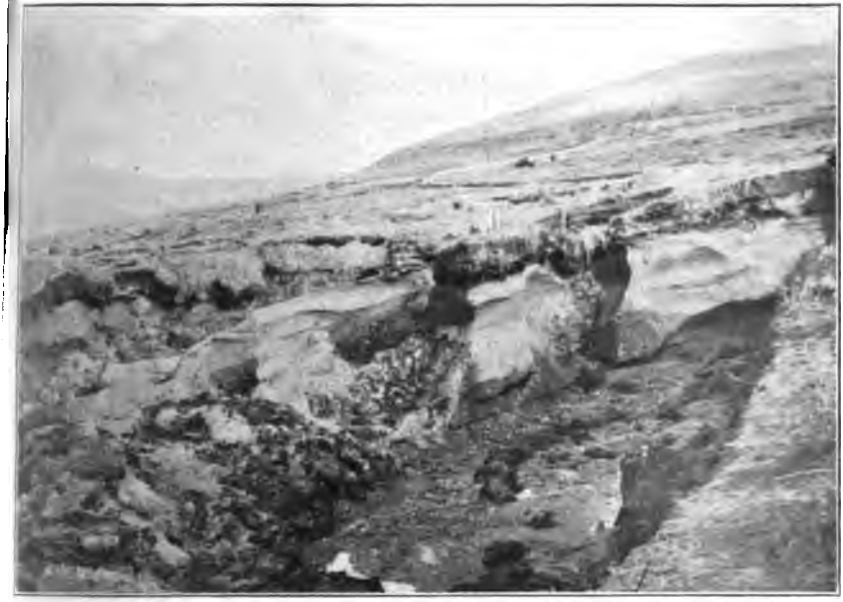
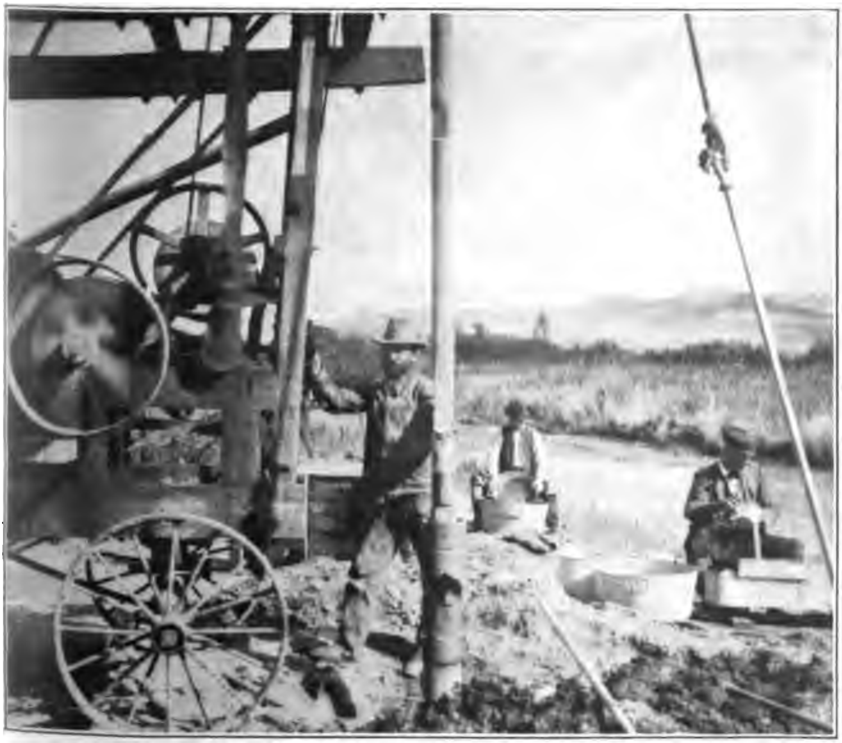
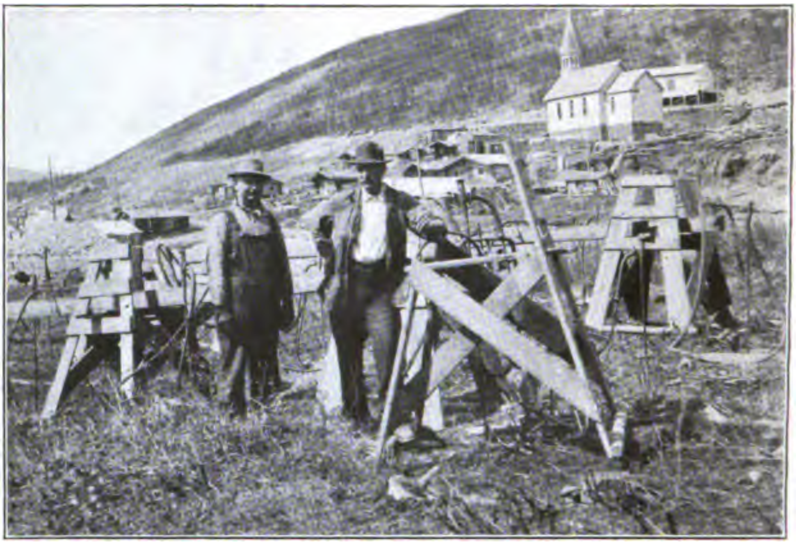
[a] Perry, O. B., Canadian Mining Inst., vol. 18, 1915, p. 42.



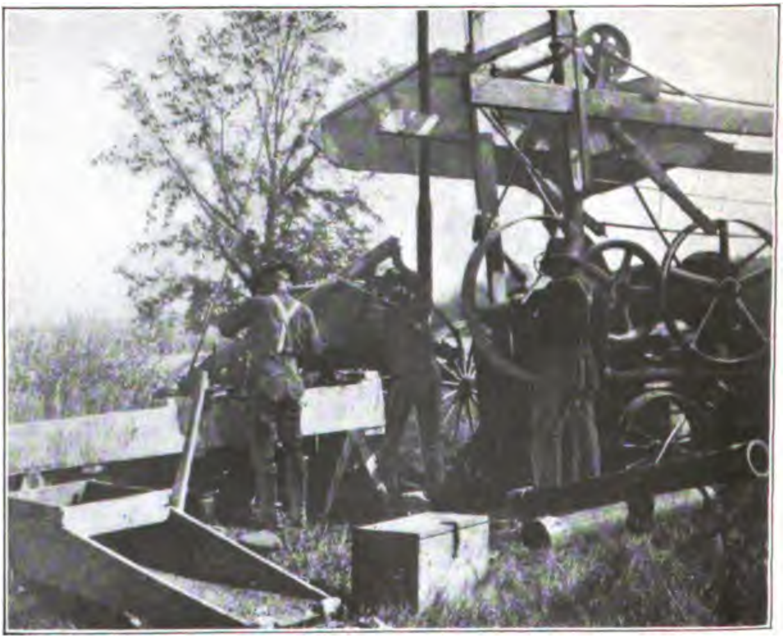
A. OVERBURDEN ON PLACER GROUND, NOME DISTRICT, ALASKA.



B. DRILLER, PANMAN, AND ENGINEER AT WORK NEAR RUBY, MONT. KEYSTONE DRILL.



A. THAWING WITH STEAM POINTS ON BONANZA CREEK, YUKON TERRITORY.



B. PROSPECTING WITH A KEYSTONE DRILL.

The creek deposits (Pl. VIII, *A*) are altogether different. They are shallow, the depths to bedrock mostly ranging about 18 to 20 feet. The gravel is angular and contains little sand, the gold is confined almost entirely to bedrock. The creek gravels are overlain with what is known locally as "muck," a deposit containing organic matter, fine sand, and water, which almost invariably is frozen, and where undisturbed is covered with a blanket of moss. The depth of this muck is 4 to 8 feet, averaging about 6 feet. In dredging on the creeks it is necessary to take up 4 to 6 feet of the bedrock, which is a massive and blocky sericite schist.

Operations in the Yukon district are subject to difficulties not met elsewhere except in Alaska. These difficulties are caused by the climate and the character of the deposits.

The extreme cold of the Northern winter limits the open season to about six months. Dredging can not begin until thawing has produced water enough to float the boats and to clear the ponds of winter ice, 5 to 7 feet thick. Work at both the start and finish of the season is hindered by cold. All moving parts, such as the tailing conveyor, must be housed and also warmed with steam coils. Also, steam must be used freely to keep the ladder and bucket line clear and to keep the gold tables in proper condition. The dredging season ends when the creek freezes, so that there is not enough water for floating the boat.

The creek deposits present many obstacles to dredging. In the first place the deposits are shallow, which necessitates frequent moving of the dredge, both "stepping ahead" and across the pond. Also the shallow ground increases the labor for necessary outside work, such as setting "deadmen" and removing stumps. The thawed muck is not difficult to handle, particularly if it is sandy, but it often contains buried stumps, roots, and other material, which tend to choke the feed hoppers and cause delays. Digging the bedrock in order to recover the gold is the most difficult part of the process. It is slow, hard work that entails heavy wear on the entire machine and necessitates continual replacement of all digging parts. Frozen ground is the greatest obstacle to dredging on a large proportion of the creeks, for the gravel must be thawed (Pl. IX, *A*) before dredging can go on successfully.

The short working season led to many departures from usual methods. Effort is concentrated on getting all that is possible out of the dredges and the result is a remarkably high efficiency for the period of operation. Each season has shown a gain in yardage per dredge hour over the season previous, until the present rate compares favorably with that of dredges of the same size in California, where conditions are much more favorable. This result is largely made possible by thorough overhauling and repair during the months of no dredging.

PROSPECTING DREDGING GROUND.[a]

To assure the success of a gold dredging venture, experience and judgment are requisite to the conduct of every detail of the work and preeminently in prospecting and examining the ground.

Many failures in dredging are traceable to lack of thorough examination of the ground before putting in a dredge. The tendency to build an expensive plant before demonstrating the value of the property was as pronounced a fault in the early days of dredging as in those of other branches of mining.

Improvements in construction and equipment and the overcoming of seemingly insurmountable difficulties have greatly broadened the dredging field and have relieved the engineer of many of the difficulties that confronted him a few years ago. A thorough investigation of a placer deposit, however, is not as simple a matter as the investor is often led to believe. Such an investigation should never be made by a man unfamiliar with the factors governing the determination of the value of placer ground and with the conditions necessary for successful dredging.

It is difficult to give any one factor as most important in the examination of dredging ground, as several conditions must be favorable to insure success. The gold value in itself may be of minor importance in valuing some ground, because a hard uneven bedrock, many bowlders, much clay, or a rough, bedrock surface contour may prevent the profitable dredging of ground with a comparatively high gold content. However, rich enough gravel, the feasibility of economic work, and enough ground to warrant the installing of at least one dredge are the salient features.

To test the value and character of the gravel and the bedrock either shafts or drill holes (Pl. X) must be sunk. Gulches, old prospect shafts, pits, or hydraulic faces may be available and give considerable information in a reconnaissance of the ground, perhaps sufficient to warrant an engineer of experience in advising his clients to make no further outlay in the examination of the property. But an engineer would not be justified in reporting favorably on the ground without making a complete examination. Besides the conditions that influence the actual operation of a dredge, such as depth and character of bedrock, possible fuel or power supply, labor conditions, etc., there are others of almost equal importance, which if unfavorable

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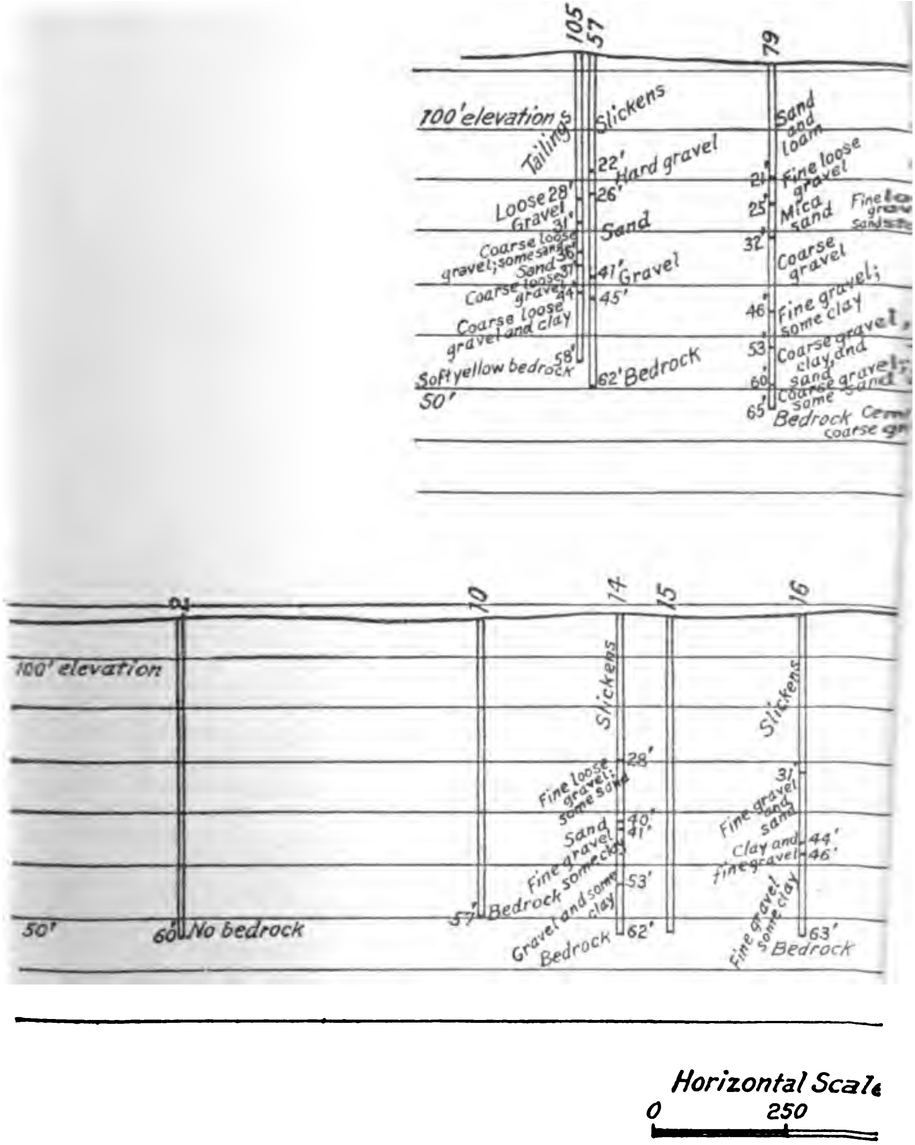
[a] In preparing this section the author has taken a chapter from "Gold Dredging in California," by W. B. Winston and Charles Janin, California State Mining Bureau, Bull. 57, 1910, as an outline. Such material as was necessary to bring the chapter up to date has been added.


105
57
79
100' elevation
Tailings
Slickens
Sand and loam
22' Hard gravel
Loose 28' Gravel 31'
26' Sand
21' Fine loose gravel
25' Mica sand
Fine loose gravel sand
Coarse loose gravel; some sand 36'
Sand 37'
Coarse loose gravel 44'
Coarse loose gravel and clay
58' Soft yellow bedrock
41' Gravel
45'
62' Bedrock
32' Coarse gravel
46' Fine gravel; some clay
53' Coarse gravel, clay, and sand
60' Coarse gravel; some sand
65' Bedrock
Cement coarse gravel
50'
2
10
14
15
16
100' elevation
Slickens
28' Fine loose gravel; some sand
Sand 40' 41'
Fine gravel; some clay 53'
Bedrock some clay
Gravel and some clay
Bedrock 62'
57' Bedrock
50'
60' No bedrock
Slickens
31' Fine gravel and sand
Clay and fine gravel 44' 46'
Fine gravel; some clay
63' Bedrock
Horizontal Scale
0
250

would result in the engineer turning down that at first glance seemed meritorious. Among these factors are transportation facilities, which govern the cost of supplies and of installation, the price of the ground, titles, and climate.

FEATURES THAT MUST BE DETERMINED.

Proper examination of a placer for dredging, therefore, involves the determination of the following conditions:

1. Value, character, and distribution of gold content.
2. Depth, character, and quantity of ground to be worked.
3. Character and contour of bedrock.
4. Water level and available water supply.
5. Costs of fuel, power obtainable, possibilities for hydroelectric power.
6. Labor, transportation, supplies, etc.
7. Surface contour and timber growth.
8. Operating costs, which is influenced by all the physical factors.
9. Cost of land, royalties, titles, etc.
10. Climatic conditions.
11. In foreign countries other factors to be considered would be taxes, duties, stability of the Government, etc.

In determining the method of prospecting, shafts should always be used, if possible, in preference to drill holes, as the latter give less reliable information in regard to the physical conditions and the probable gold content of the gravel. Shaft sinking permits a much larger sample to be taken, gives better opportunity for examining the gravel, and if the material taken out is handled carefully, lessen the chance of error in the work. Shafts, however, are necessarily limited to favorable conditions, whereas drill holes can be sunk in any class of ground. It must be remembered also that the value of nearly all the dredging ground in California and elsewhere has been computed from sampling with drills and that, if other conditions are favorable, the largest dredge operators in the world would not hesitate to finance a dredging venture the mineral content of which was estimated entirely from drilling tests by engineers whom they believed reliable.

In prospecting the chances for errors are great, and constant care is necessary. A careless drill runner may neglect to drive the pipe ahead of the drill and pump an excessive quantity of material from below the shoe when pumping the drillings, as is frequently done. Careless work may also be done in drilling compact or frozen ground where casing is not thought necessary. Sometimes such carelessness accounts for a high gold content that is not found by the subsequent dredging. In loose ground, the pipe may be driven too far before pumping and become clogged so that the volume of the core drawn up does not increase in proportion to the depth of driving. In running, sandy ground far more material than that due to the length of drive may run into the casing. Allowance must be made for this, and often the casing has to be driven several feet until it reaches a firmer stratum. Generally such loose ground is barren or nearly so; if it is not, the greatest care and judgment are necessary in handling the hole and estimating the contents. Small bowlders in loose ground may cause the displacement of considerable underlying material before a bowlder is broken or pushed aside.

The risk of salted samples from either shafts or drill holes depends upon the conditions of examination, the care taken, and the experience of the engineer. Instances of salting in both shaft and drill sampling are numerous. A common precaution when one shift is employed and the hole is unguarded overnight is to put the driving blocks on the drill stem and leave it resting in the casing. As this practice is not absolutely effective, it is well to put a pan or two of barren tailings in the bottom of the hole, to be pumped out and panned the first thing in the morning. It is more easy to salt the examining crew in drill work than in shaft sampling, but with proper care there is small danger of salting being attempted without discovery. In this connection, the following paragraph is of interest:[a]

> There are two cases in the Philippines where placer deposits were tested and reported to be of value and later were found to valueless. In one case, a dredge was purchased and installed, failing almost immediately because there was not sufficient gold value recovered. Another property was reported to be very rich, and remarkable samples were submitted by the claim owners as proof. A company was formed and stock was placed on the market, but fortunately the error in valuation was discovered before much money had been spent. It appears that the samples in the two cases cited above had been salted.

A number of attempts at salting in the United States and Alaska where the examining engineer detected the "salt" could be cited. Other cases could be given of salting and careless or inexperienced work being responsible for large losses of money spent on the property or in building a dredge.

When because of the presence of water, or for other reasons, shaft sinking is impracticable, prospecting must be done with drills. Experienced men should be engaged, and every precaution taken to insure the results indicating as nearly as possible the gold content of the ground prospected.

The gravel may be fine and sandy, medium, or coarse. In fine gravel the pebbles are usually less than 2 inches in diameter and generally are much smaller; in medium gravel, the diameter of the pebbles range up to 6 inches; and in coarse gravel much of the material is larger than 6 inches.

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[a] Eddingfield, F. T., Mine exploitation and the causes of some mine failures in the Philippines: Mineral Resources of the Philippine Islands, 1913, p. 55.

Compact gravel, if not cemented, can be worked with a pick, and a pipe can be driven in it without much difficulty. Loose gravel is easily worked with a pick, but shafts in it must be lagged. Quicksand or fine loose gravel containing an excess of water is liable to give by drilling an excess of material for the distance sunk, and the results from shafts in such ground would probably be too high as well, if indeed shaft sinking were possible.

SIZE OF PROSPECTING SHAFTS.

Prospecting shafts should be of uniform diameter from top to bottom, in order to facilitate computation if the entire content of the shaft is to be washed, but often the practice is to cut a small section along the side of the shaft from top to bottom instead of washing all the gravel. Usually it is considered good practice to check all results obtained from putting the entire contents of the shaft through a rocker or sluice box as follows: Channel down the sides, staggering the cuts at different levels, measure the cuts carefully in all dimensions at various points, then average these dimensions, calculate the volume from the averages, and weigh the gold obtained therefrom.

Square shafts are seldom used unless the ground is too loose or wet to stand and needs timbering, then shaft sinking becomes expensive and soon passes the limit of economy. In wet ground it is also the practice to use round iron caissons, usually in 3 or 4 foot sections, in place of timber lagging, and to sample the entire content of the shaft. In very wet ground, a nest of caissons made of ½-inch steel in 4-foot sections ranging from 5 feet to 3½ feet in diameter, and tapering slightly to allow them to telescope readily, has been furnished by some manufacturers. Each section has a steel band, 2 inches by ½ inch, on the outside of the top end and on the inside of the bottom. The bands stiffen the caisson and prevent slipping of the different sections. With these caissons and a good centrifugal pump operated by a gasoline engine, shafts that would be impracticable under other conditions, have been sunk in wet ground at a reasonable cost. Care must be taken at every step of the process as in drilling, and occasionally in very wet ground a small caisson is driven ahead of the shaft, and a sample is obtained before the gravel is removed from inside the large caisson. This method, it is thought, obviates most of the danger of inflow of fines from outside.

In very wet ground, however, better results may be expected from sampling with drill holes. When a heavy flow of water is encountered in a shaft, the shaft can be discontinued and the remaining distance to bedrock can be sampled with a drill hole.

A convenient size for round shafts is 41 inches in diameter, which gives approximately 1 cubic yard of material from each 3 feet of depth. Although shaft sinking is not generally considered cheaper than drill tests, under favorable conditions the cost of shaft sinkin may be considerably less than $2 per foot, according to local wage and conditions. The cost varies greatly in different places, and timbered shafts for prospecting gravel have cost as much as $25 a foot In one Colorado field, on the other hand, under favorable condition more than 100 shafts ranging from 5 feet to 30 feet deep were sun at an average cost for sinking, not including sampling, of 24 cent a foot. The cost of labor was $1.50 and $1.75 a day.

PROSPECTING BY SHAFT AND DRILL HOLES.

The following figures show the gold content indicated by drill holes sunk with a standard Keystone drilling rig, and the use of the usua factor, and by a shaft 3 by 3 feet sunk around the drill hole, all th material from the shaft being washed. The figures are of especia interest, as such tests are not often made.

Gold content indicated by a drill hole and a shaft.

Test No.	Content as indicated by drill sampling.	Content as indicated by shaft sampling.	Depth of ground.
	Cents per cubic yard.	*Cents per cubic yard.*	*Feet.*
0	3.8	6.5	12
2	34.0	35.0	25
3	9.4	6.5	29
4	9.2	50.0	25
5	16.0	14.4	15
6	13.7	25.5	14
7	5.0	7.0	17.5
8	13.2	12.3	15

Of the eight samples shown the shaft gave a higher gold content for five and the drill holes for three. Unquestionably shafts are far more reliable than drill holes for prospecting placer ground where shaft sinking is feasible, and on the ground sampled the reason for sinking shafts was that the core obtained from drill sampling was less than the theoretical core and it was desired to check the drill-hole results. Dredging on this ground has since demonstrated on other parts of it, a higher gold content than that indicated by the drill, but none of the ground prospected by shafts had been dredged at the time this was written.

In prospecting with drills in California, the Keystone No. 3 traction machine (Pl. IX, *B*) is generally used. This machine is self-contained, has an 8 or 10 horsepower boiler, and a walking beam to work the tools. Wood, coal, or oil can be used for fuel, and where electric power is conveniently available the boiler can be discarded for an electric motor.

The casing generally used has an inside diameter of about 6 inches, is $\frac{5}{16}$ inch thick, and weighs about 28 pounds to the foot. It is cut into sections 5 to 7 feet long, screw threads being added as additional pipe is required. The pitch of the casing thread generally used is about eight to the inch; but in Oroville and some other districts, heavier threads with about half this pitch are commonly used, as such a thread is thought to better stand pulling from deep holes. A number of extra lengths are kept for use at those times when all the casing can not be drawn from deep holes. This practice eliminates much loss of time and extra expense incident to cessation of operations while waiting for extra pipe or digging out pipe that can not be withdrawn by usual methods. The outside diameter of the cutting shoe is about 7½ inches. The threads of the pipe and couplings are cleaned from grit, and are generally slightly lubricated with graphite; axle grease is also used but graphite is considered preferable as the chance of getting grease in the hole is lessened. The threads of the pipes and couplings should be in good condition, to prevent loss of pipe in the hole during pulling.

Drilling without casing has been done in hard ground that will stand without the pipe, but is not good practice, and if the result shows high values, they can not be accepted as truly indicative of the gold content. Moreover, ground hard enough to stand drilling without the casing may be too hard for the dredge to handle. In Alaska it is customary to drill frozen ground without using casing, although ground that is only partly frozen should be cased.

The casing pipe is marked with plain figures for each foot in depth. If the operator desires, some private mark may be used. The drill rope is also marked. Records of depth and all details of operations are usually kept by the panner. Unless the record of pipe depth is accurately kept, it becomes almost impossible at times to determine whether the drill bit is striking above or below the cutting shoe. For the purpose of a constant base to measure from, it is best to place a fairly long, wide, and thick board on the ground near the drill hole on which the drill man can also stand while working. Also the marking of the drill rope should be occasionally checked, as ropes stretch and bits wear with use.

The bit and stem, which weigh 800 to 1,000 pounds or more, are allowed to drop with a slight slack in the cable, so that their impact loosens the gravel. Tests have demonstrated the advantage of quick, long strokes, 36 to 40 inches long, 56 to 60 a minute. With a slow stroke the loosened material is apt to settle between strokes, causing most of it to be recut and also causing possible loss of gold, aside from the loss of time. Obviously, a hole will be sunk quicker if the bit hits the clean core at each stroke, but the saving of time is a secondary consideration in sampling placer ground, accuracy of results being the prime factor.

HANDLING OF BITS.

Bits are of several types, the bit most suitable for gravel and sand being thin bladed; for drilling large bowlders or into hard bedrock this bit is sometimes replaced by a heavy rock bit that has a cutting edge of wider angle. In loose gravel such a bit would pack the material and might drive some of it from below the cutting shoe to the side of the pipe. It is important that the bit be kept sharp and well beaten out at the edges. Excessive drilling through the use of a dull bit may cause flouring of the gold.

USE OF SAND PUMP.

The sand pump is a hollow steel cylinder, 8 feet long and 4 inches in diameter, which has a valve at the bottom and a piston, or sucker rod, provided with a valve. The piston travels the whole length of the cylinder, going to the bottom of the pump when lowered, and when drawn up rapidly causing a vacuum that draws in sand, muddy water, and small stones. The efficiency of the pump depends on the rod being lifted quickly. Occasionally the valves require repairing, and sometimes an obstruction in the bottom valve seat, which can be readily repaired, causes leakage. A good pump will draw up everything in the casing that is loose and small enough to run in; gold and other minerals are drawn in with the sludge and are held in the pump by the foot valve. In drilling above the water level or in ground containing little water, some water must be kept in the casing to facilitate drilling and permit pumping. In drilling fine or medium gravel it is usually advisable to send the sand pump down immediately after driving the casing, in order to prevent soft material being forced out of the core through the coarser stones by the weight of the drill. Often it is possible to pump out the entire core without drilling at all. Various mechanical troubles occur during drilling, but are hardly worth enumerating as they are easily repaired by an experienced drill runner.

PROCEDURE IN DRILLING.

In starting a drill hole at the surface a shallow hole, like a post hole, is usually dug and the first section of the casing, with its shoe attached, is tamped in firmly, and plumbed with a spirit level. Care must be taken to keep the casing perpendicular, otherwise it may become bent and the hole lost before bedrock is reached; also a bent pipe is difficult to pull. It is usually advisable for the helper to climb the scaffold and steady the tools (bit, stem, and socket) with his hands for the first few feet of drilling, so that the tools will not hit to one side of the center and knock the casing out of plumb.

If the driving shoe strikes the sloping side of a hard bowlder, the casing may bend and cause the loss of the hole, though in fairly loose material the work may be continued by turning the pipe slightly in the hole.

If the hole is started in top soil, the casing generally is driven until firm ground is reached, and in hard gravel far enough to remain in place while drilling goes on. In loose gravel some men drive the pipe 10 or even 15 feet before beginning to drill. This procedure, though permissible in places where the upper gravel is known to carry little or no gold, is dangerous where the depth to the mineral content is not known; but the practice is often encouraged by the drill runner, as it makes time. After the pipe has been driven the desired depth the core is loosened by the drill for a depth of about one foot and then pumped out. This process is repeated until all the core, except a few inches of the bottom, has been removed. Some operators drill several feet before pumping, as greater speed is gained in this way, but pumping is generally done for each foot or less drilled, especially when drilling in the "pay streak," where several pumpings are necessary to clean out the drilled core. In drilling where the location of the pay gravel is known and the material above the pay is thick and barren, it is the general custom to drill 3 to 5 feet for each pumping while going through this overburden.

HANDLING OF DRILL CORE.

During the drilling of a hole a core sufficient to keep outside material from entering should be left in the pipe before and after every pumping. The length of this core varies with the nature of the ground, but generally is 2 to 6 inches; it should be long enough to prevent any inrush of water or material from the sides. Water is also kept in the drive pipe to offset the pressure from the outside, as well as to facilitate drilling. When drilling cemented gravel or when the drive shoe encounters hard bowlders it sometimes becomes necessary to drill below the casing. In such an event the pipe should be driven after drilling and before pumping whenever possible. Occasionally it is necessary to pump material from below the drive shoe. Any increase in the gold colors panned at such times must be especially noted and the quantity of material must be compared with that recovered under normal conditions. Some operators advocate measuring the material from each foot drilled, and while this would be good practice in drilling below the shoe the usual custom is to measure all the material at the end of the work at each hole.

The gold content of a placer is seldom evenly distributed, but is usually concentrated near the bottom, one-tenth of the material drilled often containing most of the gold; at times gold is found in different layers throughout the placer, as described in the chapter on geology, and the greatest care in drilling the rest of the ground becomes useless if an error is made while the pay streak is being drilled.

Any material that adheres to the drilling tools as they are raised from the hole is washed back into the casing, so that it may be brought up in the sand pump. The drill runner keeps a bucket of water at hand for this purpose. Two or three pumpings are generally sufficient for each foot drilled, but the number depends on the material encountered; from some holes the pump will bring up less than one-half a pan at a pumping and from others even two pans. The sand pump is used until all the drilled material is obtained. If the drill has been working below the casing excess material is likely to be obtained by repeated pumpings and a careless drill runner has often "salted" himself in this way. The contents of the sand pump are emptied into a sample box made either of wood or galvanized iron, the solid material generally being caught in a gold pan placed inside the box. The contents of the pan are washed and any gold colors counted and recorded. The tailings from the gold colors are generally segregated in sizes according to an arbitrary classification and recorded on the drill log under the headings, one, two, or three; bunches of small colors representing one or more colors of the various sizes are grouped by an experienced panner.

The following figures from the drill logs of one examination show the total number of colors, counted, under the headings, for three holes. Two of these holes were drilled within a few feet of the third in an attempt to check the results first obtained. The depth drilled in each hole was about 60 feet.

Arbitrary classification.

Hole No.	Size of colors.			Total weight of gold.	Gold content of ground.
	1	2	3		
				Mg.	*Cts. per cu. yd.*
A..........	14	79	117	725	72.4
B..........	5	39	338	327	32.3
C..........	3	8	342	125	12.5

Weighing tests are made during the first stages of the examination to determine the weight of gold colors and to enable the panner to become familiar with some of the characteristics of the gold. On extended examinations good panners become able to approximate closely the gold content of the sample when panning the material from the drill hole.

Gold colors vary greatly in different districts and a man familiar with one district may be misled at first when panning in a new field. On one examination the gold colors weighed 200 to a cent and on another 1,200 colors were required for the same weight. In different fields the colors from drilling may vary from such sizes to those weighing a milligram or more. In Alaska and in northern California colors weighing several milligrams are occasionally brought up by the sand pump.

PANNING.

Panning is done over a tub and the tailings from the pan are afterwards rocked by the panner, as is also any material left in the sample box when enough has accumulated or when the panner has time. A simple method of handling the cores is to place the rocker on its proper grade at the overflow end of the sample box. If an apron is used in the rocker and it is occasionally shaken a little to prevent caking, very little gold will get away with the overflow. Then, after the hole is down, or at any convenient time, the panner can take the residue from the tub, put it back in the sample box, measure it, and then run the whole through the rocker. In rocking care should be taken to prevent caking at the lower end by stopping occasionally and scooping the material from the lower end back into the rocker screen. The tailing from the rocker is tested by careful rerocking or panning in order to check the washing of the sample.

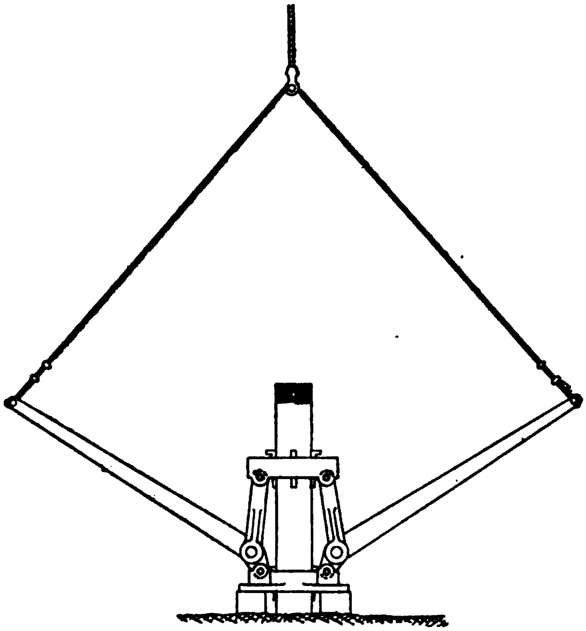

FIGURE 1.—Patented device for pulling drill-hole pipe.

After the core drilled has been removed by the sand pump, the casing is again driven and all operations continued as before until the hole is down to bedrock. When bedrock is reached, repeated use of the sand pump is necessary to remove all the drillings from the hole, and sometimes when the bedrock is soft the pump loosens material that has not been drilled. The color of the drillings from bedrock is generally noticeable and readily recognized. If the colors recovered appear excessive, the drill hole may have cut a rich seam or a small pocket and the result should be noted separately from the record of the rest of the hole.

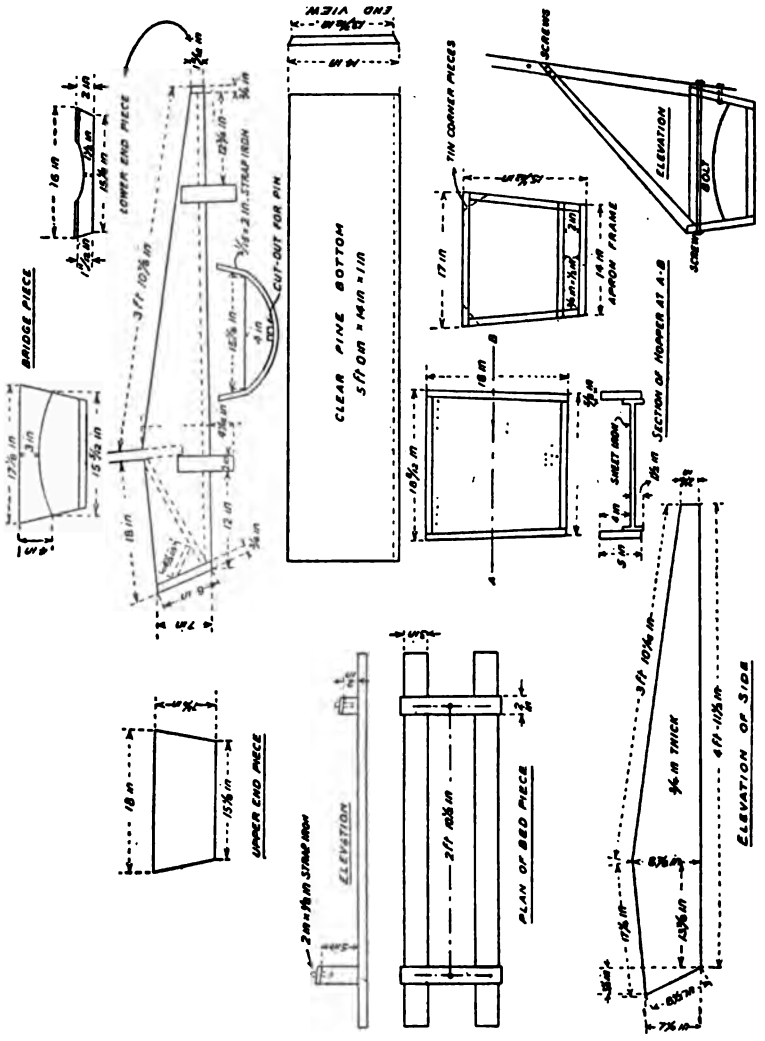

FIGURE 2.—Details of rocker.

The hole being finished, the next operation is pulling the pipe. Sometimes considerable difficulty is experienced in pulling the casing from a deep hole. A pulling jar and pulling head are generally all the special equipment necessary when a hole is shallow; with a deep hole the jars may not be able to loosen the pipe and two jackscrews may be required. A patented pulling appliance that seems to have considerable merit is shown in figure 1.

Boxes for the pumpings are made in different sizes; some are 12 by 14 inches by 8 feet long. A convenient box is made of 2-inch planks, it measures 20 by 20 inches and is 5 feet. Such a box held together with rods, is strong and easily transported. A galvanized-iron box is more easily cleaned and transported than a wooden box and will be found decidedly satisfactory on extended examinations.

The size of the rocker varies with the personal ideas of the user and is often governed by the size of the material obtainable. A good size is 18 or 20 inches wide by 4½ or 5 feet long (see fig. 2) and is used either with or without an apron. The use of the apron sometimes facilitates rapid work, because the tailings are not rerocked except occasionally to test the rocker efficiency. In the plain rocker the tailings are usually rerocked one or more times, according to the gold content of the ground. Clean water is always preferable for both panning and rocking. Some engineers line the bottom of the rocker with burlap. The writer has made tests with burlap, corduroy, canvas, and blankets and considers ordinary burlap, which is always obtainable, the most satisfactory. Some firms supply a steel rocker that probably has some advantages. For examinations, in districts where suitable lumber for a rocker might not be available experienced engineers generally carry a rocker that can be taken apart and compactly packed. A steel rocker might answer the purpose equally well.

The tubs in which the panning is done are either galvanized-iron washtubs or half barrels. The fine colors from the pannings and rockings are usually kept in a small dish and are amalgamated at the completion of the work. The amalgam is put in a small vial which is marked with the number of the hole, and the quicksilver is later separated from the gold by first using diluted and then strong nitric acid, after which the gold is washed, dried, and weighed. Some operators use no quicksilver and separate the black sand from the gold by close panning and a magnet, then dry the material, remove any remaining black sand by blowing, and weigh the gold.

If the gold does not readily amalgamate, the fact should be noted in the log book and the cause ascertained. Poor amalgamation may be due to a coating of iron oxide, the presence of arsenic or grease, or to some other cause. A common cause is the fouling of the quicksilver by grease and other impurities; these should always be kept away from the pans; but grease from the pipe couplings and from the foot valve of the sand pump will often get in. Often a little caustic potash will solve the difficulty of amalgamation, or even common salt kept in solution on top of the mercury. Sometimes so-called rusty gold will amalgamate after being rubbed with the finger against the pan. Although it is often necessary in cold weather to use warm water to facilitate amalgamation, it is best to amalgamate at as nearly as possible the same temperature as will prevail on the dredge. Methods of saving the gold in sampling that can not be duplicated or equaled on the dredge may lead to error in calculating the value of the ground. The physical characteristics of the gold determine in a great measure the percentage of recovery on the gold-saving tables of the dredge. If much light, thin, scaly, and flour gold is encountered in prospecting, the fact should be carefully noted, as there may be difficulty in saving such gold on a dredge handling large quantities of sand and gravel.

The black sand caught in the pan and riffles with the gold seldom contains anything of value. However, there is a possibility of finding platinum and various minerals of high specific gravity, and in a new district it is always advisable to investigate the black sand carefully. For this purpose a good microscope can be used to advantage.

The practice among engineers inexperienced in placer work of using the fire assay to determine gold content is rarely followed these days. A few years ago the practice was more widespread than would be imagined; numerous reports upon new districts have been based upon such methods which, needless to say, were worthless.

PROSPECTING RECORDS.

In prospecting with drills and shafts a systematic set of logs must be kept for calculations and tabulations.

It is surprising how much expensive work has been done in preliminary investigations without a systematic record being kept for future reference. Especially has the writer found this to be true of Alaskan fields; often a note of the depth and a rough estimate of the total amount of gold recovered from a drill hole or shaft that has been sunk at a cost of perhaps $100 or more are all the information available of work done, or else the whole matter was left to the memory of the drill runner. Such results might be of some interest, but would prove of little value in estimating the gold content of the ground. The logs should give all the necessary information in as simple and comprehensive a manner as possible. Some of the largest dredging companies in California have used for a number of years field, time, and prospecting logs similar to those following and find that they answer every purpose.

Form of drill log used by large California dredging company.[a]

Date ....

Hole No. .... Crew No. .... Tract ....

Started hole .... Finished ....

Depths.			Cores.			Number of pans.	Colors.			Firmness.		Lumps (clay).	Slime.	Formation.
Depth represented.	Depth of pipe.	Depth drilled below pipe.	Core after driving.	Core for depth driven.	Core after pumping.		1	2	3	Driving.	Drilling.			

Total depth ....

Depth to bedrock ....

Depth to water level ....

Measured volume ....

.... Panman.

Form of time log used by large California dredging company.[b]

Date ....

Hole No. .... Crew No. .... Tract ....

Started hole .... Finished ....

Time of day particular operation is finished.	Nature of work.	Hours and minutes consumed in—						Depth of pipe.	Remarks.
		Drilling operations.	Oiling—putting on pipe.	Pulling pipe.	Moving.	Setting up.	Delays.		

.... Panman.

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[a] Form used by company is 5½ by 9 inches and has space for 24 entries instead of 4.

[b] Form used by company is 5½ by 9 inches and has space for 30 entries instead of 4.

Form of prospecting report used by large California dredging company.[a]

Examination .... Date ....

No. of hole .... No. on map .... Crew No. ....
Location of hole ....
Method of prospecting by .... Machine used ....
Started hole .... Finished hole .... Days worked ....
Depth of top soil ....
Depth to bedrock ....
Depth drilled ....
Depth used in calculations ....
Depth to water level ....
Distribution of gold ....
Nature of top soil ....
Nature of gravel ....
Nature of gold-bearing strata ....
Nature of bedrock ....
Time consumed in: Moving .... Setting up ...... Drilling operations ......
Pulling pipe ...... Delays .... Total ....
Causes of delays ....
....
Mg. of gold recovered ....
Factors adopted .... Fineness of gold ....
Cu. ft. per ft. depth of hole .... Average value per cubic yard ....
Remarks ....
....
....
....

Signed ....
Checked by ....
Approved by ....

The field and time logs, which are kept in the field, usually by the panner or the men in charge of the drill, are made in convenient size books, about 4½ inches wide by 7 inches long, containing 50 leaves each. The paper is tough, strong, and not easily torn when wet; the books can be carried in the pocket and readily used in stormy weather. It facilitates matters to have the leaves of distinctive colors, those of the field log being usually green, and those of the time log yellow or whatever color suits the fancy of the man in charge. These books are now obtainable from technical book shops. The leaves of the field log are perforated; they are torn out at the completion of each hole and the information is copied into the prospecting log book. In the columns giving the sizes of the colors, in the field log, the largest are usually marked under No. 1 size, and the smallest under No. 3, but this method of grading the colors can, of course, be changed to suit any desired arrangement. The columns for depth and core indicate depth after pumping and the length of the core left in the drill hole after pumping. These figures added give the total depth of pipe, except where an inrush of material would cause an excess of core, which would be noted under "remarks."

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[a] Form used by company is 8¼ by 10½ inches.

In the time log is kept a record of all work done and of all difficulties and delays. The record is useful for estimating the time required to prospect similar ground. For promotors and investors who are anxious for speed in prospecting, a record of this kind would be an education.

The prospecting sheet or log is also made with perforated leaves and should give a summary of the field and time log, as well as all other necessary information pertaining to drill holes or shafts.

SPEED OF DRILLING.

No average speed can be given for test drilling. Climatic and other conditions vary greatly in different localities, and an unforeseen accident to the drill or equipment might delay operations considerably. Speed should be subservient to accuracy in prospecting. However, some idea of average speed can be gained from the time required for actual tests. At one place as much as 20 feet was drilled in 12 hours in loose gravel, whereas only 2 feet was drilled in the same time in cemented gravel. On shallow ground, the frequent moving and resetting cause considerable delay, the amount depending on the distances between the holes and the difficulty in moving.

The speed of drilling generally decreases with depth. More time is required to raise and lower the tools, the casing is driven more slowly because of frictional resistance, and the casing is harder to pull. Sometimes it takes more time to pull the casing than to drill the hole, and occasionally it is economical to leave casing that does not pull easily.

On one property, the average depth drilled, including delays caused by bad weather, moving, etc., over a total period of thirty-three 10-hour days, was approximately 12.6 feet per day. The holes averaged 24.9 feet deep, the character of the deposit was medium coarse gravel, free from clay and overlain by hydraulic tailings. As the wages of the drill crew were $12.25 per day, the labor cost was seemingly $1.01 per foot drilled. In another examination three 70-foot holes required eight 24-hour days, including pulling pipe, moving, etc., and the cost for labor was $1.71 per foot. In one examination 2,500 feet of holes was drilled; the total cost of drilling and sampling on the property, exclusive of the salary of the man in charge, approximated $1.30 a foot.

COST OF DRILLING.

The following figures are from drilling in a rather loose deposit of sand and gravel, the bowlders of granite being easily broken by the drill. Twenty-four holes of an average depth of 23¼ feet, or a total of

558 feet, were sunk in twenty-six 9-hour shifts, including all delays. The cost of work on the property was a little less than $1 a foot; but as some of the men were brought from a distance their wages and expenses while traveling made the cost of the examination slightly higher.

The drill crew consists of one drill man or drill runner, one fireman or helper, one waterman and team, and one panner. When more than one shift is used, one waterman is usually able to supply the drill with fuel and water and to help in moving the drill and tools from one hole to another. The panner generally works during the daytime only, his place being taken during the night shift by a watchman, who keeps the time log, while the night runner takes care of the pannings, placing the pans in a tub in regular order, so that in the morning the panner by reference to the log can determine just where the pans belong. On some examinations where good artificial light is available and in the far north with the long period of daylight hours, two or three full crews are employed and sampling is continuous.

The daily wages of the drill crew are usually as follows:

Drill man	$4.00 to $5.00
Helper	3.00 to 3.50
Panner (white)	4.00 to 7.00
Panner (Chinese)	2.00 to 3.00
Waterman and team	4.00 to 5.00

Other expenses beside the wages of the drill crew include fuel, repairs, maintenance, and hire or purchase of a drill. In the winter months heavy rains increase the cost of transportation, making moving difficult, and greatly delaying work. A serious accident to the drill machinery may also delay the examination for several days and increase the cost considerably. In California the cost of drilling is generally considered to be $1.50 to $2.50 a foot under favorable conditions, and these figures can be used as approximations for other fields.

Too much stress can not be laid upon the care necessary to assure the samples representing as nearly as possible the value per cubic yard of each place drilled. If the work has been properly done in every detail, an estimate of the gold content of the property can be made with some degree of accuracy; if, on the other hand, the work has been carelessly done by unreliable or inexperienced men the results obtained are worthless.

PLACING OF PROSPECT HOLES.

It is generally conceded that careful washing of the material sampled will equal or exceed the recovery by the gold-saving tables of the dredge; hence, if the work has been properly done each hole may be taken as indicative of the value of that part of the ground in which it is sunk. The placing of holes is important, for even though the mechanical details of sampling each hole be most carefully done, the average values obtained may be far from representing the gold content of an area under consideration. Too many holes sunk in the richer ground would make the indicated average content too high.

In preliminary examinations (Pl. VIII, *B*) and in prospecting large areas, a few holes are first sunk to determine the probable value and extent of the workable ground. The preliminary holes should indicate whether the gold occurs in wide or narrow channels or whether its distribution is fairly even. A common method in preliminary prospecting is to choose what appears to be the most promising ground—possibly by contour features or by old workings—and to sink a row of holes across the channel, if one is found. Then another such spot is chosen and another row of holes put across. If the gold content is sufficient, an intermediate row is sunk; and if the results justify the expense of a thorough examination, the ground is laid out in regular divisions and prospected systematically. For careful prospecting the holes are not drilled in squares, but are staggered. The ground under examination is then divided into sections or squares and the holes are proportioned to each section. Where sampling has disclosed the existence of a channel, or narrow belt having a higher mineral content than the area as a whole, the outlines of the channel or belt should be determined as closely as seems warranted.

The points for taking samples can be determined only after a study of the ground. There is no rule as to the number of holes to be sunk, but ground that shows an uneven distribution of values obviously demands more tests than grounds in which the distribution is fairly uniform. Great economy of time is possible when the work is carefully planned and the sampling crew consists of men well trained and well directed. In large tracts, where the time for the examination is limited, tests to confirm samples taken in preliminary prospecting may be concentrated on an area sufficient to determine the advisability of purchasing the property and of installing at least one dredge.

VOLUME OF DRILLINGS.

The quantity of material recovered per linear foot drilled varies greatly with different material. In drilling compact gravel, the volume of core will be greater than where a number of hard, coarse pebbles are encountered. In drilling sand or fine gravel containing much water, an excess of material may be pumped, or if the casing is driven too far ahead of the drill, the pipe may become clogged and too little material be obtained. Some operators use a measuring box holding 1 cubic foot, and measure the material as it goes to the rocker, calculating that 20 cubic feet of material so measured represents 1 cubic yard actually drilled. This factor was obtained from results of numerous experiments in one field and can not be applied as a general rule. Other engineers have found that generally the quantity of material obtained exceeded the theoretical quantity by a larger margin.

Assumptions that the expansion of gravel on being taken out of the hole is very large are unwarranted; the material crushed by the drill is poured from the pan and laid down under water, under conditions closely similar to those under which it was laid down originally.

In some fields the actual recovery of material from some drill holes, in compact gravel, has been 95 per cent of the theoretical quantity; in running ground the recovery is often in excess of the theoretical quantity of core drilled. Good practice is to measure the total quantity of material pumped from each hole, and to compare the results from all holes drilled in similar material; thus, a fair average can be obtained and used as a check. As previously mentioned, greater care is necessary with material containing the mineral content than in barren ground.

A simple method of making the measurement of the total amount of material is to dig at the end of the rocker a hole about 3 feet square, and of sufficient depth to hold all material from the hole to be drilled. All the tailing from the rocker may be run into this hole and allowed to settle; the cubic contents can then be determined and tabulated.

In drilling dry ground containing small gravel with considerable sand the material recovered will be closer to the theoretical quantity than in drilling in other kinds of gravel. The theoretical quantity that should be recovered in drill tests is represented by the contents of a cylinder having a base the diameter of the cutting shoe and a length equal to the depth drilled.

DETERMINING GOLD CONTENT OF GROUND.

The area of a circle the size of the cutting shoe ordinarily used with the power drill is about 0.3 square foot, so for every linear foot drilled there would be excavated 0.3 cubic foot, or 0.011 cubic yard. As the cutting shoe soon becomes worn, the figure used is 0.01 cubic yard, or 1 cubic yard for each 100 feet drilled. Using this factor, the method of determining values is as follows:

FACTORS USED.

The value of gold recovered from a drill hole is determined in cents. In a new district, the fineness of the gold must be first determined or assumed. The value in cents is divided by the number of feet drilled, and the result multiplied by 100, which gives the value per cubic yard. A less simple method is to use the factor 0.27, multiplying the depth drilled by 0.27 and dividing the figure for the value of the recovered gold in cents by the product, the result being the value per cubic foot. To get the value per cubic yard the result is multiplied by 27.

One operator, who concedes the correctness of the results obtained by the use of these factors for calculating the results of drilling compact gravel, believes that different factors are necessary when holes are sunk in other material; he uses the following factors as numerous tests have demonstrated to his own satisfaction that they are correct:

Factors for computing value of drill-hole material.

(*a*) For compact gravel	0.01
(*b*) For medium gravel	.011
(*c*) For loose gravel	.012
(*d*) For loose gravel, and sand with much water	.013

In using these factors, the number of feet drilled is multiplied by the factor selected and the product is divided by the value in cents of the gold in the sample, the result being the value per cubic yard. The selection of the correct factor is highly important, and demands much experience.

If factor *a* of the method last mentioned be taken, the results of using any one of the three methods will be the same, as is shown by making a calculation for a hole 40 feet deep that returns gold to the value of 15 cents.

First method:

$$(15 \div 40) \times 100 = 0.375 \times 100 = 37.5 \text{ cents per cubic yard.}$$

Second method:

$$15 \div (40 \times 0.27) \times 27 = (15 \div 10.80) \times 27 = 1.389 \times 27 = 37.5 \text{ cents.}$$

Third method:

$$15 \div (40 \times 0.01) = 15 \div 0.40 = 37.5 \text{ cents per cubic yard.}$$

However, if the factor *c* be used, the result is:

$$15 \div (40 \times 0.012)\ 15 \div 0.48 = 31.2 \text{ cents per cubic yard.}$$

Some engineers in calculating the results of sampling by drills employ a factor different from the one in general use. They reduce the figure for the gold content of the place sampled to a value that their experience indicates more nearly represents the recovery. This practice, practically a combination of drill-hole factor and recovery estimate, involves a discount from the contents that would otherwise be assumed from the regular drill factor. Engineers following such a practice should mention it clearly in their report. Failure to do so might cause the future of a property to be further discounted by parties to whom the report was presented if they were not conversant with the methods of computation used.

One large operating company uses the factor 33⅓ instead of 27 in these calculations, the formula for the example mentioned above being:

$$15 \div (40 \times 0.333) \times 27 = 15 \div 13.32 \times 27 = 30.6 \text{ cents per cubic yard.}$$

As will be seen, the use of this factor discounts the gold content assumed for the drill hole by nearly 20 per cent as compared with the use of the factor 27.

To estimate the gold content of the area examined, the value per cubic yard as found above is multiplied by the depth of the hole in feet. The first step is to see that no erratic values for such holes are included in the general average, as the use of one such high value might vitiate the total result. A rich sample indicates a rich spot, but does not show its extent. The general practice is to reduce erratic values to a value more nearly in keeping with the general average. In this connection, Eddingfield[a] says:

> Several years ago an engineer put down 9 holes to test a certain tract of placer ground. One hole showed almost 21 pesos per cubic meter (16 pesos per cubic yard); the other 8 holes each showed only 1.30 pesos per cubic meter (1 peso per cubic yard). The engineer averaged the 9 holes and by doing so obtained a value almost three times as large as it would have been had the one high value been lacking. The error of such a method is apparent.

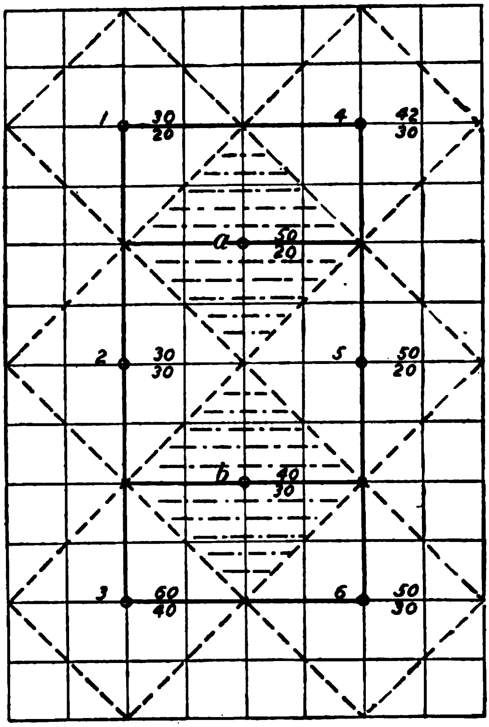

FIGURE 3.—Diagram showing weight to be given results from different holes.

After the reduction of any high values, the sum of the figures for all holes, in cents per foot, is divided by the sum, in feet, of the depths of all holes, provided the drill holes have been equally spaced and each represents practically the same amount of ground. The main difficulty in this calculation is to select the area represented by each hole. If we assume that the given property can be divided into a number of squares of equal size and a drill hole put in the center of each square then the areas represented by the individual drill holes will be of the same size. This condition is rarely found in practice, and the following example of an extreme condition is given simply to illustrate the method of calculation though in practice it would be unlikely that the holes would be so inefficiently placed. The calculations are made to cover the area inclosed within the heavy lines shown in figure 3, though the drill holes could properly represent the area covered by the eight squares.

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[a] Eddingfield, F. T., Mine exploitation and the causes of some mine failures in the Philippines: Mineral Resources of the Philippine Islands, 1913, p. 54.

Holes *a* and *b* would represent the shaded squares, and holes 1 to 6, while representing the same area as *a* and *b*, could be used only with a weight proportional to the part of their respective square that is inclosed within the heavy lines.

Data used in calculating gold content of area shown in figure 3.

Hole No.	Value of gold content, cents per cubic yard.	Depth, feet.	Proportional area.	Volume (column 3 x column 4).	Total foot cents.
1	2	3	4	5	6
1..........	30	20	0.25	5	150
2..........	30	30	.50	15	450
3..........	60	40	.25	10	600
a..........	50	20	1	20	1,000
b..........	40	30	1	30	1,200
4..........	42	30	.25	7.50	315
5..........	50	20	.50	10	500
6..........	50	20	.25	7.5	375
Total...	....	....	....	105.0	4,590

$\frac{4590}{105}=43.7$ cents per cubic yard.

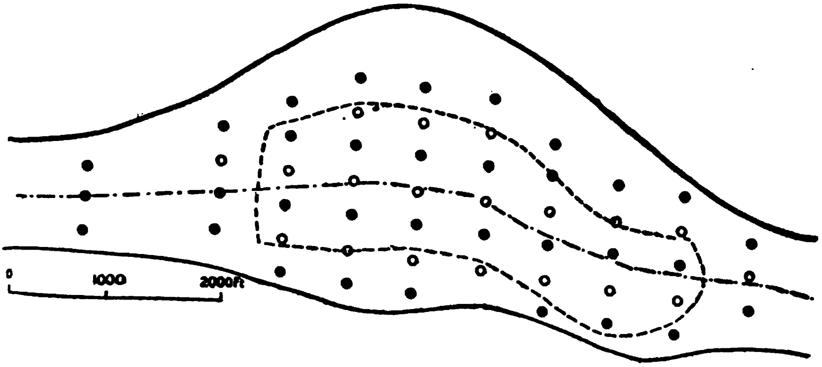


FIGURE 4.—Map showing arrangement of drill-holes in a certain area. Solid circles show first drill holes; open circles show second series of drill holes; dotted line shows area of pay gravel; dot-and-dash line shows deepest part of deposit.

Figures 4 and 5 represent prospecting maps of two different areas of dredging ground and show the method of arranging drill holes. These areas were purchased and equipped as a result of the prospecting work indicated by the maps. The illustrations show actual and not theoretical field conditions.

Sometimes it is advisable to disregard the results for certain sections, perhaps one end of a property under examination, the results for the remainder indicating that its size and gold content warrant development, whereas, if the results for all the holes drilled were averaged, the figure so reached might not indicate development to be warranted. There are instances in California where dredging of part of a property would have been profitable, but the necessity of having to purchase a large unproductive acreage in order to procure the desirable part made the project an unattractive mining investment.

In estimating the value of the gold contained in an area to be dredged, it is the general custom to take a percentage, generally 75 to 80 per cent of the total value indicated by the prospect drilling as indicative of the amount of the gold content recoverable by dredging. The percentages mentioned are based on several tests in the Oroville and other California fields.

The following reference to the Pato property is taken from the report of the Oroville Dredging Co., dated September 30, 1911, to show the prevailing practice in discounting the indications furnished by prospecting:

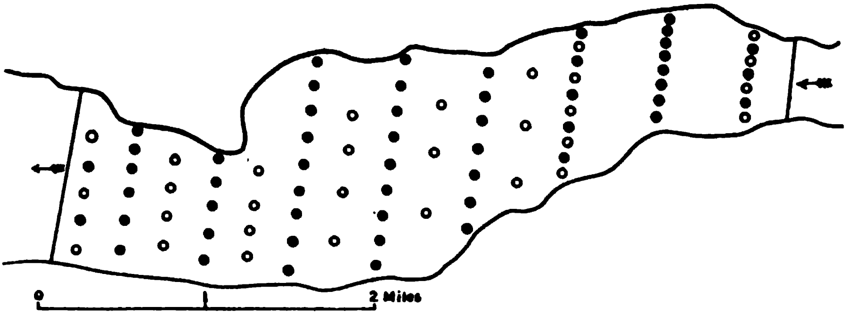
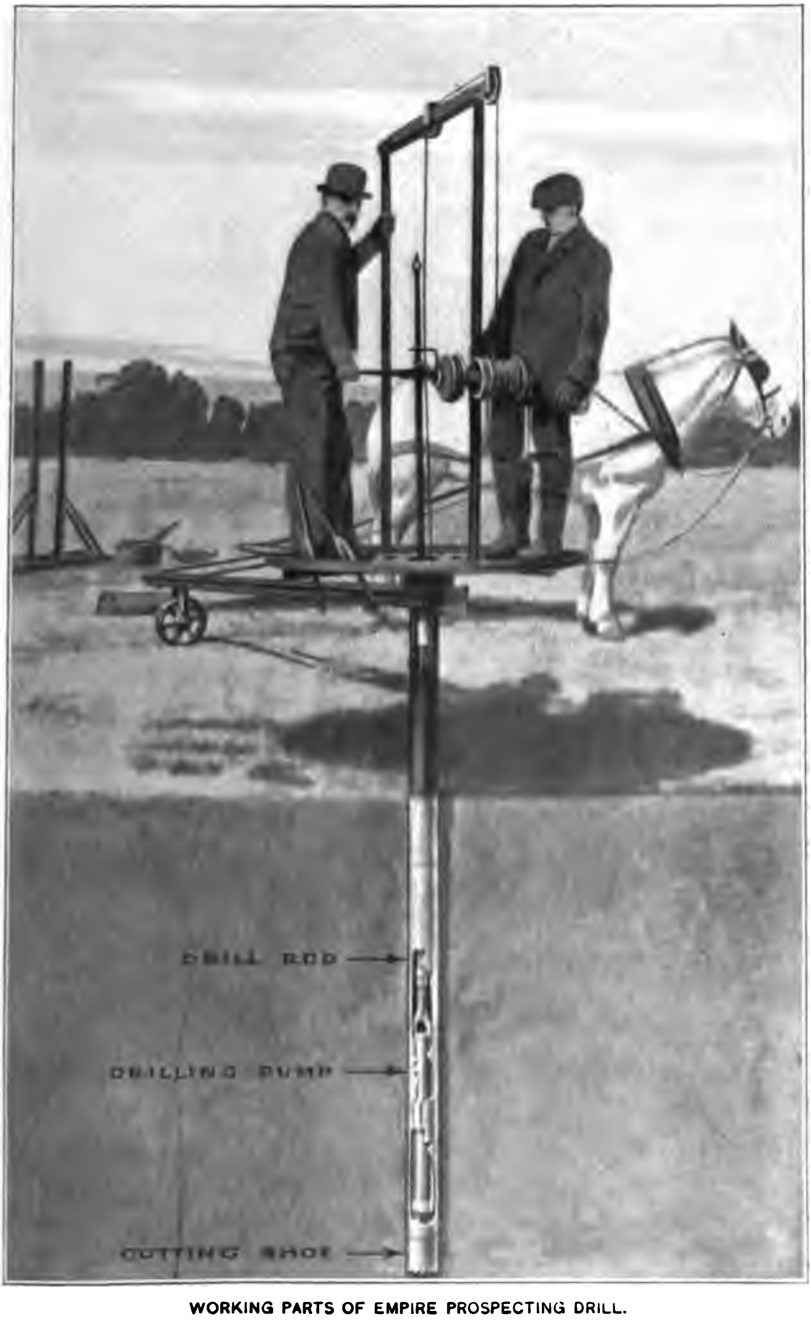


FIGURE 5.—Map showing arrangement of drill holes in another area. Solid circles show first series of drill holes; shaded circles show second series of drill holes; open circles show last series.

> The original estimates of the engineers give us an area containing 13,637,000 cubic yards of material yielding gross $3,202,986, there having been deducted from the values recovered by prospecting tests 25 per cent for variations, possible losses, etc.

It is, however, impossible to give any fixed percentage to offset the effects of various conditions of examination and operation.

The reliability of the tests and the accuracy of the average value indicated either by drilling or by sinking shafts depend on the care taken in prospecting, the number of samples procured, the location of the holes in relation to the deposit, and the experience and ability of the men in charge. If these factors are satisfactory, the estimate of the value of the property should require no discounting other than to recognize that the recovery of the dredge is always less than the gold content indicated by prospecting and that the returns on the investment should be enough to compensate for any risks taken. After the gold content of the property has been estimated it is necessary to determine whether the returns over working expenses and



WORKING PARTS OF EMPIRE PROSPECTING DRILL.

amortization of capital invested will allow sufficient dividends to make investment attractive.

TYPES OF HAND DRILLS.

The calculations outlined are based on sampling with a Keystone or similar power drill. However, in recent years the hand drill has been used extensively with great success, particularly in preliminary tests where the lower first cost of the hand drill and its lighter weight, which permit transport by either pack animals or men, are important considerations.

EMPIRE DRILL.

Of the hand drills in use in the United States and Alaska, the Empire drill (Pl. XI) is the most widely used. Briefly, the drill consists of a pipe, a platform, the rotating and driving devices, and the boring, drilling, pumping, and pulling tools. The operation is as follows:

The pipe, having a toothed cutting shoe on the bottom and a platform on the top, is placed in an erect position at the spot where the sample is to be taken. Where practicable, a small auger the size of the casing is generally used to start the hole. The material removed by the auger is washed with the rest. Some of the crew steady the pipe and others stand on the platform and with a battering ram drive the pipe into the ground until it will stand without help. The rotating device is then attached and a combination drilling and pumping tool screwed to the rods is used as a churn drill, the pipe being rotated simultaneously. While it is thus kept loose in the ground, it is sunk (1) by the weight of the pipe, (2) by the weight of the platform attached to the top of the pipe, (3) by the weight of the drill men who stand on the platform, (4) by the weight of the rod and tools that the drill men operate while standing on the platform, (5) by the jarring effect of the tool striking on the bottom of the hole. As the pipe sinks into the ground a core accumulates, and the churning crushes this core and forces it into the pump. Despite the contention of manufacturers, the hand drill usually requires more than rotation and weight to sink it and driving with a battering ram will almost always be necessary.

When the pump is filled it is removed from the pipe and its contents are emptied. It is again lowered into the pipe and the operation repeated. The contents are washed and the estimated weight and the characteristics of the gold are noted. These operations are repeated, lengths of pipe and drill rod being added as required, until bedrock is reached. The pipe is then pulled from the hole by leverage, and the apparatus is moved to another spot, and another hole is started. The drill acts like a combined core drill and churn drill, for the toothed shoe cuts a core and the drilling and pumping tool performs the work of a churn drill as well as of a pump.

Calculations are based on the same plan as for the power drill, except that the diameter of the pipe of the Empire drill is smaller than that of the power drill. The material handled with the power drill is nearly two and one-half times the amount handled with the Empire drill; that is, the dirt recovered during 238 feet of drilling with an Empire drill would represent 1 cubic yard, whereas the dirt from 100 feet of drilling with a Keystone drill would approximate this amount.

Under favorable conditions the speed of the Empire drill exceeds that of the Keystone, and its ease of transportation is at times an important feature (see Pl. XII). Where labor is cheap, especially in some foreign countries, the cost of drilling with a hand machine is much less than that with a power drill.

Hand drills of the percussion type proved expensive and impracticable when used in Alaska. In examining shallow deposits, 6 to 10 feet deep, it is at times possible to improvise, at small expense, a hand-drill equipment that will give a rough indication of the gold content of the ground. Thus, a piece of casing may be driven directly to bedrock, then pulled, and the core may be washed. Needless to say, the sample obtained has not the value of a sample obtained with either of the two drills mentioned above.

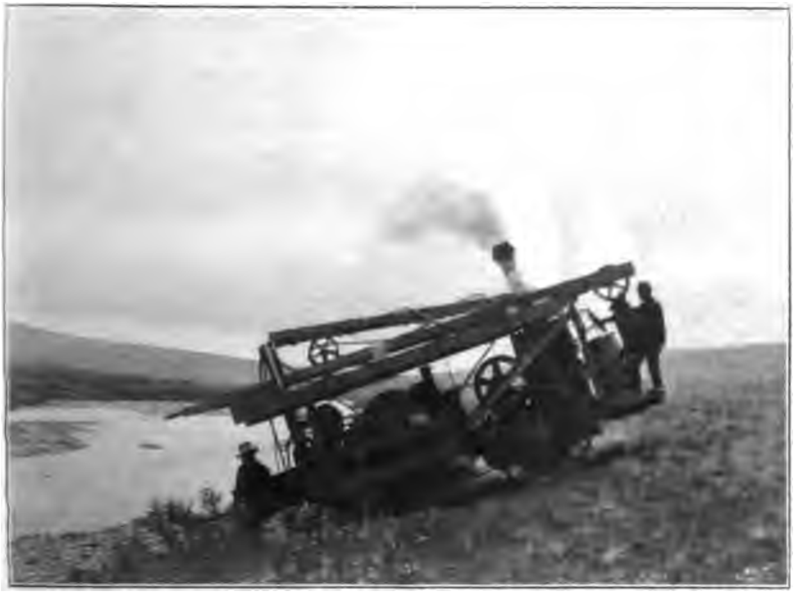
NEW TYPE OF LIGHT POWER DRILL.

A type of light power drill (Pl. XIII) has recently been put on the market and is favorably regarded by some operators and engineers. It is a drill of the percussion type, somewhat similar to but smaller than the Keystone drill, and is operated by a small gasoline engine. It is light, and when erected on a property is easily moved to a new location.

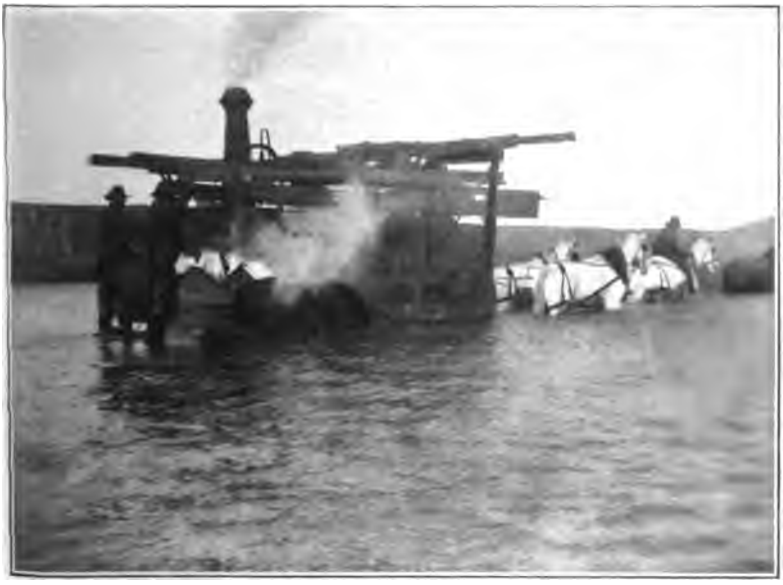
DETERMINING PERIOD OF DREDGING PROJECT.

The life of a dredging property can be estimated much more closely than can that of a vein mine. Also, the ore in sight in vein mining is figured as a guaranty for the return of only a part of the capital invested, whereas in gold dredging allowance must be made for recovery of the cost of the property and the equipment during the life in sight. The dredge equipment is usually of sufficient capacity to turn over the ground in 7 to 10 years, as that period is generally conceded to represent the life of a dredge with a wooden hull, although a few such hulls have lasted more than 13 years and are still in use.

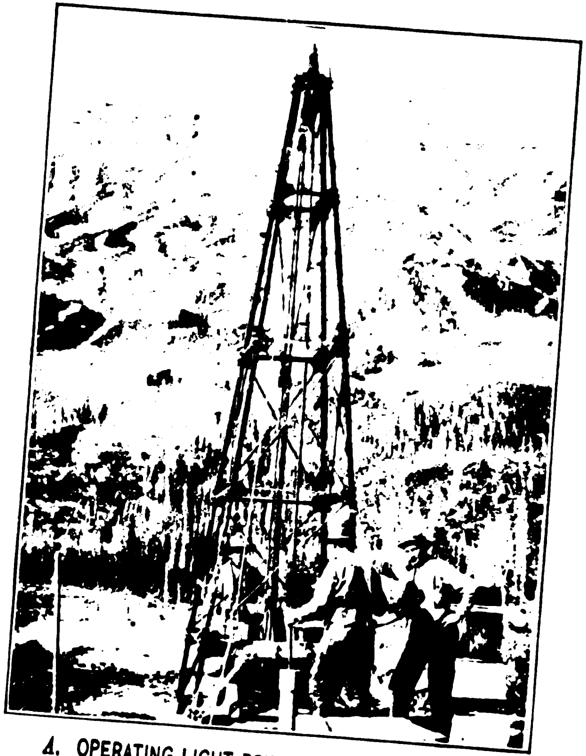
Although the rate of interest desired on the investment varies with the ideas of different operators, it is generally conceded that after allowing for amortization of capital the minimum profit to be figured for a dredging project is 10 per cent per annum.



A. KEYSTONE TRACTION DRILL DESCENDING A RIVER BANK.



B. KEYSTONE NONTRACTION DRILL FORDING A RIVER.



A. OPERATING LIGHT POWER PROSPECTING DRILL.

B. RECENT TYPE OF LIGHT POWER DRILL BEING MOVED.

From the working results of different size dredges over a considerable period, as shown in the section on working costs, the time required to work a given area of ground by different size dredges and the working costs under average conditions in California can be closely approximated. The following example will indicate how these figures can be applied.

An acre contains 4,840 square yards, so that an acre 11 yards deep would contain 53,240 cubic yards, which, with allowance for ground lost in turning and for uneven bedrock, can be figured at 50,000 cubic yards. If an examination of an area on which dredging is projected indicates 500 acres of workable ground 11 yards deep, the contents in cubic yards would be estimated as 25,000,000.

Under favorable conditions a boat having a capacity of 15 cubic feet would work out about 500 acres 11 yards deep, or 25,000,000 cubic yards of gravel of moderate compactness, in about 8 years, allowing 80 per cent working time, at a total operating cost of 4 to 5 cents per cubic yard. To work out this same area, two 7-cubic-foot boats would require 10 years, at a cost of 5 to 7 cents per cubic yard, and three 5-cubic-foot boats would require 11 years, at a cost of 5½ to 8½ cents per cubic yard. The cost given per cubic yard includes the total working and administration charges.

SELECTION OF DREDGE FOR A GIVEN PROJECT.

Decision as to the most desirable size of boat for a projected dredging operation sometimes requires considerable study, and here again is where the judgment of the experienced engineer is valuable. It is safe to say that the experience of the California operators is such that under certain conditions the large 15-foot boats would not be recommended as the most economical. Where compact and cemented gravel predominate, probably a 9-foot or 10-foot boat would now be selected for such work, partly on account of lower first cost, but more especially on account of expenses and loss of time for repairs with the large boats, and the difficulty of handling the heavy parts.

No hard and fast rule can be prescribed for determining the best size of dredge, and because of the development of the gold dredge to its present efficiency, the question is often raised as to when the maximum size for economic installation will be reached. All depends upon operating conditions. Under the favorable conditions at Idaho City, the 18-foot bucket of the No. 4 dredge of the Boston & Idaho Co. demonstrates the economic efficiency of the large dredge. There is no question as to the mechanical possibility of buckets even larger than those now in use, as is discussed under "Reclamation of dredged land," page 205.

In addition to the gold to be recovered from the gravel, the probable value of land reclaimed for agriculture might, in some regions, be considered in estimating the total profit.

Any estimate of the working cost of a dredge of this character is, of course, pure speculation, but there seems every reason to expect that, under favorable conditions, or in redredging some previously dredged areas, a very low operating cost would be obtained.

USE OF UNUSUAL TYPE OF DRILL IN ALASKA.

Contract drilling for a placer examination is not usual, nor under most circumstances is it wise. The following description of an unusual kind of drilling machine employed on contract during an examination on the Seward peninsula in 1912 is of interest. A steam drill on the property was found unsuitable, partly because of the soft nature of the tundra over which it had to be hauled, and partly because the machine was an inferior type and in poor repair. Hand drills of the percussion type were tried but were not successful. The cost of drilling, on account of frozen ground and of high wages ($7 a man daily) was excessive, as only a few feet a day could be drilled. The drill procured for the work was made by a contractor who had had considerable experience with drilling in frozen ground. It was in type and action similar to a Keystone drill. Power was furnished by a gasoline engine of about 25 horsepower. The width of tread of the machine was 16 feet 6 inches; the drive wheels were 6 feet wide, being designed for the soft tundra. The drill could be driven under its own power over the tundra where in places even walking was difficult and also over tailing piles and uneven ground. The drill stem was made of two ordinary Keystone stems, and a heavy 6½-inch bit was used; the weight of bit and stem was estimated at 1,850 pounds. In order to obtain the services of the contractor, it was necessary to enter into an agreement for the drilling. Under different circumstances, considerable risk might be assumed in drilling placer ground by contract, but care was taken to see that the terms of the contract were such that the operation was entirely under the direction of the engineer, and moreover the contractor's references for ability and reliability were excellent. A minimum of linear drilling distance of 2,500 feet was guaranteed, with the right to drill up to 5,000 feet additional if desired.

The contractor furnished the drilling machine and all supplies for it and agreed to take personal charge of the drilling and to furnish helpers. He also supplied board for his men and made repairs. The engineer furnished panners and rocker men (two to each shift). In addition either the engineer or an assistant was in charge of the work during the entire time. The terms of contract were 80 cents a foot for frozen ground and $1.50 for thawed ground. Casing was necessary in thawed ground, but not in frozen ground except for one length of pipe at the commencement of each hole. There was approximately 2,500 feet of drilling done from June 28 to July 29. One shift was employed at first until the drill, which had been out of commission for some time, had been put in good running order. The second shift commenced about July 3. The shifts were of 12 hours each. Wages paid for panners were 50 cents an hour and board.

The rate of drilling, averaging a little more than 40 feet a shift, was far less than the contractor usually obtained. He stated that he generally averaged 100 feet or more a shift. The lowered rate was caused by the greater care insisted upon in drilling, by the time required for pumping every 2 feet in the upper part of drill holes, and every foot when colors were encountered as compared with every 5 feet in all ground, the usual requirement there. A greater number of pumpings for each section were also taken than was usual at Nome. To test the reliability of the pump, several customary tests were made, such as pouring a certain number of small shot in the hole, and obtaining the same number after pumping.

The action of the drill was similar to that of the Keystone. Pumpings were dumped into a rectangular zinc box and from there washed with fresh water to pans held over galvanized tubs; all dirt recovered from each section was panned separately and the number of colors counted; afterwards all tailing from the panning was rocked, and any gold recovered from rocking was added to that from panning. The rocker tailings were frequently sampled to ascertain whether there was any loss of gold. The volume of dirt from each hole was measured by digging a hole 3 feet square and 1½ feet deep, at the end of the rocker. The tailing from the rocker was run into this hole and allowed to settle and the cubic contents afterwards determined. From 3 to 5 pumpings were made from each section. Sections were taken every foot in "lower" ground, and every 2 feet in "upper" ground or overburden. The bit was measured before and after each hole was drilled and an average was taken in calculating the volume taken out. The figure so obtained was checked by measuring the dirt from the hole, and a proper factor for calculations was ascertained. It was impossible to get the best results with one panner; hence two were used on each shift and were generally kept busy.

It was not easy to arrive at the total cost of sampling on account of difficulty in properly proportioning the time of the men. Actual charges against the drill, engineering fees, etc., being omitted, amounted to about $1.20 a foot.

DETAILS OF CONSTRUCTION OF GOLD DREDGES.

DETERMINING SUITABLE SIZE OF DREDGE.

After the ground has been prospected, the selection of a dredge best suited to the work is the next important step, provided the results of the examination have been favorable. Some costly errors have been made; in fact, in some projects the selection of the wrong type or size of dredge has meant failure instead of success. In general it may be said that if the ground is deep, the gravel not cemented, and the property large, then the larger the dredge the lower will be the cost of operation. On the other hand, if the ground is shallow, say, 30 feet or less, a large dredge is undesirable, as the hull for a large dredge would have to be 10 or 12 feet deep and its flotation would make trouble. For the same reason, if the ground is shallow, the grade of the bedrock is also a factor in determining the size of the dredge. If the ground is tightly cemented the buckets should not be of the largest size but should be small and strongly built. Where bowlders are present the large buckets make for ease of handling.

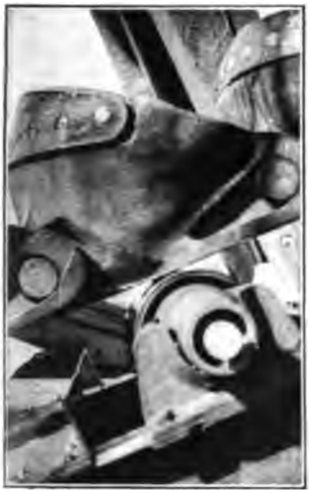
When the ground can be dug without undue difficulties, the production with different sizes of buckets should be about as follows:

Probable production with various sizes of buckets in fairly loose ground.

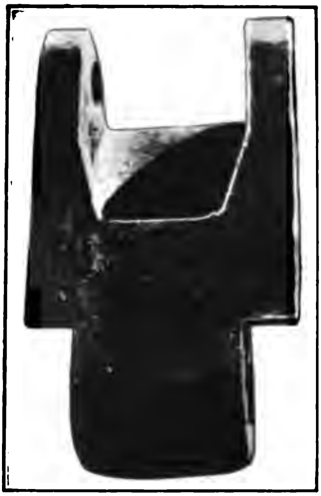
Capacity of buckets, cubic feet.	Monthly production, cubic yards.
3	50,000
5	85,000
7	120,000
9	180,000
16	300,000

BUCKET LINE AND TUMBLERS.

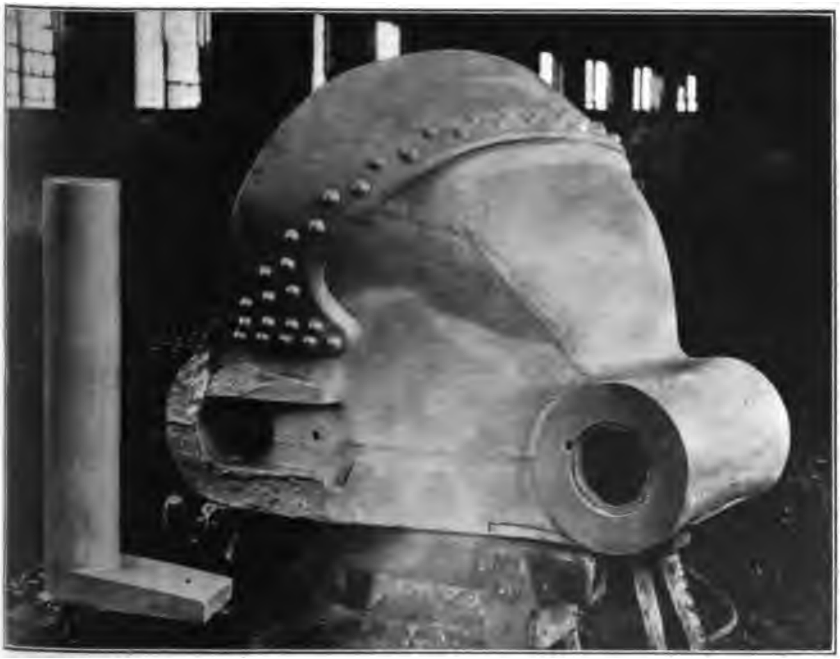
The bucket line and tumblers still present the greatest difficulties to the designer and operator, and the fact that on practically every new dredge experiments are made with new types is evidence that there is still a wide difference of opinion as to the best design. It is conservative to say that the bucket lines of more than half of the dredges going into new fields are failures, or at least entirely unsuited to their work. And when it is realized that the bucket line of some large dredges costs $50,000 or more installed, the seriousness of such failures becomes evident. Many failures can be traced to the design of the equipment which has not been suited to the ground to be worked.



A. LOW-BACK BUCKET FOR DIGGING BOWLDERS; ALSO SHOWING LADDER ROLLER.



B. BASE OF NEW TYPE BUCKET FOR USE WITH ROUND LOWER TUMBLER.



C. MANGANESE STEEL BUCKET OF 16 CUBIC FEET CAPACITY, CONREY NO. 4 DREDGE; SHOWS FULL CIRCULAR BUSHING IN BACK EYE.

The character and the depth of the ground to be dug as well as the size of the gravel determine the shape and weight of the bucket. A bucket well designed for working in small gravel would be unsuited for bowlders and vice versa. When much clay is present still another shape would be more suitable. A bucket to handle bowlders (Pl. XIV, *A*) should be designed with a low back so that when bowlders are picked up and the bucket carrying the rock comes around the lower tumbler to the horizontal position the bowlder will not crush the back of the preceding bucket. Plate XV shows a dimensioned drawing of a 16-foot bucket.

The buckets should not be filled so full that excessive spilling will occur. Some ground is of such a character that the bank stands firm and it is necessary to dig to the top; in such digging the bucket line is almost in a horizontal position, and all gravel in excess of the natural repose of the material will wash out, part of it being lost over the side of the ladder, and the remainder being returned by the ladder pan to the front of the ladder and redug, resulting in a loss of efficiency. Hence the lip of the bucket should not be much higher than determined by the natural angle of repose of material.

In many fields the top of the bank caves, so that the ladder of the dredge is always kept at a considerable depth. In such digging the lip of the bucket can be higher than when the bucket line is horizontal. The bucket should be as high as the factors mentioned above will permit, as of course the higher the bucket the greater the capacity.

The present practice is for the buckets to have two front eyes and one rear eye (Pl. XVI, *A*). The so-called "three-eye buckets" with three eyes in front and two behind have been discontinued, as the lugs broke readily and the pins were taken out with difficulty. They were more expensive to manufacture and it was necessary to design the inside of the buckets with pockets which interfered with the dumping of the material. The idea underlying the design of these buckets was that owing to the three supports the pins would not break so readily.

The width of the bottom of the bucket should vary directly with the length of the bucket line to be used, and also with the hardness of the material to be dug (Pl. XIV, *B*). It is also necessary to have sufficient bearing area so there will not be excessive wear where the bucket seats on the upper tumbler. In order to increase this bearing surface the "square bottom" bucket is now being largely used. This type has several other advantages. The wear on the bucket where it bears on the lower tumbler is reduced, as is also the wear on the ladder rollers. The wear on the flanges of the lower tumbler and the side plates on the upper tumbler is negligible, whereas with the old buckets considerable trouble was experienced at these points. Plate XVI, *B*, shows the eye of a bucket elongated by wear.

If the material to be dredged is clayey the bucket should be wide and short to facilitate dumping. The buckets on the early dredges were long and narrow and when any sticky material was encountered it would not be discharged. This trouble was increased on headline dredges in early days, as the clay was dug in a mass. The present practice in handling clay is to swing the dredge by means of spuds and to dig lightly so that the material enters the bucket in comparatively thin slices.

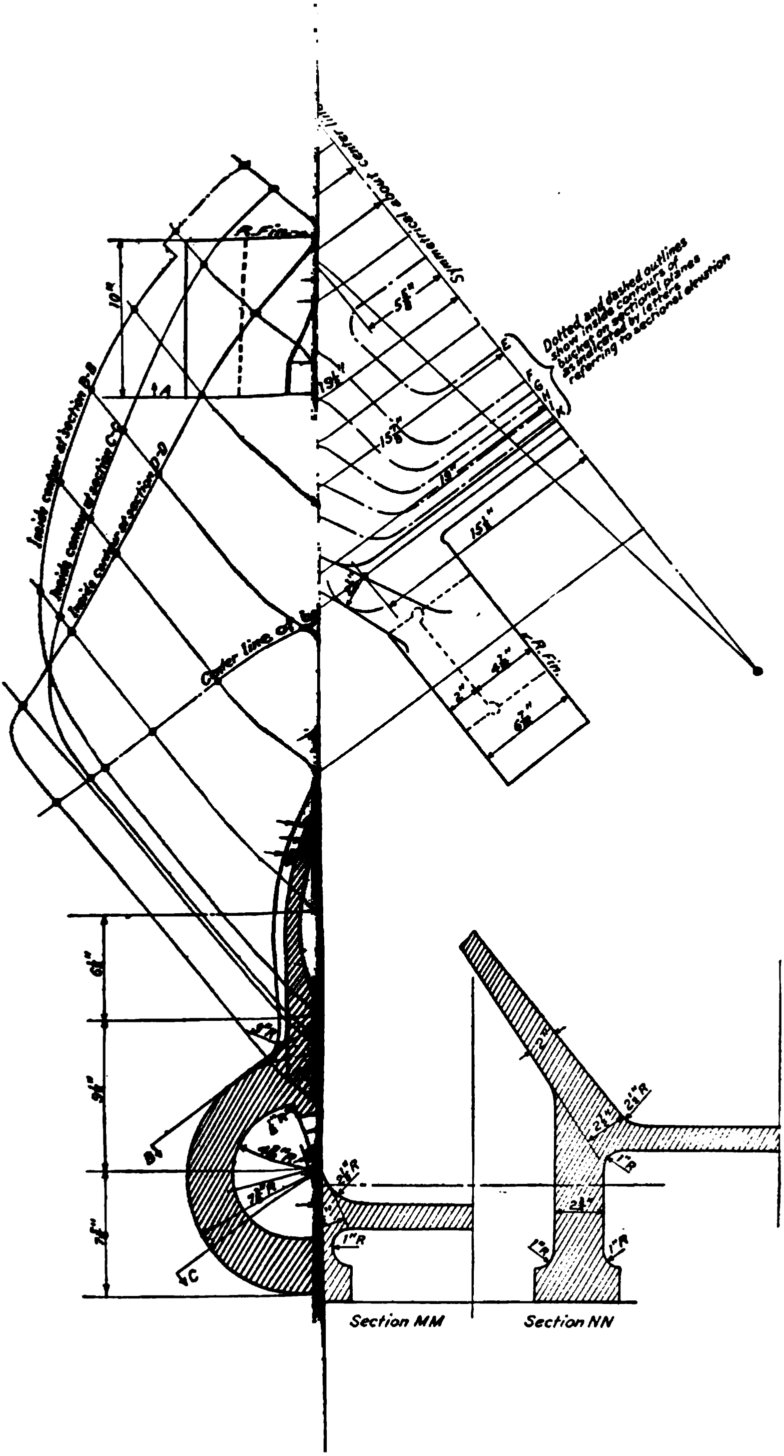
The original California type of bucket consisted of three parts—the bottom, the hood, and the lip (Pl. XVII). The bottom was always cast, either of special carbon, manganese, or nickel-chrome steel. The hood was pressed or cast steel, and the lip was manganese steel. This design necessitated a large number of rivets which frequently became loose and caused trouble. During the past 10 years the art of making steel castings has been so developed that it is possible to cast the hood and bottom in one piece (Pl. XVIII, *A*), which does away with leakage and is cheaper in first cost. With this type it is more difficult to cast the lips to make an accurate fit on the cast hood, but experience is remedying this drawback. It is important to have the lip fit snugly to the hood, so that the lip will not have to be drawn in with bolts before being riveted, as the spring of the lip will put an initial strain on the rivets that will ultimately cause them to become loose. It is generally better to have a few large rivets than a greater number of small ones. On 16-cubic-foot buckets 1¼-inch rivets are not too large and 7-foot buckets should have 1⅛-inch rivets. Tendency of the rivets to shear should be obviated by having the lip butt against a lug on the bucket.

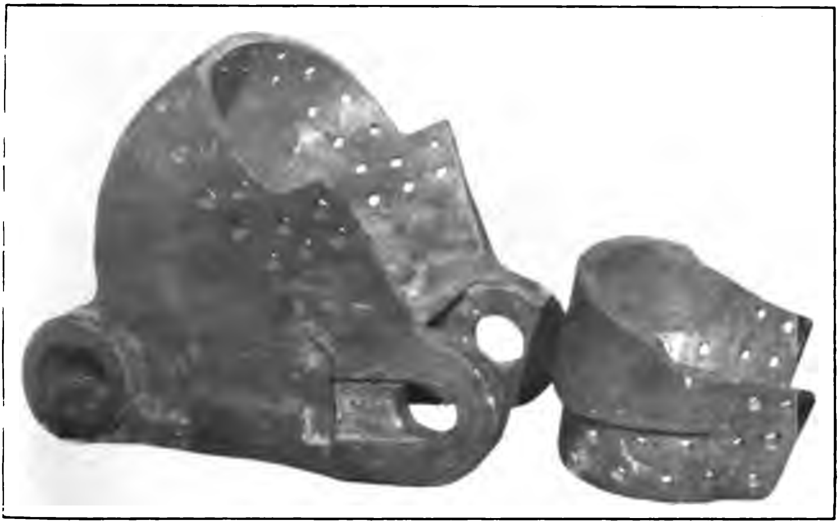
A common error in bucket design is making the arch of the lip too flat. This fault frequently leads to crushing of the lip.

When the hood and the bottom are cast together they are usually made of manganese or nickel-chrome steel. When made of nickel-chrome steel a manganese-steel insert plate (Pl. XVI, *C*), is peened into the bottom of the bucket where the wear comes on the upper tumbler. Care should be taken that this insert is set ahead far enough so that the corner in the recess that takes the insert plate will not come near the corner of the bushing seat as a crack is likely to occur between these points. A large proportion of buckets that fail break at the back eye whether they have insert plates or not, but the plate is an additional source of trouble. Plate XVIII, *B*, shows a 2-cubic yard bucket used on a Panama Canal dredge, and Plate XIX, *A*, shows a 6-foot bucket used on the Guamos dredge in the Philippine Islands.

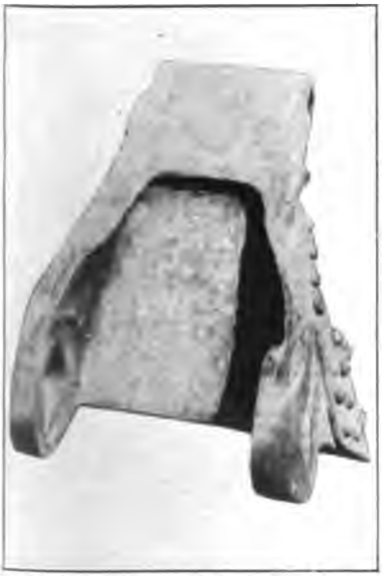
PINS.

The diameter of the bucket pins should not be limited to the size necessary to withstand calculated breaking strains, for wear of the pins has been found to be the determining factor. It is true that

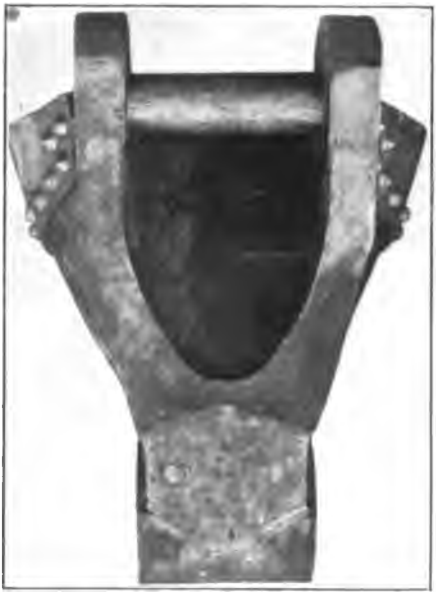

Symmetrical about center line
R.Fin.
10"
A
19¼"
5⅝"
15⅝"
18"
15¼"
E
F
G
H
I
K
Dotted and dashed outlines show inside contours of bucket on sectional planes as indicated by letters referring to sectional elevation
R.Fin.
2"
4½"
6⅞"
Inside contour at section B-B
Inside contour at section C-C
Inside contour at section D-D
Center line of bucket
6½"
9¼"
7½"
B
C
1"R
2½"R
2¼"
Section MM
Section NN



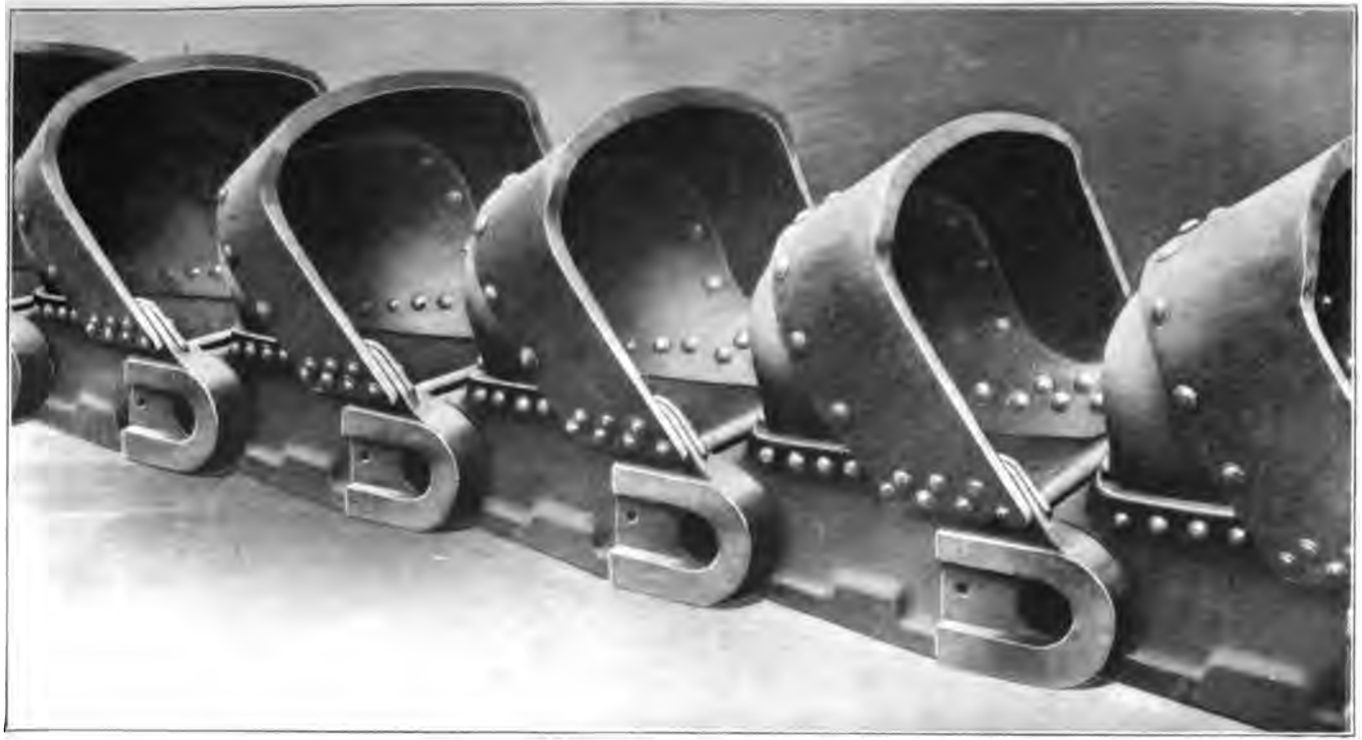
A. ONE-PIECE MANGANESE-STEEL BUCKET WITH SPECIAL TYPE OF LIP.



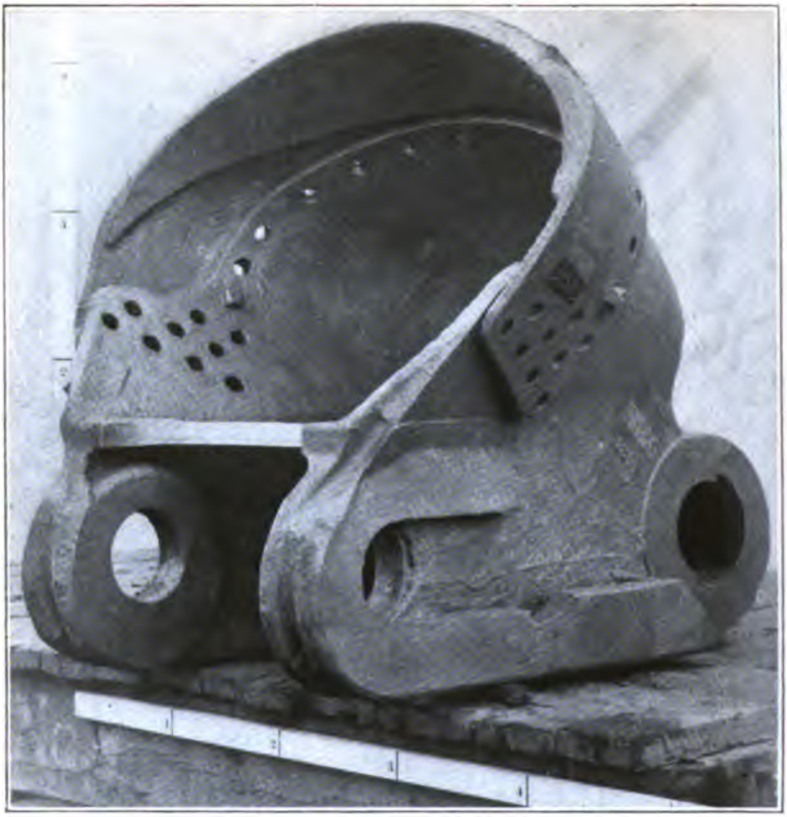
B. BUCKET WITH EYES ELONGATED BY WEAR.



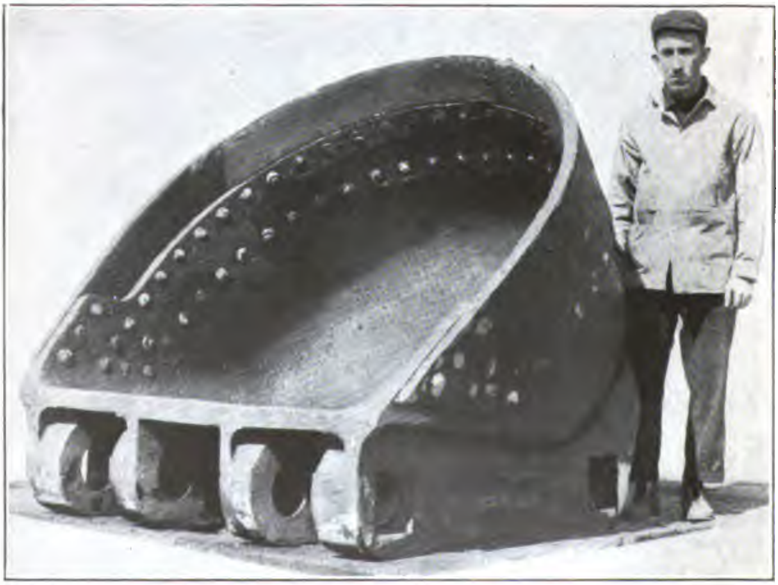
C. BUCKET WITH INSERTED PLATE OF MANGANESE STEEL TO RESIST WEAR ON TUMBLER.



ORIGINAL CALIFORNIA TYPE OF BUCKETS.



A. RECENT TYPE OF 16-FOOT BUCKET WITH BOTTOM AND HOOD CAST AS ONE PIECE.



B. BUCKET USED ON A PANAMA CANAL DREDGE; CAPACITY, 54 CUBIC FEET.

pins often break in service, but most breakage is due to inefficient heat treatment and not to small diameter. The chief cause of bucket-line trouble is the elongation of the pitch of the buckets and the decrease of the effective pitch of the tumblers owing to wear. Not only does a pin wear but it wears the bucket eyes and the bushings. Hence the larger the pin the longer will be the life of the bucket. In the standard type of bucket the front eyes are not designed with the idea of having the pins turn in them but the pins are held against turning by lugs. A bushing is placed in the back eye to take the wear when the bucket turns on the pin. The half bushing is in most general use but some circular bushings are employed (see Pl. XIV, *C*, p. 54).

The lugs on the pins should be long and thick and fit snugly in the lug seats; otherwise the play resulting from looseness will soon cause the front eye of the bucket to wear rapidly. Pins are usually made of forged and tempered nickel-chrome steel, although manganese pins, machine ground, are being used with success. Several methods have been used to hold the pin in position but lugs only have proved successful.

Another type of bucket used on two or three dredges in California is designed so that the two front eyes have a combined length equal to the back eye. The pin is a straight piece of shafting free to turn in either the back or the front eye. Both the front eyes and the back eyes of this bucket have circular bushings.

LOWER TUMBLER.

The round lower tumbler (Pls. XIX, *B*, and XX, *A*) is the greatest improvement made in gold dredges in recent years, and as it is or will be universally used none of the other various types is here discussed. Most of the round tumblers at present are of manganese steel made in one piece. Some manufacturers make them of chrome-nickel steel owing to difficulty of producing a sound casting of this size and shape in manganese. Other companies make the tumbler in halves of chrome-nickel steel and use manganese wearing plates. The pressure used in setting in the shafts is about 125 to 175 tons, and it is not necessary to key the shaft in a round tumbler.

The following advantages apply to the round tumbler. With the hexagonal type (Pl. XIX, *C*), the difference in pitch of the bucket and tumbler is noticeable a short time after the bucket line is started and continues to increase until there is a difference of several inches. As the buckets ride around the lower tumbler, the slack caused by the difference in pitch accumulates for three or four buckets until the tension in the bucket line becomes great enough to cause the buckets to slide on the tumbler, and this slipping causes the wear on the tread and flanges of the tumbler. With a round tumbler such slipping is entirely eliminated. When the first experiments were made with the round tumbler it was expected to prove harder on the buckets and to cause them to break through the middle, as the buckets used in the experiments were worn and not designed especially for a round tumbler. It was also thought that a rounded surface would be worn on the bottom of the bucket where it had its seat on the tumbler, but such wear did not occur.

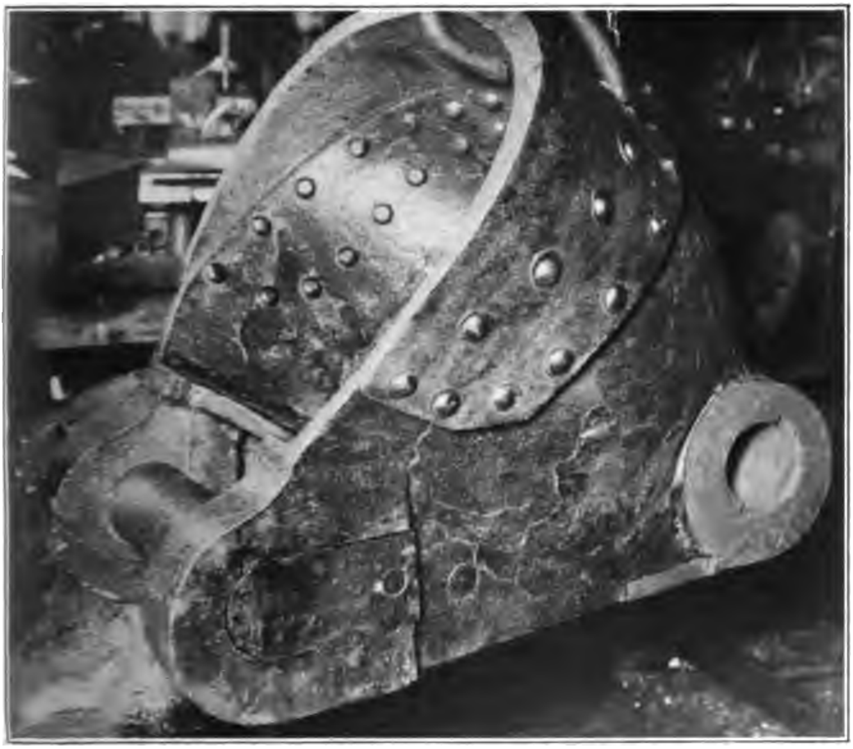
Round tumblers should be of such diameter as to obviate undue wear by the buckets. It is generally the custom to make the pitch of the tumbler a little greater than six times the pitch of the bucket. Thus, each successive bucket seats at a place different from that at which any preceding bucket has seated, causing the wear to be equally distributed all around the tumbler. Clearly, the efficiency of the round tumbler is not impaired by the elongation of the bucket pitch.

Tumbler shafts should be made enough larger than the bore of the tumblers to insure a tight fit. The oversize should vary from 1/1000 of an inch for each inch of diameter for shafts up to 6 inches in diameter to 1/2000 per inch of diameter for shafts 15 inches in diameter.

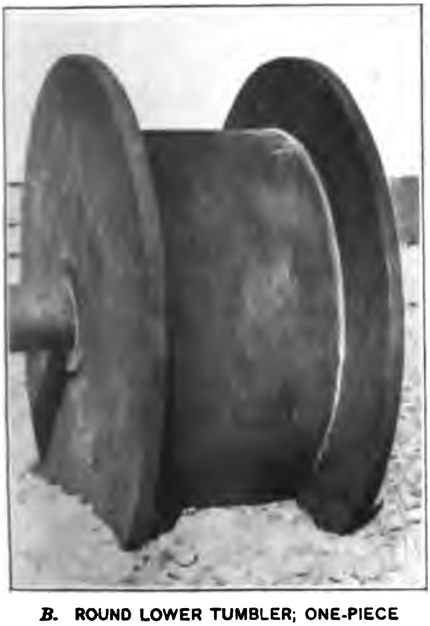
The shaft should be bored so that rods can be placed to hold the ladder from spreading. One method is to forge thrust collars on the end of the rod. If the ladder should spread when these are taken out the rods can be heated and put in place hot and the shrinkage will bring the ladder to the original position. Sometimes the tumbler shaft is turned so that collars are formed on the ends, and several other methods are used to keep the ladder from spreading (see Pl. XX, *A*, Pl. XX, *B*, and Pl. XXI).

There is still a difference of opinion regarding the most desirable type of lower tumbler bearing. Cast-iron bushings are generally used, but some companies prefer babbitt. Babbitt bearings do not wear so long, but they do not break as the cast bushings sometimes do, causing much damage to the shaft. However, opinion seems general that cast iron is preferable as the shaft wears more evenly. When a new bushing is to be installed the shaft is measured and the new bushing bored to suit the worn shaft. It is good practice to shrink nickel-chrome wearing sleeves on the shaft, as they wear at about the same rate as the bushings, and the sleeves and bushings can be renewed together.

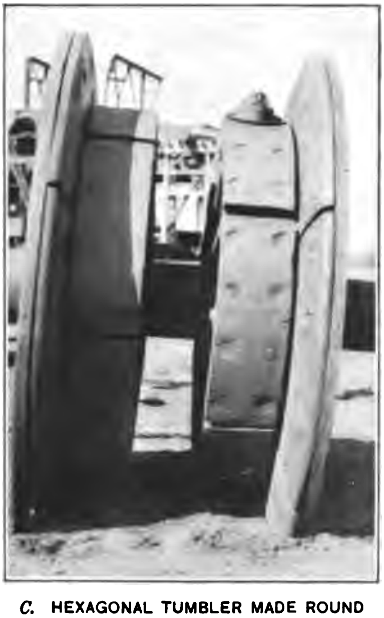
Another type of lower-tumbler bearings worked out in the Ruby district has proved successful and is in general use there. The main steel bearing for the lower-tumbler shaft is fastened to the end of the ladder with two large bolts. The bearing is lined with a circular iron bushing to take the wear. On the outside end of the bearing a large steel cap is bolted with packing between the cap and the bearing; on the inside next to the tumbler a steel ring with



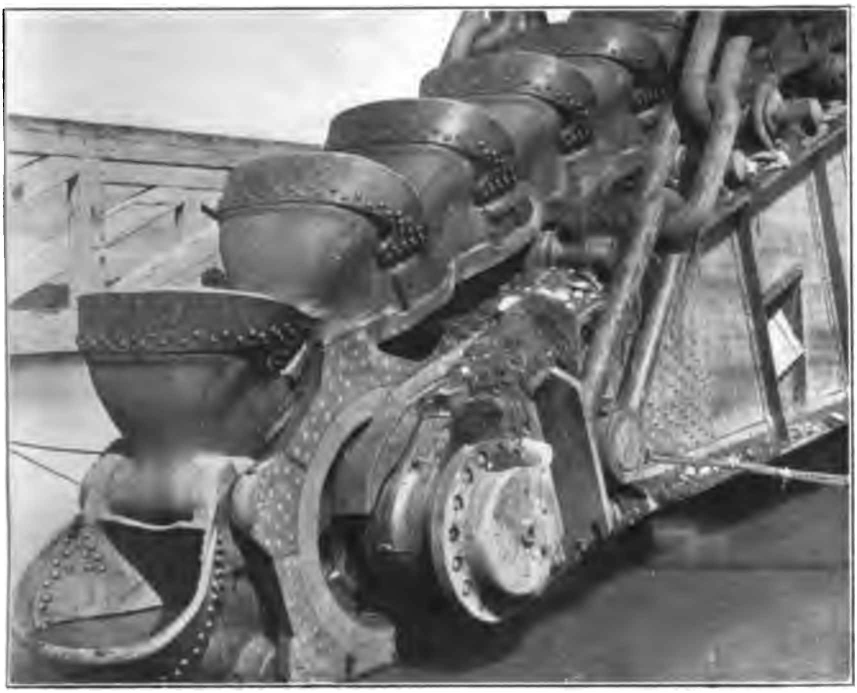
A. SIX-FOOT BUCKET OF GUAMOS DREDGE.



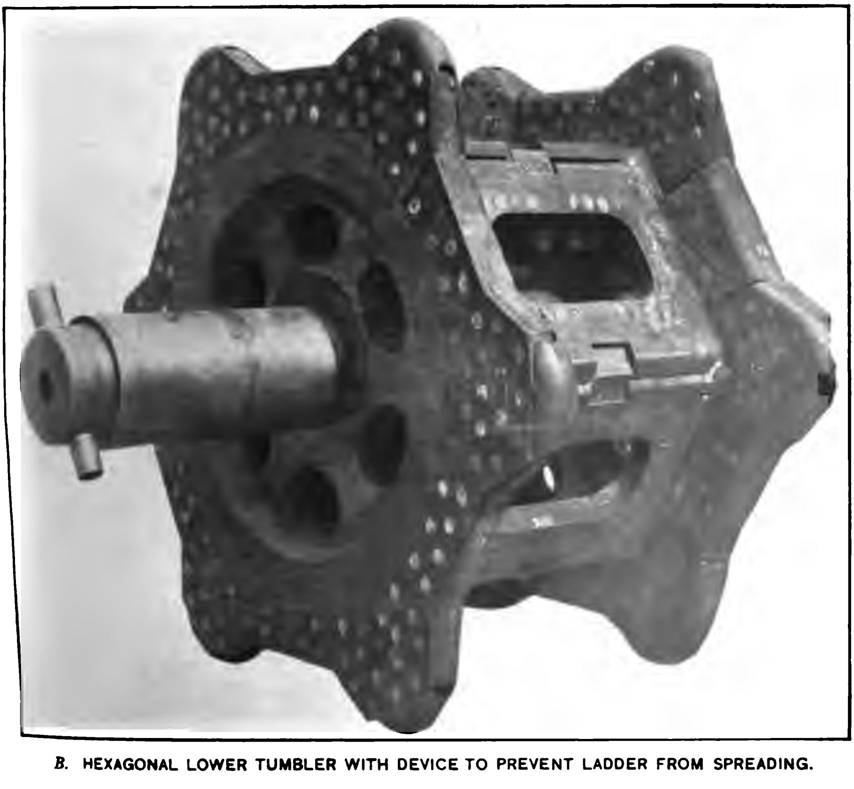
B. ROUND LOWER TUMBLER; ONE-PIECE PATTERN; MANGANESE STEEL.



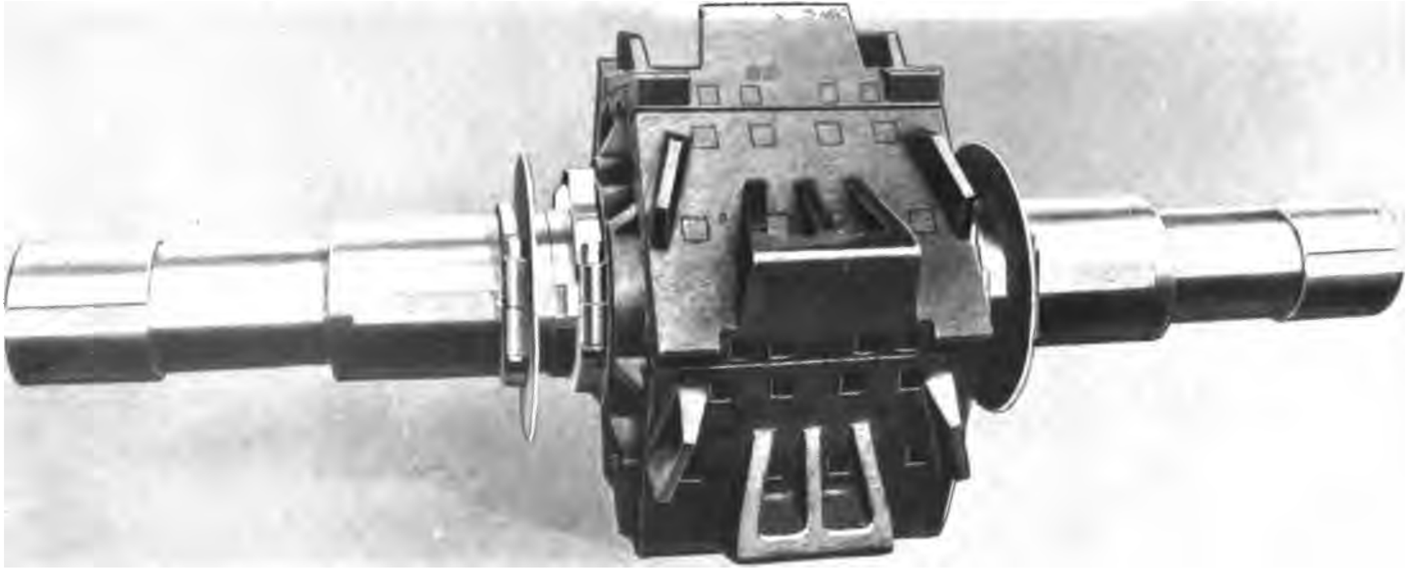
C. HEXAGONAL TUMBLER MADE ROUND BY ADDING CAST-STEEL SEGMENTS.



A. BEARING OF LOWER TUMBLER ON DREDGE AT RUBY, MONT.



B. HEXAGONAL LOWER TUMBLER WITH DEVICE TO PREVENT LADDER FROM SPREADING.



CUSHION-PLATE TYPE OF UPPER TUMBLER.

the smaller diameter a trifle larger than the shaft is held by stud bolts in the end of the bearing casting. The smaller diameter of the ring is beveled to take square flax packing, which is compressed against the cast-iron bushing in the bearing and the shaft by tightening up the stud bolts. The cap on the outside of the bearing is large enough to allow a large steel collar, pinned on the end of the tumbler shaft, to turn. This collar has a bearing against the outer end face of the bearing to take the side thrust in digging. Grease is forced into the space surrounding the thrust collar inside of the cap to give perfect lubrication to all wearing parts. The ladder roller bearings are essentially the same as those for the lower tumbler shaft. Each bearing consists of two castings: First, a base bolted to the top of the ladder; second, the bearing proper fastened to the base by means of a horizontal pin to give vertical play to the bearing if the roller gets out of alignment. The bearing proper has only one end opening, through which the roller shaft projects, and after the shaft is in place a ring, similar to the one on the lower tumbler bearing carrying flax packing, is drawn up against the face of the bearing by stud bolts. The outer end of the bearing is part of the casting and has a brass plate fastened to the inside to take any end thrust from the roller. The bearing bushing is a half-circular brass casting set into a recess in the bearing casting to prevent turning.

UPPER TUMBLER.

As yet no type of upper tumbler has been designed that will take care of the increase in pitch of buckets. The most satisfactory device yet tried is to have easily removable wearing plates made of different thickness and to build up the tumbler as the pitch increases. Formerly the general practice was to use a plain hexagonal tumbler body and to have six cushion plates that covered the entire face of the tumbler. The cushion plates included the flanges or side lugs, as well as the heel. Later practice was to use an additional wearing plate on the surface of the cushion plate that did the driving (see Pl. XXI). This was called the heel plate and it is this plate that is made in a series of thicknesses. Where long heavy bucket lines are necessary, the heel plates under some conditions will wear out in three or four weeks and the cushion plate in about six months. The cushion-plate type of upper tumbler is being eliminated, as the upkeep is too high, tumblers are being designed to take only flat wearing plates.

It seems impossible to get away from wearing plates on upper tumblers, and the consequent shutdowns from loose bolts, etc., because of necessity the wear is confined to one area. In those fields where the service is not severe the problem is not so difficult. The wearing plate covering each face of the tumbler should be in one piece, for when a plate is in several pieces (Pl. XXII, *A*) the pieces work loose and the tumbler face is worn unevenly.

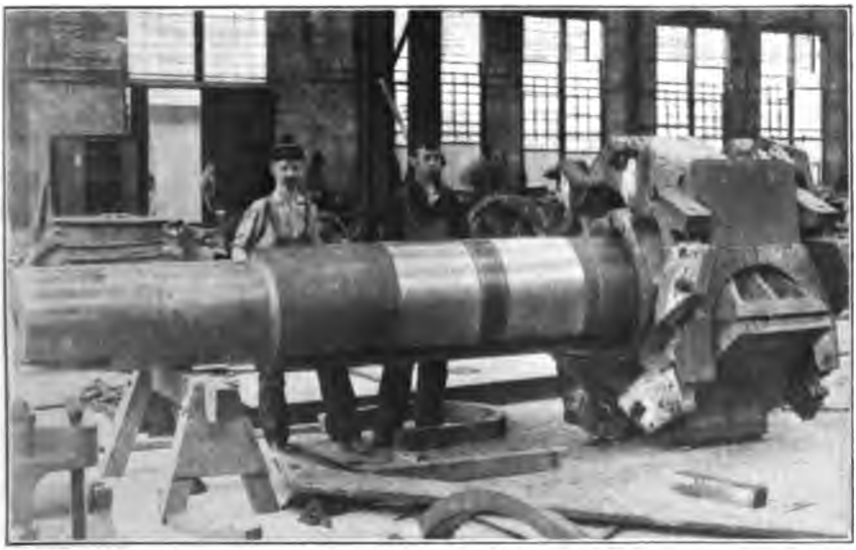
Several of the larger California operators have recently adopted the "one-piece" upper tumbler (Pl. XXII, *B*), which consists of the shaft, tumbler body, and cushion plates, cast in one piece of some special steel. Wearing plates are separate and are of large proportions so that the metal under them will retain a plane surface. The advantage of having the shaft cast with the tumbler is that greater strength is obtained with equal weight and trouble from loose shafts is eliminated. Shrinking the tumbler on the shaft and setting a ring around in a groove, as generally practiced, will hold the tumbler tight but weakens the shaft; hence the groove should not be cut as the shaft starts to crack at this groove. An added element of strength in the one-piece shaft and tumbler is that the ribs in the tumbler body can be extended out on the shaft where the bending moment is greatest and thus be made to strengthen the shaft where it is weakest. When a shaft is not cast with the tumbler, the tumbler should be heated and shrunk on and not pressed on. The relation between bore of tumbler and diameter of shaft should be the same as for the lower tumbler, as mentioned previously.

The increase in the number of sides of the upper tumbler has not kept pace with the increase in size of the dredges. The original dredges in this country followed the practice of those in New Zealand and used the 4-sided tumblers. The number was increased to 5 and then to 6 and has so remained for all sizes. For the 16-cubic-foot dredges there seems to be no reason why 8 sides would not be much superior and this type will no doubt be adopted in the future for dredges digging in ground that can be worked without undue difficulty. Plate XXIII, *A*, shows a method of transporting large tumblers.

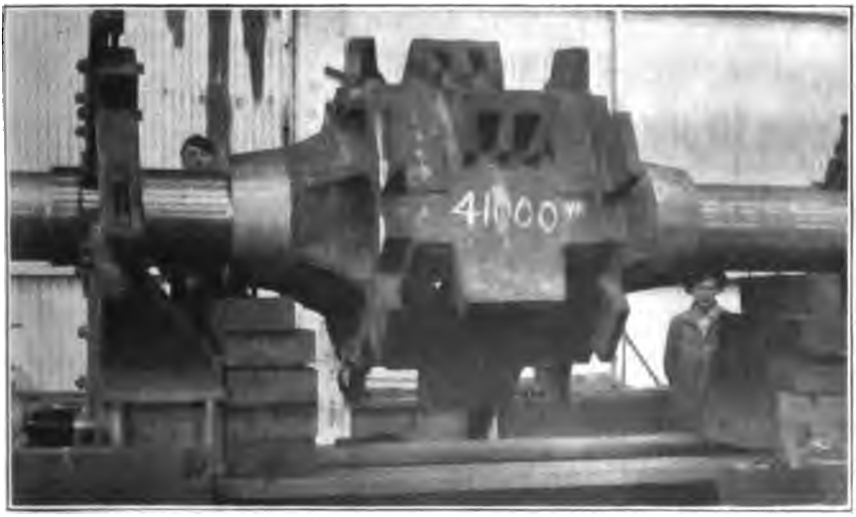
MAIN DRIVE.

The designing of gears (Pl. XXIV) to drive the bucket lines of the smaller dredges—with capacities up to 7 cubic feet—was relatively simple; the gears were made with a sufficiently large factor of safety, so that most gears wore as long as the dredges, but it was necessary to replace the pinions at intervals of two or three years. The "bull gears" were usually 10 to 12 feet in diameter, and the face about 8 to 10 inches wide, and the pitch about 2½ to 3 inches. The gears should be so proportioned that the bucket line will travel about 50 linear feet a minute.

During the year 1916 three 16-foot dredges were designed by different California companies, each one handling the problem in a different way. Two of the companies kept to the same general dimensions as before, the bull gears being 12 feet in diameter and the face 12 inches, but instead of using the ordinary carbon steel for

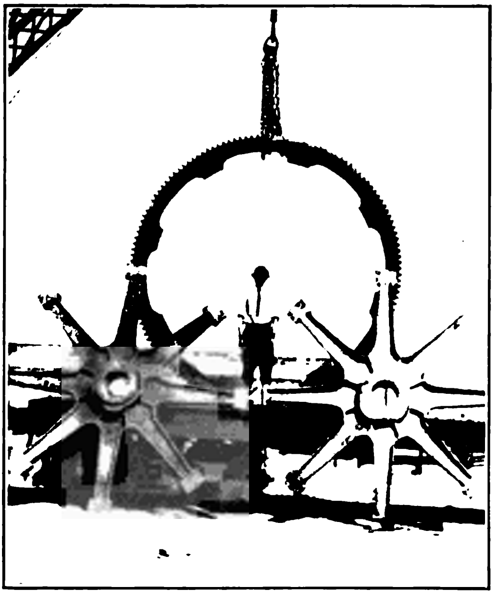


A. HEXAGONAL UPPER TUMBLER WITH THREE-PIECE WEARING PLATES, CONREY NO. 4 DREDGE. ILLUSTRATION SHOWS FITTING OF TUMBLER TO SHAFT.

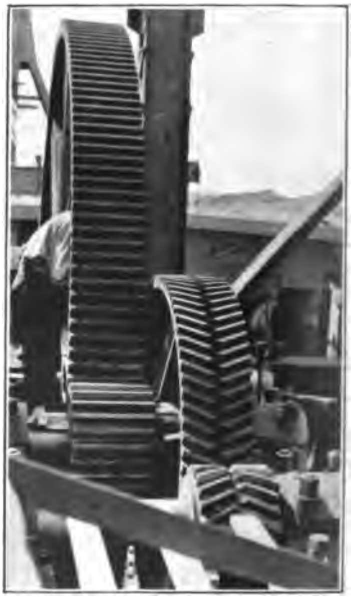


B. ONE-PIECE UPPER TUMBLER, YUBA NO. 14 DREDGE.

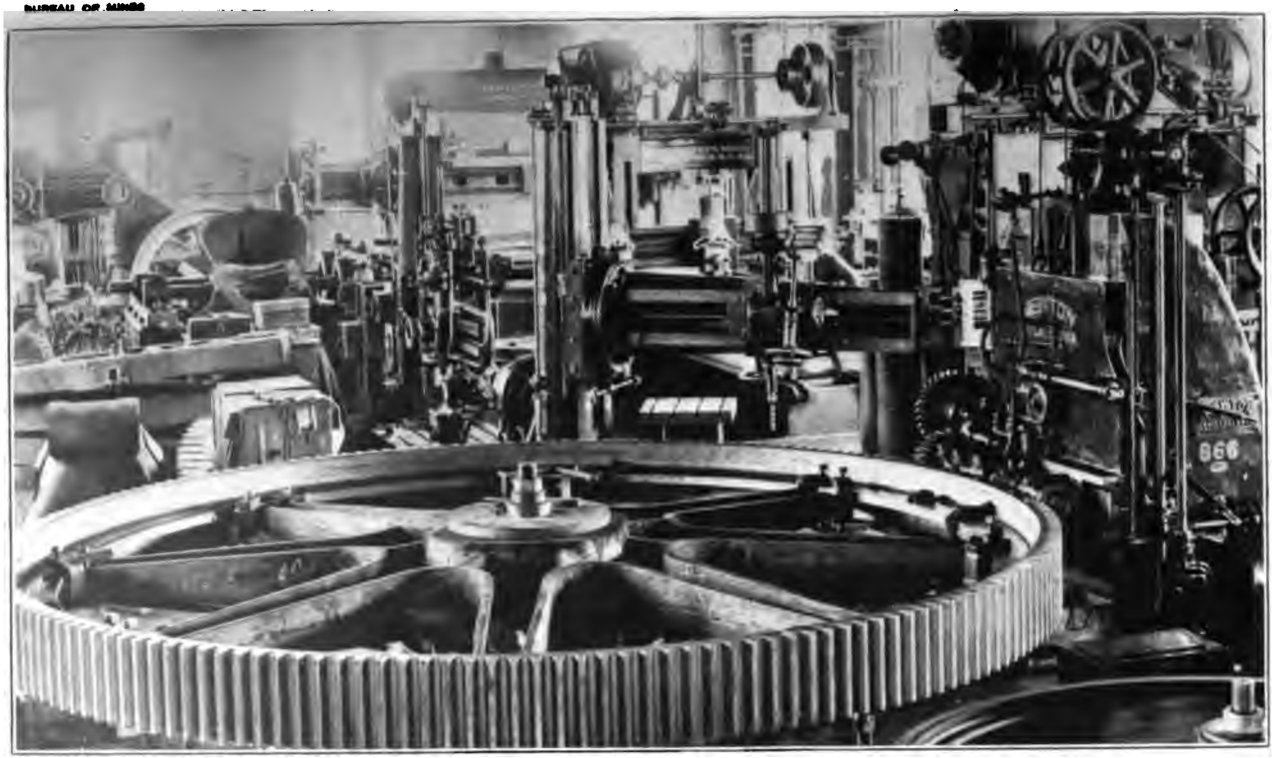
A. HAULING A LARGE TUMBLER.



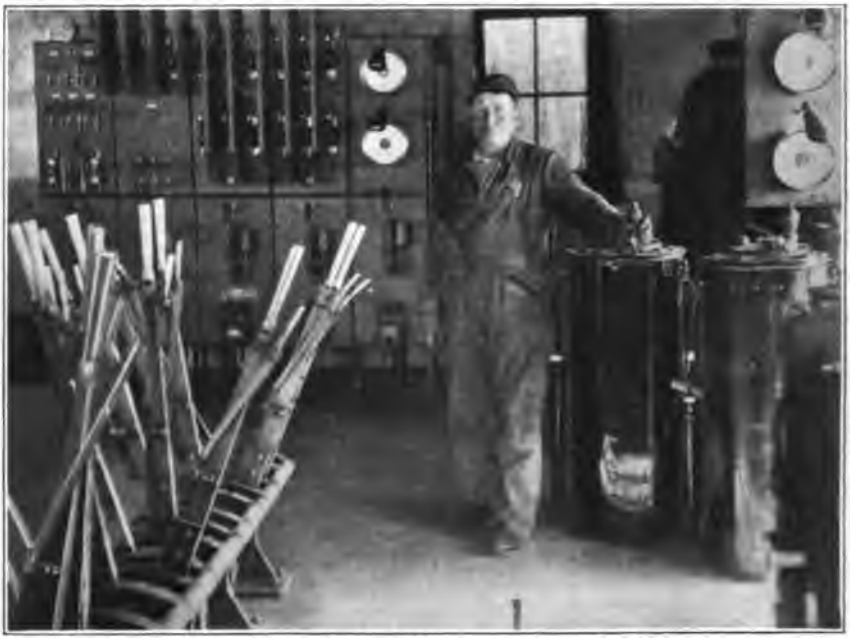
B. "BULL GEAR" READY FOR SHIPMENT.



C. INTERMEDIATE GEARS, HERRING-BONE TYPE.



MILLING THE TEETH OF A BULL WHEEL



A. WINCH ROOM ON CONREY NO. 4 DREDGE.

B. WINCH ROOM AND LEVERS ON YUBA NO. 4 DREDGE.

the bull-wheel rims, one company cast them of chrome-nickel and the other company made them of manganese steel. Use of these steels not only increases greatly the resistance of the teeth to wear but insures against breakage. A 12-foot gear is as large as can be shipped by freight (Pl. XXIII, *B*) if cast in one piece, a consideration that no doubt accounts for retention of that diameter. The third company increased the diameter to 14 feet with a wider face and cut teeth, but cast the wheel in halves which could be conveniently shipped. The material used was carbon steel.

The pinions are made of the same material as the gears they drive. Each of these three designs will no doubt solve the problem of rapid wear, but only actual service will determine which is best.

The intermediate gears for heavy service should be made of chrome nickel and have cut teeth. One of the most recent dredges is using cut gears of the herring-bone type (Pl. XXIII, *C*), which gives a more even running gear and does away with much of the noise. With this type it is essential that the shafts be kept in perfect alignment and that provision be made against end thrust. On dredges with wooden hulls these requirements can not be met, but should be easily met on dredges with rigidly built steel hulls.

On the early small-size dredges the tumbler was driven by one bull-wheel gear, but the torsional strain set up in the tumbler shaft caused the shaft to break. At present practically all dredges have a gear on each end with two trains of gears driving through the intermediate shaft from the pulley shaft. The most difficult problem connected with the main drive is to get an equal pressure on each of these sets of gears. It is necessary to make arrangements so that adjustments can be made to meet the more rapid wear of some of the gears. On most of the dredges this adjustment is made possible by having the pulley shaft in two parts connected with a coupling. A device consisting of an equalizer or differential mounted on the pulley shaft now takes care of this adjustment automatically.

Another comparatively recent improvement is to have two short shafts instead of having the intermediate shaft take both gears. An advantage of this arrangement is that the hopper may be made larger where it goes between the shafts; formerly it was limited to a size such that it would not interfere with the intermediate shaft.

As mentioned in the chapter on electrical equipment, opinions differ as to advisability of driving by a direct-connected motor or by a belt, but obviously it is essential to provide a drive of some kind that will slip when necessary, and a belt seems the simplest means of accomplishing this end.

WINCHES.

Winches are used to raise the ladder, drive the bucket line, swing the dredge, raise the spuds and stacker, and for repair work. Different combinations are used, but the most general arrangement is to use a winch called the ladder-hoist winch for hoisting the ladder and driving the buckets. The swing winch contains the bow and stern swing drums, the spud hoist, the spare drums for wrecking and repairing, and also the head-line drum, if one is used. The stacker hoist is usually placed separately at the rear of the dredge on the upper deck, as this arrangement does away with a number of sheaves. Levers used in operating winches are shown in Plate XXV. Plate XXVI shows a dimensioned drawing of the starboard winch operating lever for a 9-foot dredge, and Plate XXVII shows the general arrangement of winch equipment on a 9-foot dredge.

LADDER-HOIST WINCH.

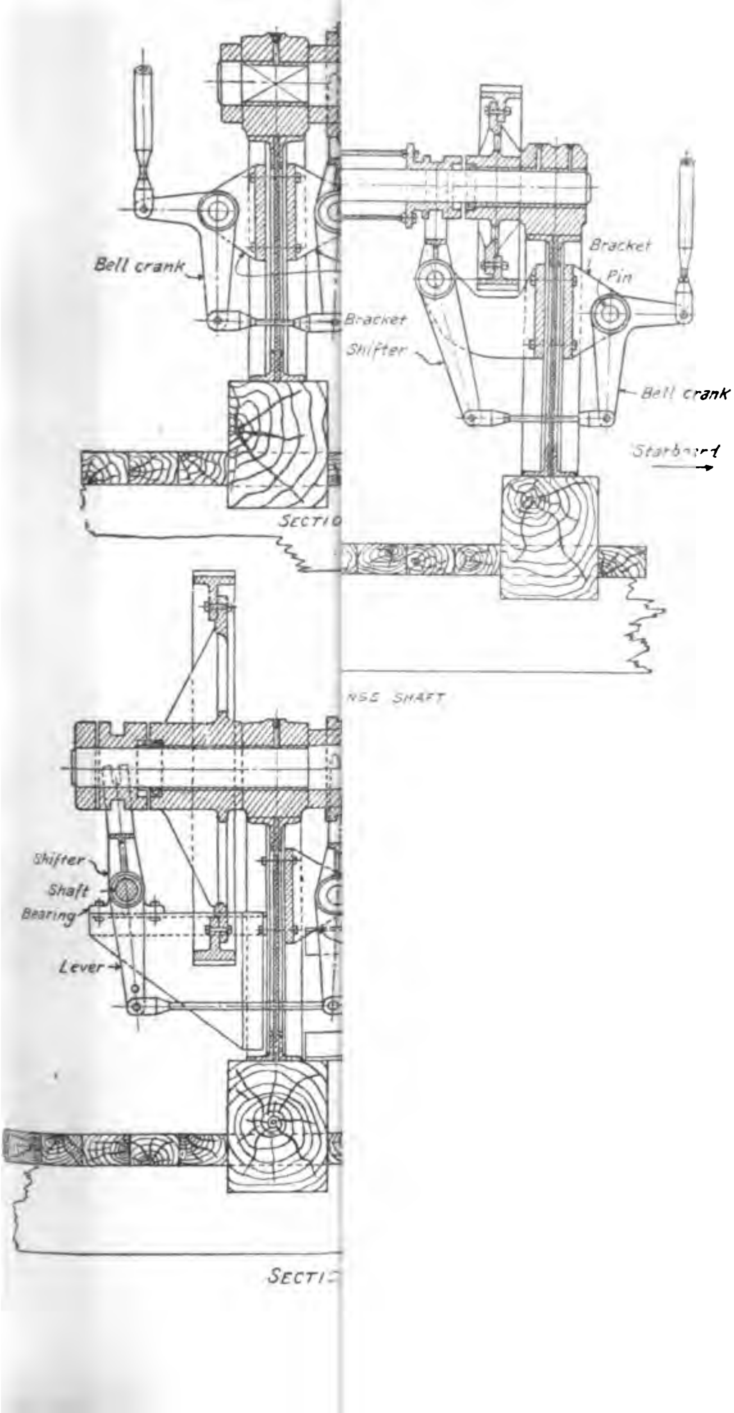
Some dredges have a separate winch for raising the ladder, the bucket-drive motor being placed at the main drive. Doubtless there are many advantages in having the winch for raising and lowering the ladder driven by its own motor and operated by a controller in the winch room, especially when the bedrock is uneven or where it is desirable to drop the ladder at certain parts of the cut, which is necessary in practically all dredging. If there is no separate winch for this it is necessary to raise and lower the ladder by means of levers, and the winch man may neglect to work the ground to the best advantage. The winch man should be able to operate the ladder and run the buckets independently of each other, and sometimes run the buckets while the ladder is being raised. Plate XXVIII shows a dimensioned drawing of the general arrangement of ladder-hoist winch equipment for a 9-foot dredge.

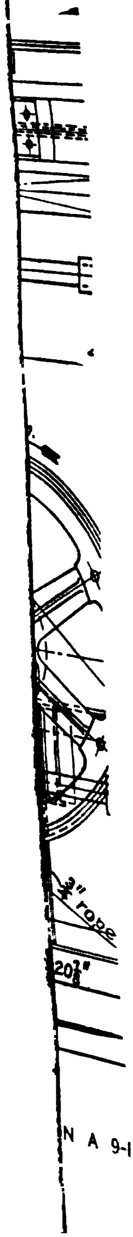
If the same motor is used to drive the bucket line and raise the ladder it must be large enough to perform both operations at the same time. When the bucket line only is being driven, the motor is not running economically, the load being much below its normal capacity.

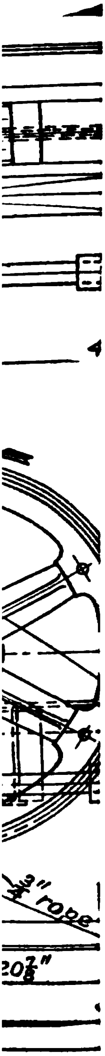
Another advantage of the separate hoisting winch (Pl. XXIX, *A*) is that there is no danger of dropping the ladder, a danger always present with the other type as the separate winch is so constructed that the motor must be run while lowering as well as while raising. The ladder is also lowered more evenly than with brakes.

When the ladder hoist and the bucket drive are combined, the brake bands should be placed directly on the drum; otherwise the chance of dropping the ladder is increased. The brakes should be powerful and they should be of the double-action type; that is, the tension from the hand lever should be thrown on the band on one side of the drum, which in turn should throw the increased tension on the other band.

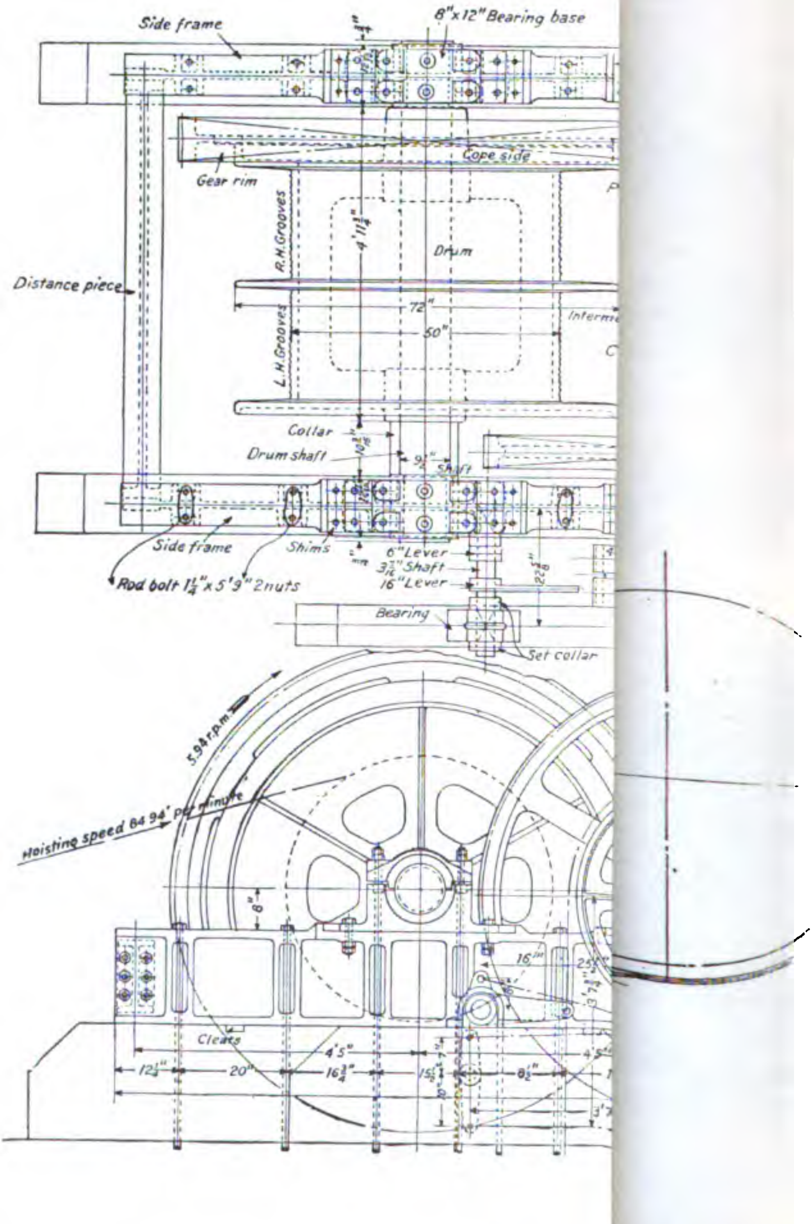
All revolving parts of the clutches should be built of cast steel. If made of cast iron and the ladder should drop faster than desired,


Bell crank
Bracket
Shifter
SECTIO
Bracket
Pin
Bell crank
Starboard
NSE SHAFT
Shifter
Shaft
Bearing
Lever
SECTIO


¾" rope
20⅞"
N A 9-1



N A 9-F


Side frame
8"x12" Bearing base
Cope side
Gear rim
R.H. Grooves
4'11¾"
Drum
Distance piece
72"
50"
L.H. Grooves
Interme
Collar
Drum shaft
9½"
Shaft
Side frame
Shims
Rod bolt 1¼"x5'9" 2 nuts
6" Lever
3 7/16" Shaft
16" Lever
Bearing
Set collar
5.94 r.p.m.
Hoisting speed 84.94' per minute
8"
16"
Cleats
4'5"
12¼"
20"
16¾"
15½"
8½"
3'7"

the clutches would revolve at a terrific speed and fly into pieces owing to the centrifugal force. This accident has happened on a number of dredges and should be guarded against. The teeth of high-speed gears should be cut. The drums should not be grooved, as it is not practical to make the drum sufficiently large to take all of the hoisting rope in one layer. Therefore, as there are several wrappings of rope, the lead should be arranged so that the wrappings of the line will be close together with no space between; otherwise wedging of one coil in an open space may hold the rope briefly and cause the ladder to go down in short jumps, endangering the dredge.

Several serious accidents to the ladder have been caused by the breaking of the rods leading to the brakes. These rods are as a rule made of 2-inch pipe, the different lengths being fastened together with standard couplings. The pipes being thin, break at the threads, owing to the strain and the vibration of the dredge. The pipes should therefore be extra heavy and fastened together with special couplings.

Maple blocks are generally used in the friction clutches. These wear out in a comparatively short time, and some experiments have been made with asbestos blocks. The latter are made in an iron mold under great pressure. Small copper wires are run through the blocks to bind the asbestos together. The coefficient of friction of these blocks is higher than that of the wood blocks, they wear longer, and the first cost is not great, but the objection to them is they wear out the steel surface against which they rub.

SWING WINCH.

The swing winch (Pl. XXIX, *B*) should be of the eight-drum type with two drums on each shaft and a bearing at the center of each shaft as well as at each end. It has been the custom to have one drum at the front end of the winch for the spare drum, but it has been found that a great saving of time has been effected in repairing wrecks if two spare drums are available. If there are not two spare drums it is necessary to use one of the swing drums and to take the rope from that drum and to wind on another. The time saved in one use of the spare drum more than offsets the extra cost. Another advantage of having two drums for spares is that in case of trouble with one of the other drums it can be taken out for repairs and one of the spares substituted without a serious shutdown of the dredge. For this reason all shafts with the drums and clutches should be interchangeable. Some of the shafts are subjected to much heavier strains than others because of the duty they perform, and it is often the practice to make some of the winch shafts lighter than others, but this practice is not considered good.

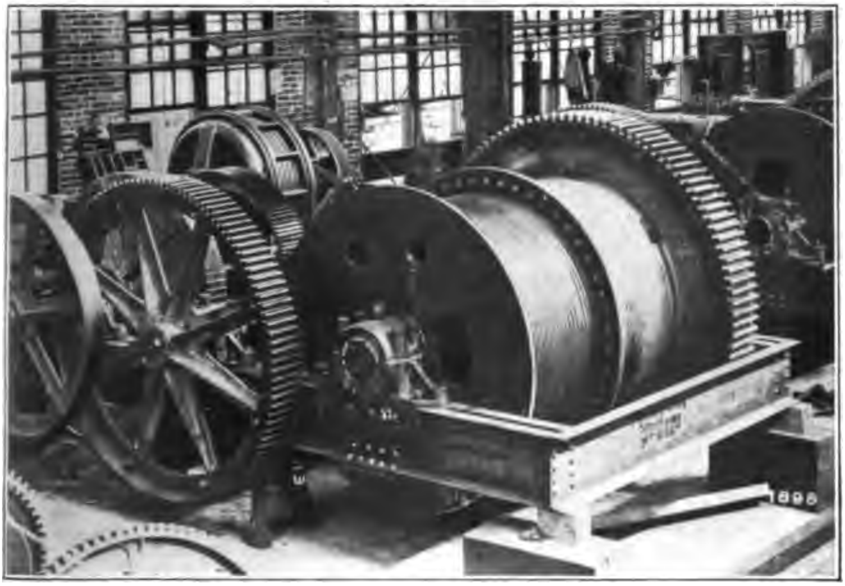
The spare drums should be placed at the bow end of the winch, the bow-swing lines next, then the spud hoist, and the stern-swing line drum at the stern end. The spud shaft should be arranged so that it will not revolve unless the spuds are being lowered or raised, as there is a constant tension on the drums causing the bushings to wear unnecessarily if the shaft revolves continuously. It is also desirable, but not so important, to arrange the spare drums in the same manner. The winch for a large dredge recently built was designed so that the bow drums and gear could be driven separately, the other gears remaining stationary except when desired. The arrangement proved satisfactory.

As the swing winch is usually placed on the starboard side of the boat, the port bow line must be run to the opposite side of the dredge. This is done either by leading the line to the stern and across the dredge amidships and then to the bow, or by bringing it forward and across the dredge through a sheave hung on the bow gantry. The latter arrangement is preferable, as the rope does not have to run around so many sheaves. If the first method is used, a very small sheave must be used to deflect the rope parallel to the deck, owing to the small amount of room under the winch; but when the rope is taken forward this deflecting sheave can be made as large as desired, as it is out in the clear. In taking the rope over the bow gantry a large sheave should be suspended at the center of the gantry. The size of the sheaves is of much more importance than is usually considered; it is discussed on another page under the subject of wire rope.

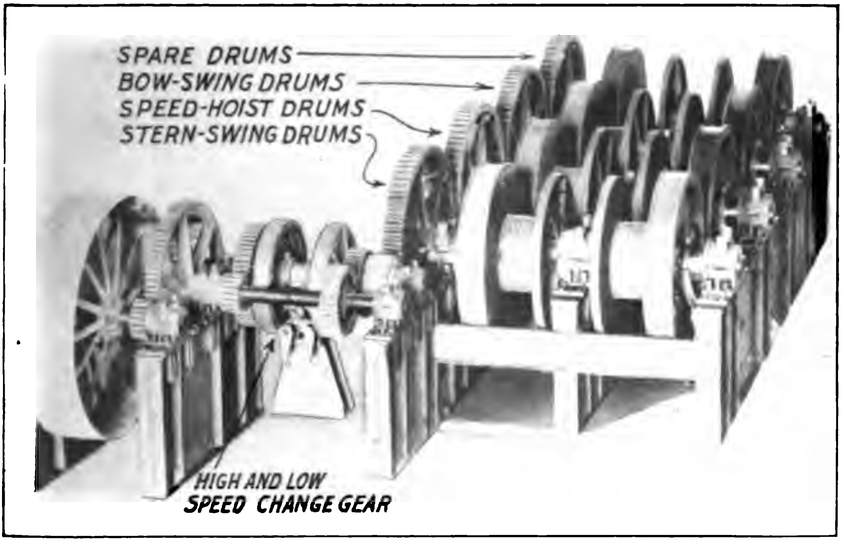
Most swing winches are constructed so that either of two speeds can be used for swinging the dredge. But in practice when the proper swinging speed has been determined the change speed is rarely used. In fact, most dredge operators disconnect it so that the winchmen will not be tempted to use it. For average ground the swinging speed at the lower tumbler is about 50 feet per minute. Experiments are being made by some of the larger companies with a view to increasing this speed, as it is realized that the buckets should be fed to the ground as fast as practicable. Strains in the side lines and other parts of the dredge are less when a rapid thin cut is taken than when a heavy slow cut is made.

When digging clayey ground it is especially desirable to take a thin cut as the clay is more easily broken in the screen when in thin slices than when in lumps.

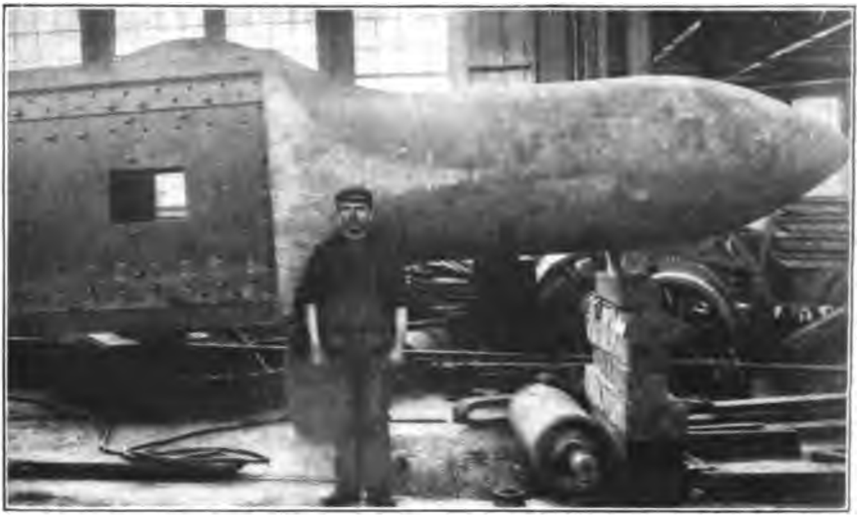
It is desirable to have a slow-speed mechanism on the winch so that in an emergency it can be used, for example, in repairing a broken ladder, or if it has been disconnected, it can be readily reconnected. On the older dredges this speed change was accomplished by jaw clutches in connection with two gear ratios. In modern large dredges such gear clutches have been displaced by friction clutches; these cost considerably more but are more easily operated.



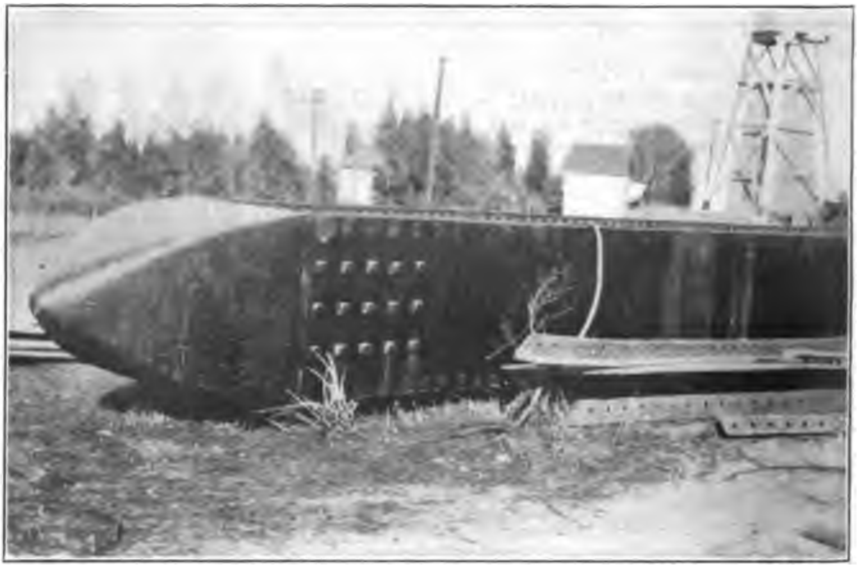
A. LADDER-HOIST WINCH FOR LARGE DREDGE, DIRECT CONNECTED.



B. STANDARD TYPE OF SWING WINCH.



A. CAST-STEEL ROUND POINT FOR SPUD.



B. STEEL SPUD, WITH SHORT SQUARE HEAD.

Most of the swing winches are driven with a belt, but the silent chain drive has been tried and will no doubt come into more general use. The winch motor should be supported above the winch, for when it is set on the deck there is always danger of water getting into it when the decks are washed.

On some dredges the designers have saved on the first cost by using jaw clutches instead of friction clutches on each drum, but these have not proved satisfactory, as time is lost at each swing of the dredge in getting the jaws in mesh. Another disadvantage is that the dredge is reversed with a jar. This jar is reduced by using a friction clutch. As clutches on dredges are subjected to severe service, the standard type of friction clutch is not strong enough, so a special type of internal expanding band clutch has been developed for this work and has proven efficient.

SPUDS AND SPUD SUPPORTS.

The most important item to consider in connection with spuds is strength. Spuds not only take the thrust caused by digging but they are also subjected to a torsional stress from the swinging of the dredge from side to side. The torsion can be reduced to a minimum by making the spud-point casting very long, and the cross section round except where the point is riveted to the structural-steel part of the spud. The spud is in the sand to a depth of several feet, and the round point (Pl. XXX, *A*) offers less resistance to the swinging of the dredge than does the rectangular section that was until recently in general use.

The designing of a sufficiently strong spud would seem simple, the spud being in effect an overhanging beam supported at the hull and the upper truss chords. However, with perhaps one or two exceptions, a spud has rarely stood up during the life of the dredge for which it was built.

Wooden spuds were used for the early dredges, but were not found suitable. Later, some dredges used a steel spud for digging and a wooden spud for stepping ahead, and some dredges even now use that arrangement. This practice is not good for two reasons—first, either spud should be available for digging as conditions often arise that make the use of both necessary, and second, there should always be a spare spud for use in digging if breakage occurs; the spare should be carried on the dredge, as often it can not be transported to the dredge. Thus, if one spud cracks or the rivets loosen, digging on it can be discontinued before it actually breaks and it can be used as the stepping spud until the dredge is in a suitable location to permit its removal for repairs or the substitution of a new spud.

There are two types of steel spuds in general use, one having a rectangular cross section (Pl. XXX, *B*, and fig. 6) and the other of the same general shape, but with a center web running the length of the spud (fig. 7). The latter type is much more difficult to build and to repair. Tests have been shown that if the center web be omitted and the metal used to thicken the outside plates, the life of the spud is much longer. On a large dredge recently built an experimental spud is made of four 30-inch I beams bolted together (fig. 8).

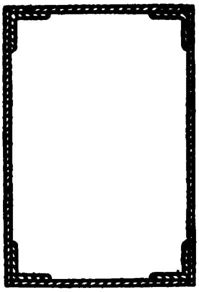


FIGURE 6.—Cross section of box type of spud.

Breakage of many spuds theoretically strong enough is caused by the rivets becoming loose and allowing the plates to fail. It is difficult and often impossible to keep the rivets tight on most of the dredges. The tremendous bearing pressure on the spud kick is distributed over three or four rivets only, causing them to become loose. During the past year most of the large dredges have been designed to overcome this by making a swivel casting which bears against the spud at all positions for a distance of 3 or 4 feet, thereby distributing the bearing pressure over a great number of rivets. Plate XXXI, *A*, shows one method of transporting a spud.

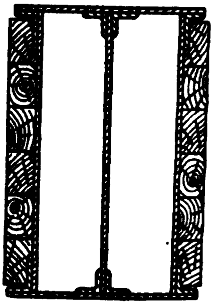


FIGURE 7.—Cross section of spud with center web.

The life of most dredge hulls is shortened by their failure at points where the spuds bear against them. In many dredges there is a steel plate between the hull and the spud. On such dredges the plate is apt to be driven into the wooden hull, the intermittent pressure loosening the bolts in the hull and allowing the water to leak through. In some of the steel dredges trial was made of a small casting riveted to the hull plates, but the result was the same. It is now the practice to rivet a casting several feet wide the entire depth of the hull.

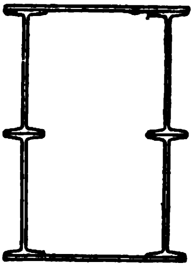
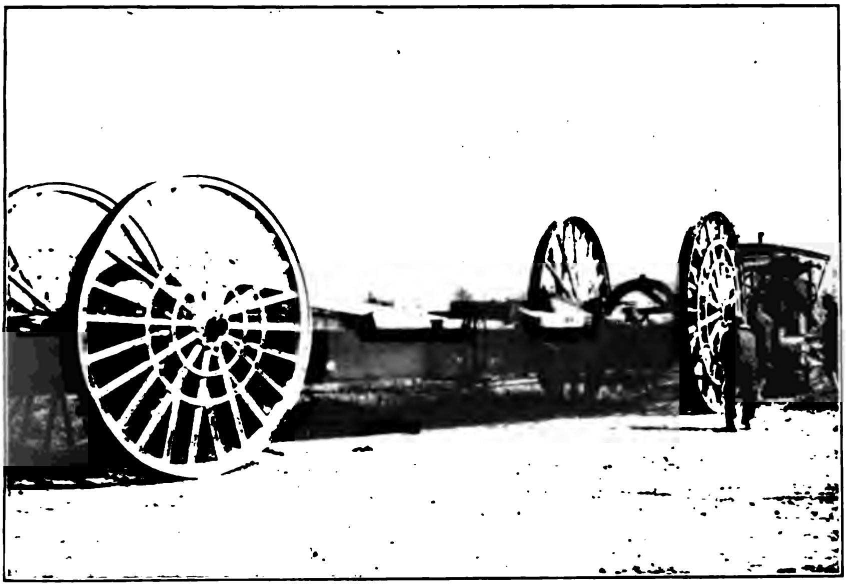
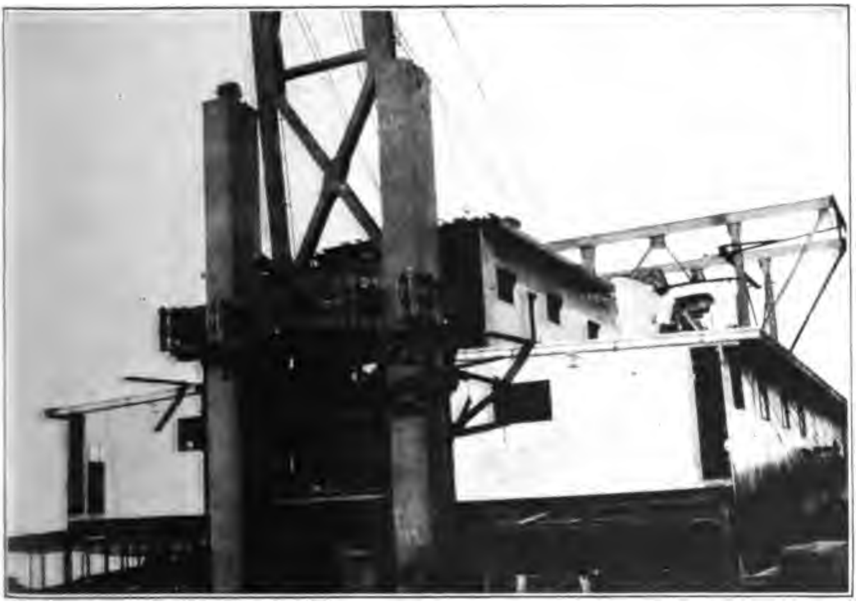


FIGURE 8.—Cross section of spud made of four 30-inch I beams bolted together.

For a while wooden casings just large enough to receive the spud were built on the hull from the upper truss to the deck. Timbers were bolted to the spuds on the sides. The casings served no good purpose and caused trouble. Small stones, nuts, or other pieces of material dropped between the casing and the spud and necessitated the use of more power to raise the latter. Casings should be left open except at the upper truss chord and at the hull where the stresses can be taken care of. It is the usual practice to use buffer springs at the upper supports to do away with excessive jar (Pl. XXXI, *B*). Springs have also



A. HAULING A SPUD.



B. EQUALIZING SUPPORT FOR SPUDS.

been used at the hull, but as it is necessary to put them under the water, where it is impossible to examine them at intervals, the practice has been discontinued, although they never gave trouble.

Several spuds have been bolted together instead of riveted, lock washers and sometimes turned bolts which were made a driving fit being used. Bolts stay tight much better than rivets and if they become loose they can be tightened. They serve the purpose very well and will no doubt do away with much of the trouble experienced with rivets.

A single digging spud was placed on each of several dredges with the idea that by placing it in line with the bucket ladder, to oppose the digging force, the stresses in the dredge could be more effectively distributed. It is possible to take care of the eccentric load caused by the spuds being placed off center in the usual manner, so there seems no good reason why the single spud should be used. With these dredges a light stepping spud was placed as in the standard dredge. The use of only one digging spud entails many disadvantages, and the dredges so equipped have given much trouble—the spud comes at the end of the screen and all of the material has to be by-passed through a curved chute; the stacker must be placed to one side and run out at an angle, which complicates the suspension, and the spud is in the way when the screen and the drive are being repaired. A still greater disadvantage is that when the spud is out of order or the hull at the spud needs repairing, the dredge is shut down; this is not necessary if the dredge is equipped with two spuds. When two spuds are used and the rivets on one become loose, the other is put into use and the loose rivets are tightened while the dredge remains in operation.

The length of the spud is determined by the depth of digging and the height of the top truss chord. The following table shows how the sizes of spuds vary according to the ideas of the constructors and the conditions of operation:

Data regarding lengths of spuds used on different dredges.

Size of bucket.	Size of spud.	Depth of digging below water.	Length of spud.
Cu. feet.	*Inches.*	*Feet.*	*Feet.*
4	26 by 36	35	59
5	24 by 36	50	53
5	24 by 36	30	45
6	24 by 36	60	53
7	24 by 36	30	48
9	24 by 36	65	61
13	34 by 54	45	60
15	30 by 42	35	61
16	38 by 50	65	70
16	36 by 54	55	80
16	38 by 60	70	62
16	36 by 54	40	60

HULLS AND SUPERSTRUCTURES.

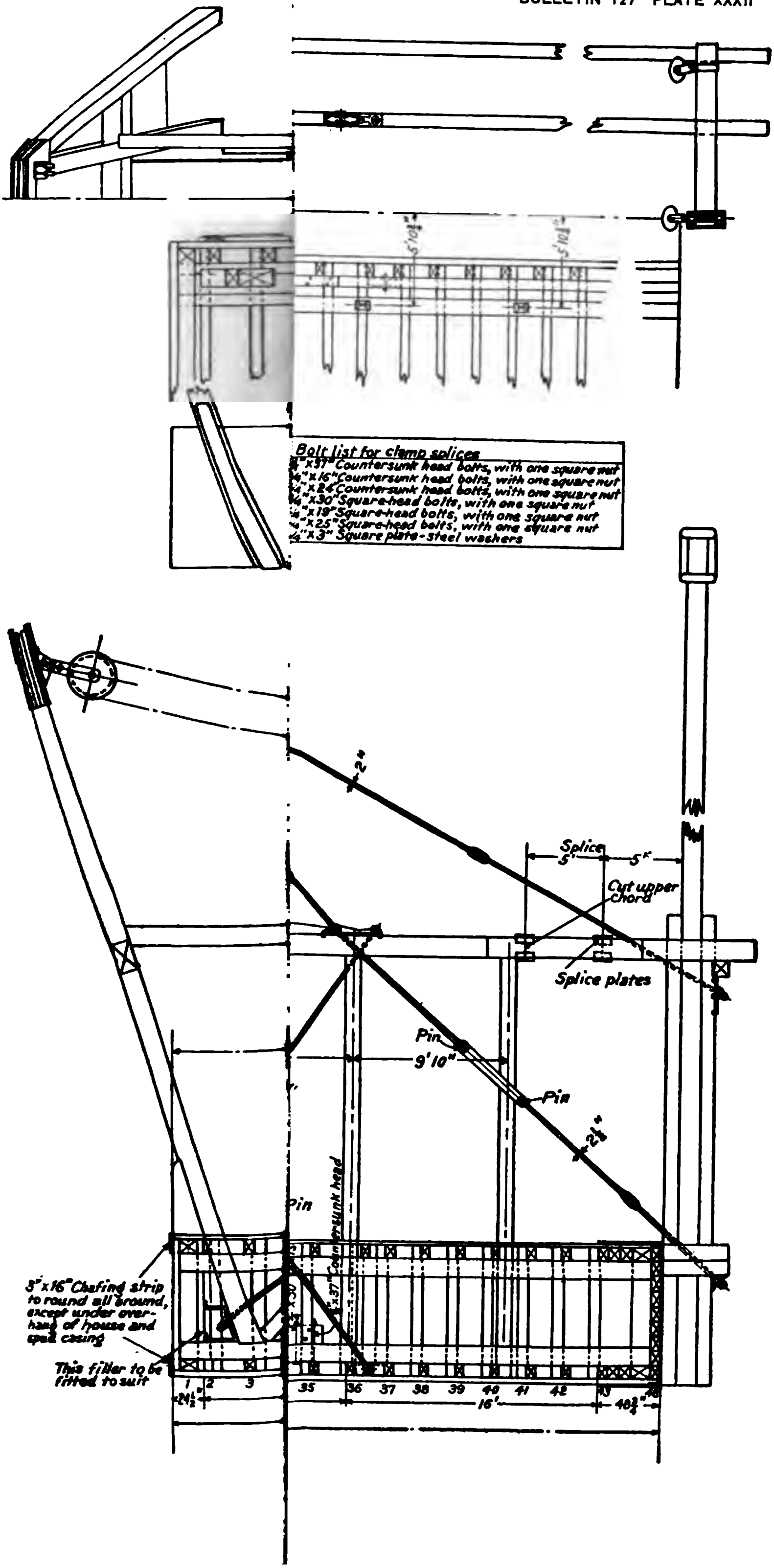
The hull is rectangular, with an opening or well hole running from the bow to about the center. The ladder and bucket line extend through the well hole, and the well hole should be no longer than is absolutely necessary for the operation of the buckets when the ladder is in a position 45° with the vertical, this being the maximum angle at which it should operate. The outside corners of the pontoons are cut off at the bow on a long curve. The object of making the hull narrow at the bow is to avoid interference with the bank when digging in corners. The early California dredges dug shallow ground; the ladder did not extend much ahead of the hull, and it was necessary to make the hull narrow at the bow. With long digging ladders the corners should be cut as little as possible, and under certain conditions they need not be cut off at all. The most important reason for this is that a larger proportion of the ladder is supported by the bow gantry which in turn is supported by the forward end of the hull, and it is necessary to get as much flotation at this point as possible. Consequently, the front wall of the hull should be square with the deck and not beveled, as on some dredges. Plate XXXII shows an elevation of a dredge hull and Plate XXXIII shows details of hull construction.

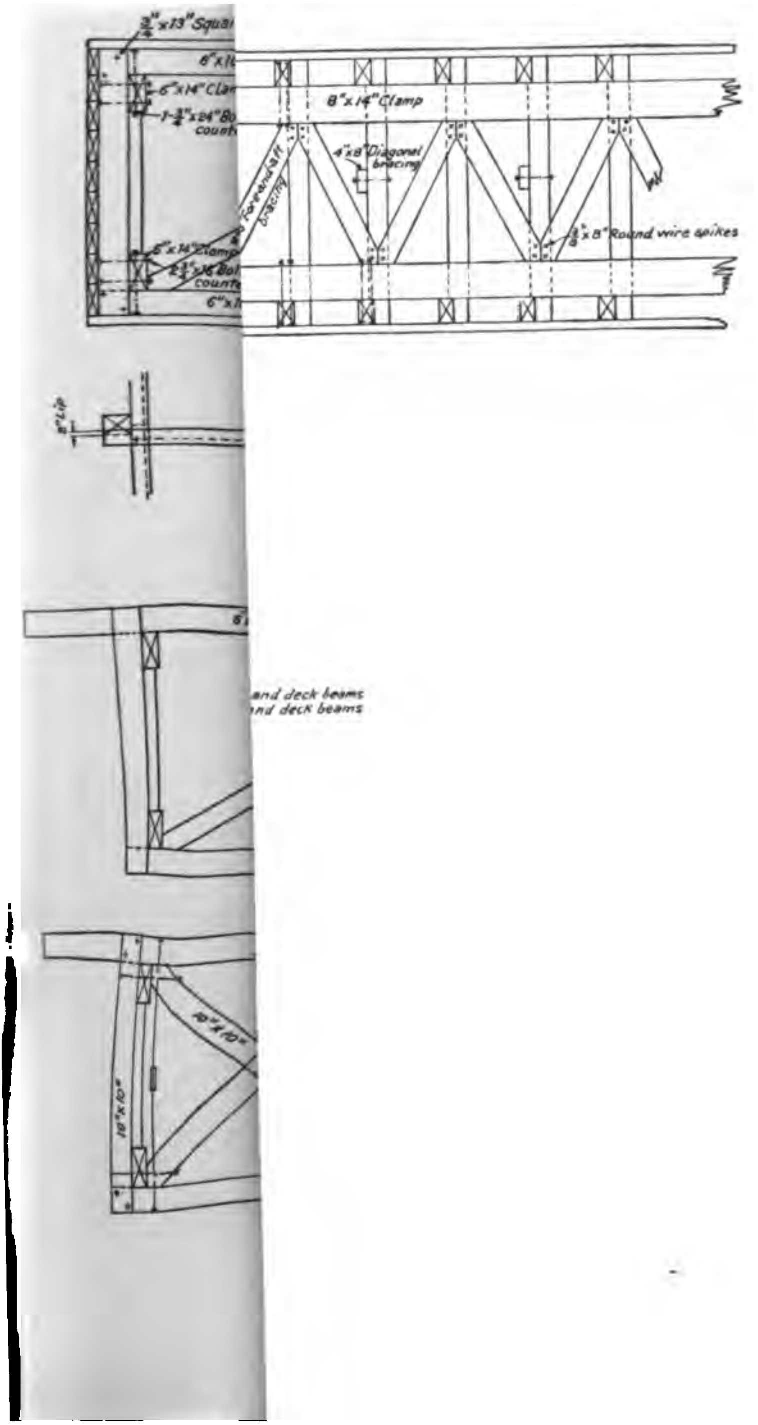
If much sand is to be encountered at the stern of the dredge, it is good practice to cut off the corners at the bottom of the stern, so that when the dredge is swung these corners will be clear of the sand tailings (see Pl. XXXIV, *A*). The deck is usually allowed to extend over the hull 3 to 5 feet on each side, but this extension is desirable only if there is not sufficient room on the deck for the winches and pumps.

Hatchways are placed in the deck to give entrance to the hull at different points, but they should be arranged so they are water-tight when the dredge is in operation, so that if the hull is injured it will not fill with water.

The material of which the hull should be built is determined by local conditions, such as the comparative cost of steel, whether suitable timber is near the site of operation, the life of the dredging ground, the availability of skilled labor, and whether parasites or chemicals are present in the water. The greatest advantage obtained by the use of steel hulls (Pls. XXXIV, *B*, XXXV, and XXXVI) is the elimination of the fire hazard, thus not only forestalling losses occasioned by fire but effecting a great saving on insurance. The steel hull has other important advantages.

The life of the property is, however, not such an important item as is sometimes considered, for there is no reason why a wooden hull should not be built to last 15 years, and by that time the machinery


5'10½"
5'10½"
Bolt list for clamp splices
"x37" Countersunk head bolts, with one square nut
"x16" Countersunk head bolts, with one square nut
"x24 Countersunk head bolts, with one square nut
"x30" Square-head bolts, with one square nut
"x19" Square-head bolts, with one square nut
"x25" Square-head bolts, with one square nut
"x3" Square plate-steel washers
2"
Splice
5'
5"
Cut upper chord
Splice plates
Pin
9'10"
Pin
2½"
Pin
x37" Countersunk head
x30
5"x16" Chafing strip to round all around, except under overhang of house and spud casing
This filler to be fitted to suit
1
2
3
35
36
37
38
39
40
41
42
43
44
24½"
16'
48¾"


3/4"x13" Squa
6"x1
6"x14" Cla
1-3/4"x24" Bo
count
8"x14" Clamp
4"x8" Diagonal bracing
Fore and aft bracing
6"x14" Clamp
2 3/4"x16 Bol
count
6"x1
3/8"x8" Round wire spikes
8" Lip
6
and deck beams
and deck beams
10"x10"
10"x10"

will have outlived its usefulness, not only on account of wear but because it will have become obsolete.

The usual life of wooden hulls is only 8 to 10 years, as practically all of them are built of wood that has not been treated in any way by preservatives or has had only an external coating of preserving liquid. Careful tests have been made by the Government with rock scows and barges which operate under conditions similar to those encountered by dredges, and it has been determined that when built of untreated timbers they will last approximately eight years. However, barges built of timbers whose pores had been thoroughly filled with creosote by the vacuum and pressure process were excellently preserved at the end of 15 years. There is no doubt that all wood hulls should be built of creosoted timbers regardless of the probable life of the dredging operations. The additional cost for timbers treated in this manner is about 5 per cent. The timbers used in construction should also be as dry as possible; some constructors contend that dredge hulls built in the winter season will not have as long life as those built in dry weather.

It is often necessary to replace timbers that have been subjected to heavy strains or that have rotted from moisture. On some dredges the upper truss chord is under tension and bending strains due to the weight of the main drive. As timbers should never be depended upon to take tension, rods should be placed to take this strain. Timbers should not be used to support the screen, as the leakage causes rapid decay, and the vibration and the weight force the bearings out of line. It is not uncommon to replace the screen timbers every three or four years. These supports should therefore be built of structural steel, which is more economical and more satisfactory in every way than wood.

The truss chord and screen supports are the only timbers that give trouble above the deck, as the bow and stern gantries are usually in good condition when the dredge is dismantled except at the deck and where they project into the hull. These timbers as well as the truss posts should have a casing built around them at the deck, into which asphaltum or other material can be poured to keep out moisture. All heavy timber should be set in cast-steel or cast-iron bases at the bottom and have caps at the top, not only in order to keep out moisture, but to prevent the end of the timber from crushing the supporting member. Each of these bases and caps should have about three times as much bearing surface along the cross timber as on the end of the uprights; Douglas fir or Oregon pine, for example, should not have to support a weight of more than 250 pounds per square inch across the grain, an endwise pressure of 800 pounds per square inch is safe.

Decay always sets in at contact surfaces when moisture is present and extreme care must be used in protecting such places. Probably, however, the greatest source of trouble is the practice of running the truss rods through the timbers, as the water from the deck runs down along these rods into the timbers. Double truss rods should be used and should be placed outside of the timbers and connected at the ends to the bases and caps. This arrangement is somewhat complicated but it should always be followed.

The life of practically all wooden hulls is determined by the life of the stern, where the spud thrust is taken up. As mentioned under the subject of spuds, the latter should not bear directly against the hull, but a casting should be placed between in order to distribute the bearing strain to all of the main timbers running lengthwise of the dredge.

A mechanical circulation of air through the hull considerably prolongs the life of the timbers. The hull should be so designed and the machinery so placed that the bow of the dredge is slightly higher than the stern when the ladder is in the highest position, so that at all times the water will drain to the stern, where it can be pumped out.

Data on sizes of hulls for dredges of different capacities and digging depths are given in the table following:

Data on hulls of different dredges.

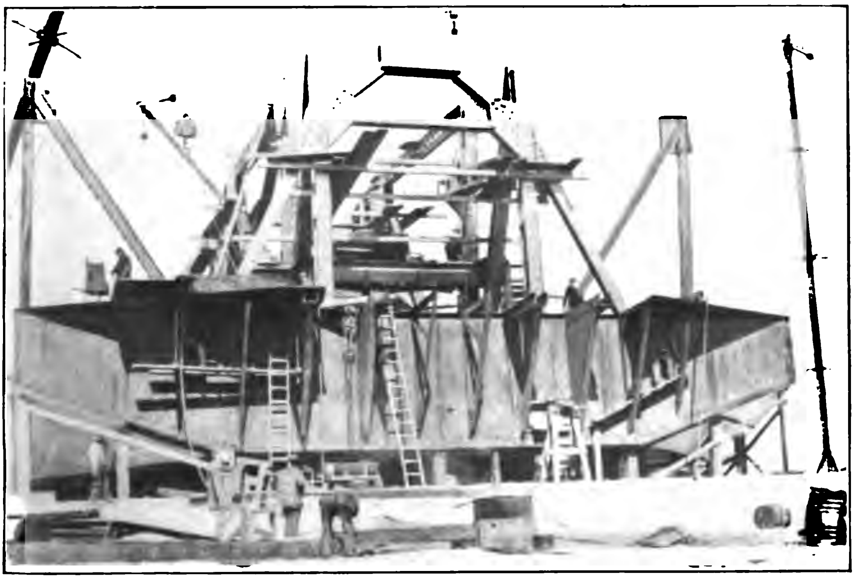
Size of buckets.	Digging depth.	Material of hull.	Width of hull.	Length of hull.	Depth of hull.	
Cubic feet.	*Feet.*		*Feet.*	*Feet.*	*Ft.*	*in.*
5	35	Wood...	36	106	7	9
5	55	...do....	36	115	8	0
7	25	Steel....	40	115	8	0
7	65	....	42¾	130	8	6
9	25	Wood...	46	130	8	6
9	65	...do....	50	145	9	0
13	35	...do....	56	135	[a]14 [b]12	6 0
16	65	...do....	58	155	12	6
16	65	Steel....	58	155	12	0

[a] Bow. [b] Stern.

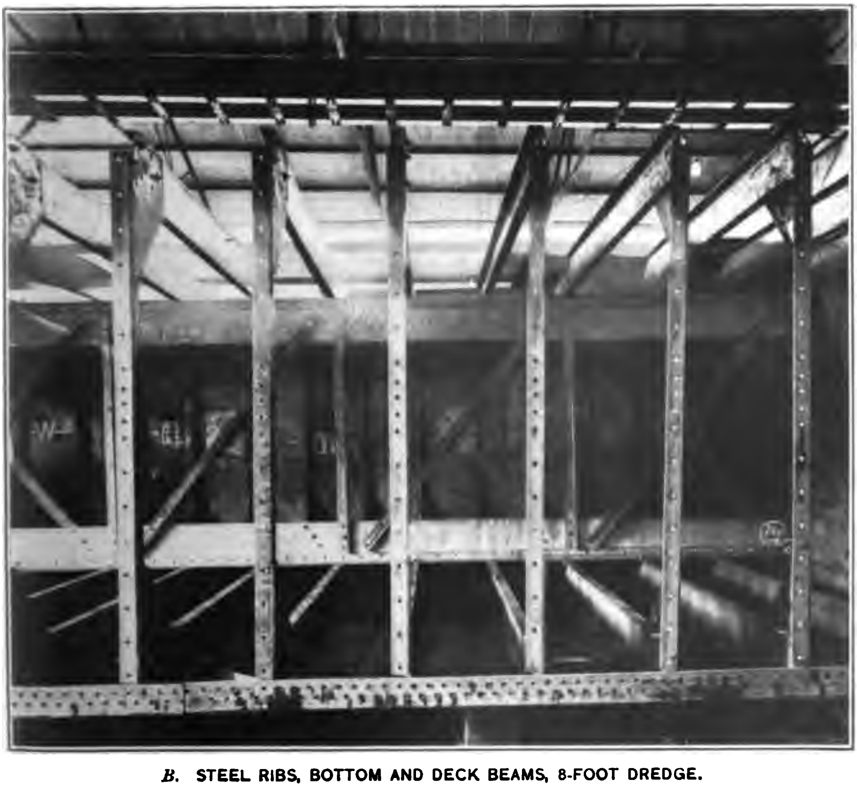
A hull constructed of steel can be divided into several water-tight compartments, so that if one compartment suffers accident the water is confined to it. The most vulnerable parts are at the bow and stern, and therefore a compartment is built at the bow on each side, another compartment is built across the stern, and others are constructed on each side and at the end of the well hole.

BOW GANTRY.

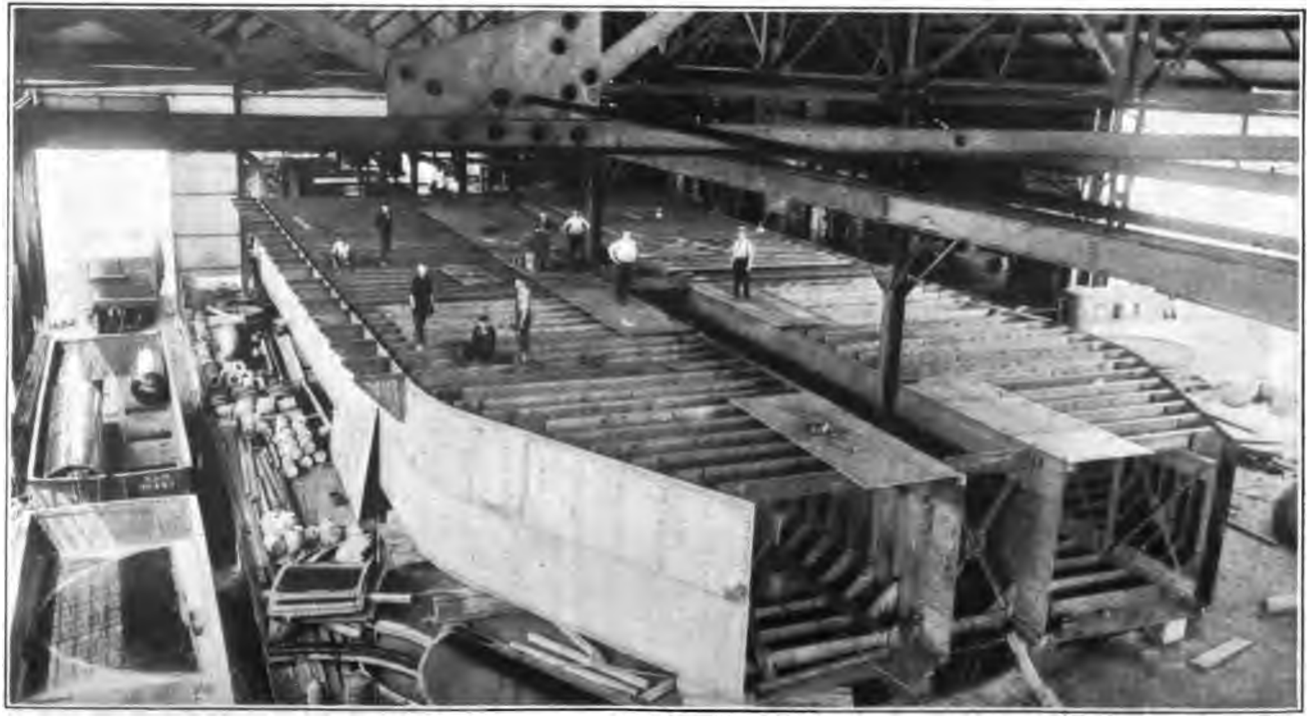
The bow gantry should not be placed near the front edge of the hull, but back a considerable distance so as to give as much flotation surface as possible under the gantry, as mentioned previously.



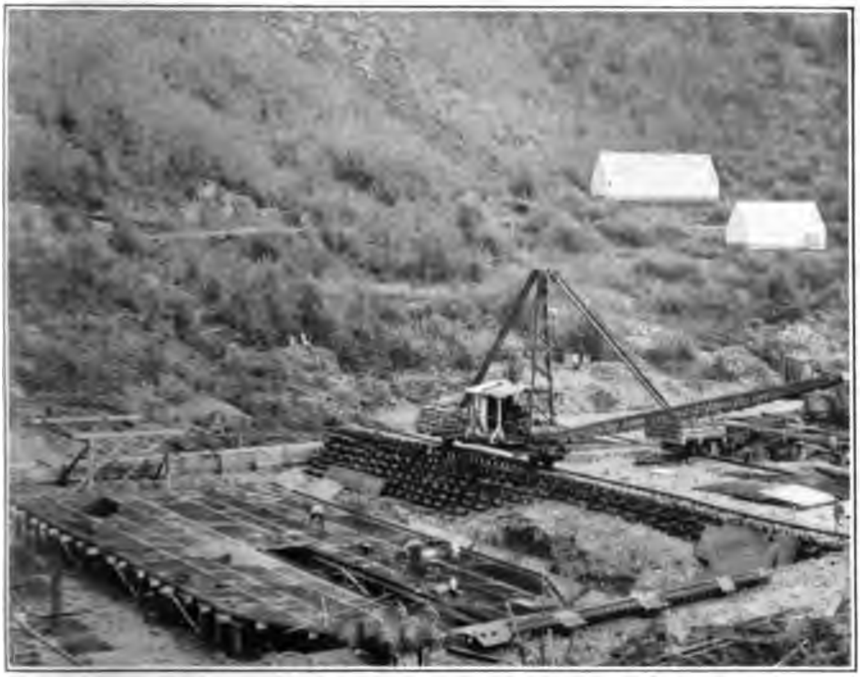
A. STEEL HULL WITH BEVELED STERN, YUBA NO. 15 DREDGE.



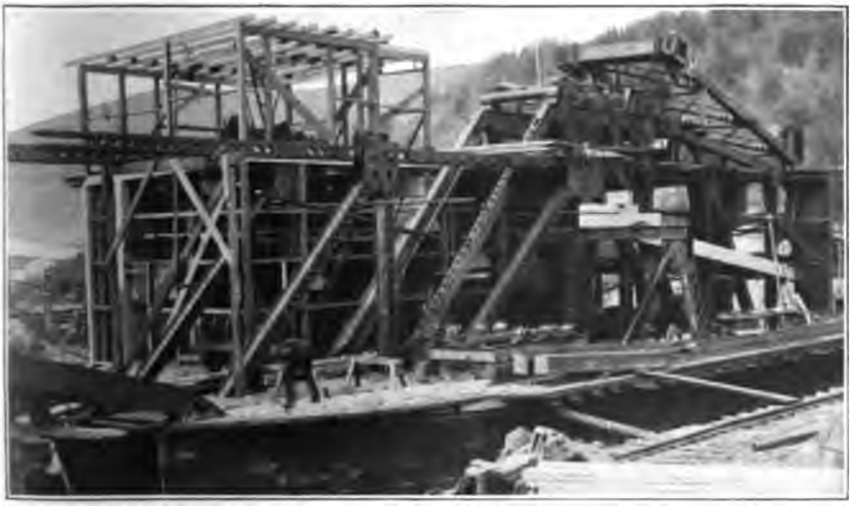
B. STEEL RIBS, BOTTOM AND DECK BEAMS, 8-FOOT DREDGE.



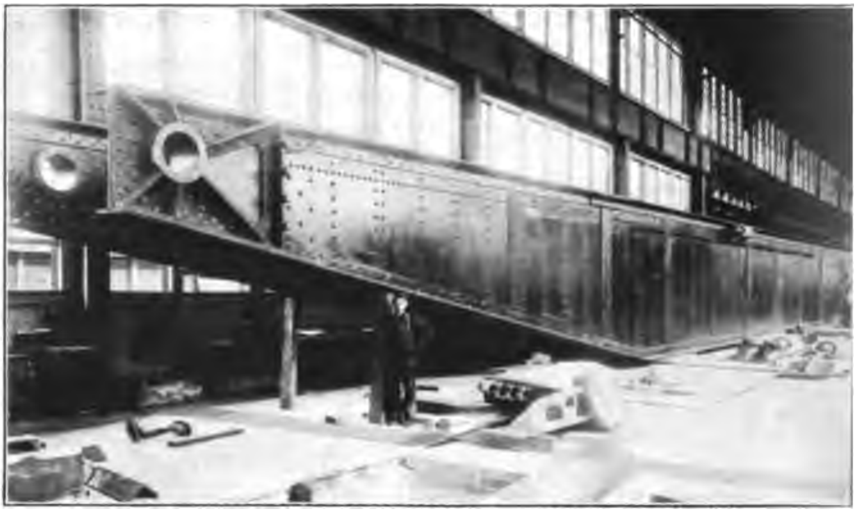
STEEL HULL UNDER CONSTRUCTION.



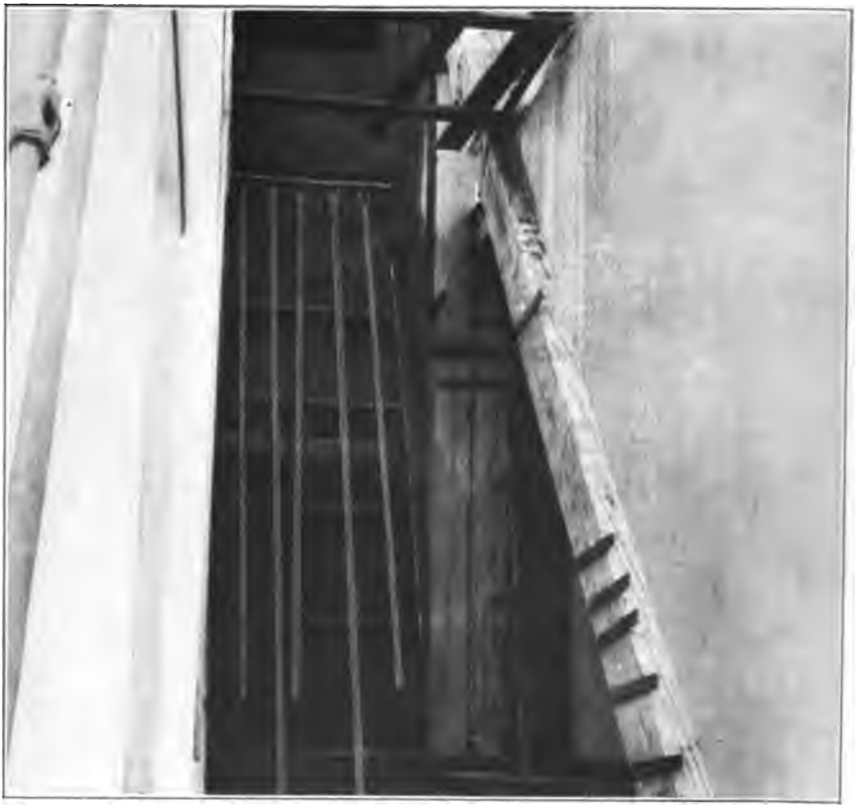
A. YUKON NO. 9 DREDGE UNDER CONSTRUCTION.



B. SUPERSTRUCTURE OF A STEEL DREDGE,



A. PLATE-GIRDER DIGGING LADDER.



B. SAVE-ALL SLUICE, CONREY NO. 2 DREDGE.

The most desirable height of the bow gantry depends on the length of the ladder (Pl. XXXVII, *A*). It is necessary to make the gantry high enough so that the ladder lines will not pull at too small an angle when the ladder is at its highest position. The bracing of the gantry must be arranged so the ladder can be raised to such a height that the lower tumbler can be rested on the bank in front of the dredge during repairs, for otherwise it is necessary to dig away the bank. The present practice is to build the bow gantry as a unit and to set it in bases that allow pivoting. This is a great convenience during erecting as the lower end can be placed in the bracket and the gantry swung into place. The old method was to build the posts into the hull.

The stern gantry supports the outer end of the stacker and the spuds. It should be high enough to allow the spuds to be raised clear of the lower supports so that they can be taken out without dismantling the supports.

LADDER ROLLERS AND BEARINGS.

Ladder rollers except for a few minor details, have become standardized. Formerly three hubs were used—one on each end and one in the middle, the shaft being pressed in. The intent was to have a press fit at the three hubs, but owing to the fact that it was practically impossible to apply calipers to the center hub accurately this center bearing is no longer favored by most dredge makers, as it adds nothing to the strength of either the roller or the shaft. Many dredge makers use manganese rollers, which are well suited to this service. The only objection to a manganese roller is that if it becomes loose on the shaft the bore wears oblong and it can not be used with another shaft. Nickel-chrome steel is the next best material for roller construction, and next to that high-carbon steel.

The ladder-roller shafts wear out rapidly and some shafts are being used with manganese or nickel-chrome sleeves. The design of the bearings determines the life of the shaft. The method of suspending the bearing on lugs that extend below the bearing (Pl. XXXVIII, *A*) has not been found the best practice on account of the greater movement and consequent wear when this type is used as compared to the method of pivoting the bearing on the center line of the shaft (Pl. XXXVIII, *B*).

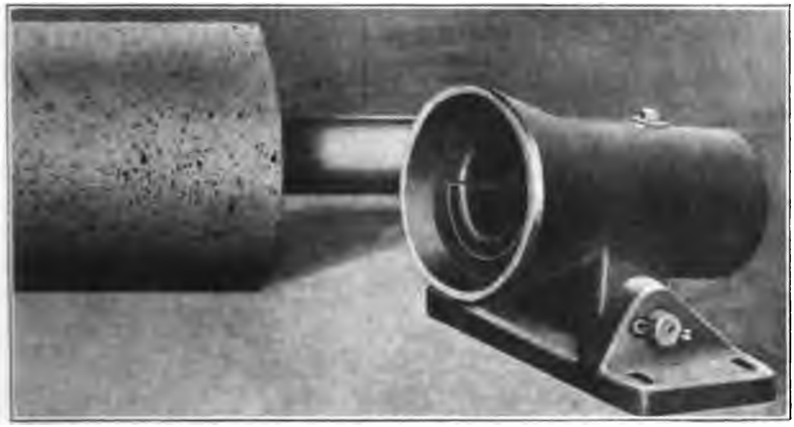
Ladder-roller bearings are about the only parts of the modern dredge made of cast iron, and the cast-iron bearings are being replaced with steel.

MAIN HOPPER, SAVE-ALL, AND BUCKET IDLERS.

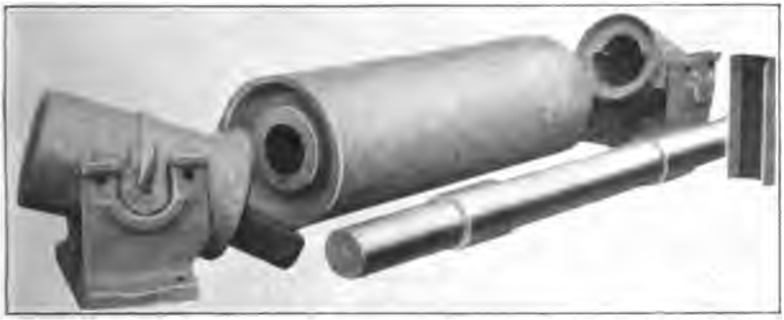
For several reasons it is desirable to have the upper tumbler no higher above the deck than necessary. Such construction not only lowers the center of gravity and therefore increases the stability of the dredge, but it keeps down the weight. Of greater importance is the waste in power in raising the gravel the extra distance necessitated by an unduly high upper tumbler, and in hoisting the additional weight of the ladder and in the bucket line.

The distance between the upper tumbler and the deck can be kept to a minimum only if the main hopper into which the material is dumped from the buckets is designed to avoid a long drop. The hopper must also be designed to prevent excessive spilling. In general shape the main hopper is like a box with the top removed and one side cut away to within a short distance of the bottom. This distance is just great enough to keep the material from running over and should allow minimum clearance of the buckets when they are digging the maximum depth. The spout leading from the hopper to the screen is set about 6 to 8 inches above the bottom so that the hopper fills with gravel to that distance and there is no wear on the bottom. The side removed should be just wide enough for the buckets to pass through. Chutes are placed on each side to carry into the hopper material spilled over the sides. Water jets should play against the sides of the buckets and on the inside so that no material will be carried back to the pond. The inside of the hopper should be lined with manganese steel bars so designed that when the first set of bucket pins need replacement the worn pins can be used to line the hopper. Worn bucket pins can be used to great advantage here.

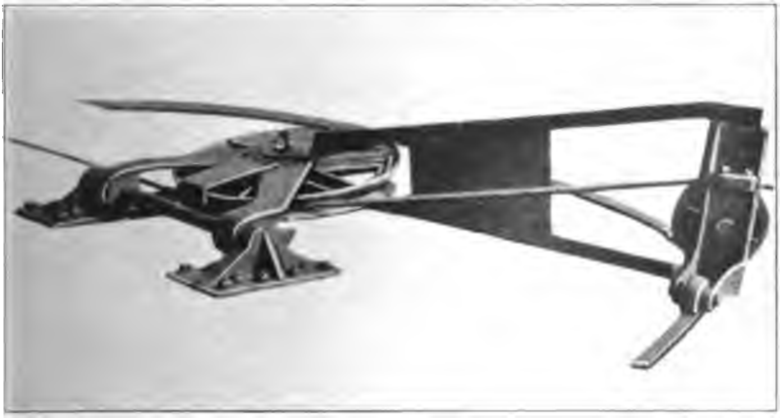
The save-all sluices (Pl. XXXVII, *B*) should have a greater pitch than the tables owing to larger wash. It is difficult to make them so that they are easily accessible for cleaning. A grizzly (Pl. XXXIX, *A*) with about 2-inch openings should be placed above them to keep out the larger rock. The most important point in connection with the grizzly is to make it so that it can be easily repaired. With the old type of bars repair was very difficult, as the bars were held together by rods running through holes in the bars with a spacer between. This grizzly was in one piece and owing to the construction at this part of the dredge it was usually necessary to run the rods through the large timbers supporting the main drive, building them into the dredge. The removal of the rods was difficult; it was necessary to handle the bars as a unit, and some of the grizzlies were 8 feet wide and 14 feet long and weighed over 6,000 pounds. The best practice is to cast a rack in which the bars can be set, each bar being removable, as the wear is limited to a small area and it is not necessary to throw the whole grizzly away because this area becomes worn. When one end of the bar is worn it can be turned end for end and then moved to the edge where the service is not severe. This type of grizzly, which is shown in Plate XXXIX, *A*, will wear many times as long as the old type.



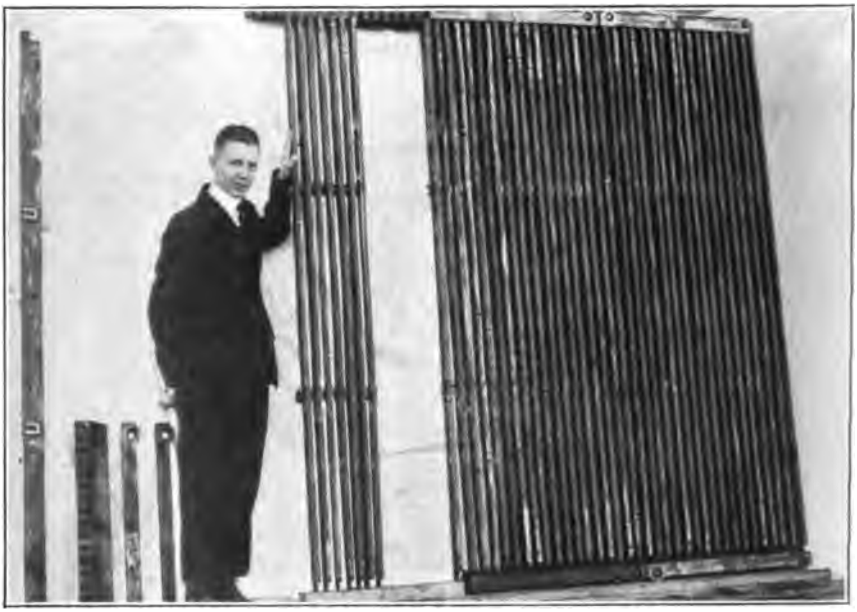
A. LADDER-ROLLER BEARING SUSPENDED ON LUGS.



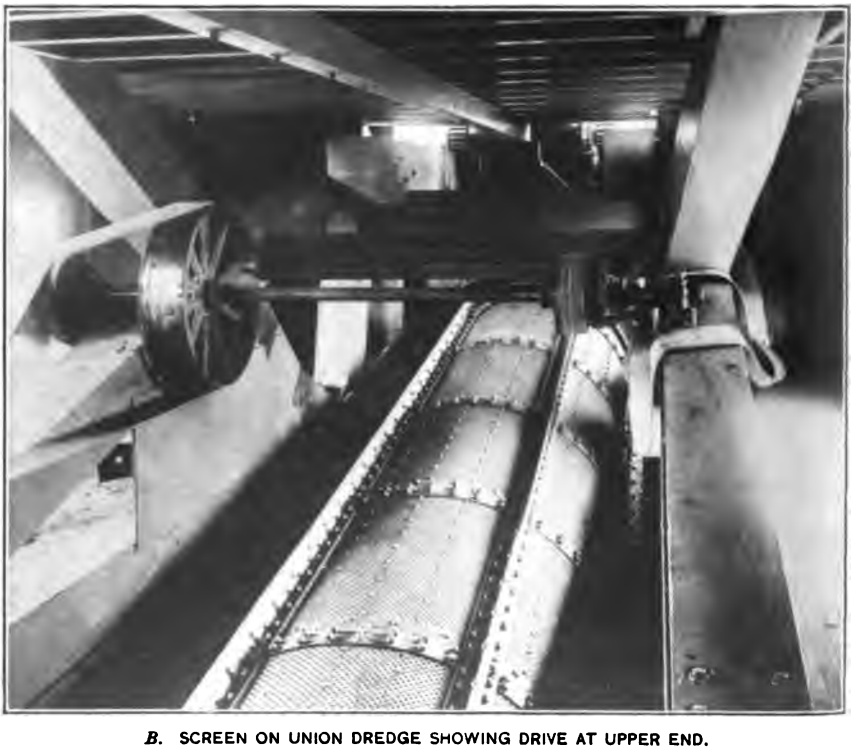
B. LADDER-ROLLER BEARING, PIVOTED TYPE.



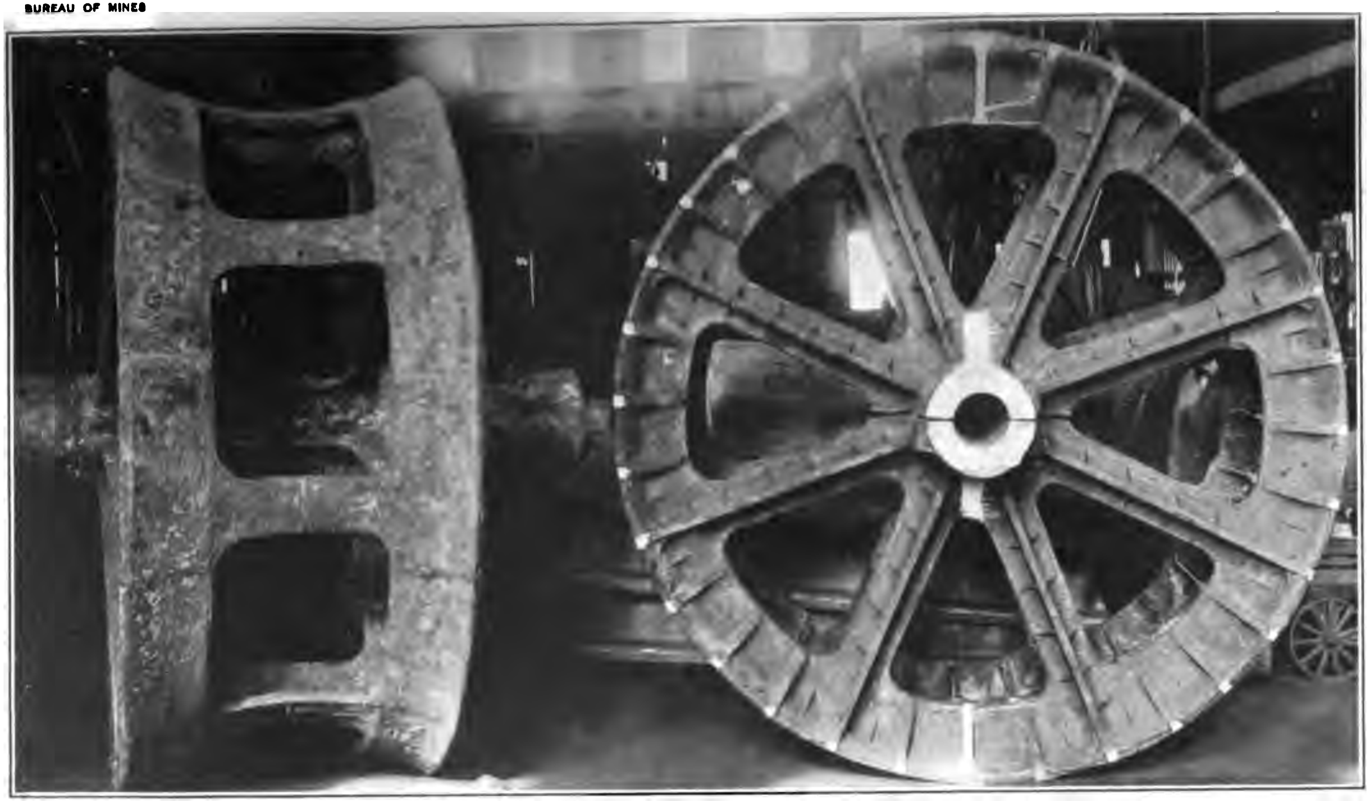
C. SHEAVE FOR BOWLINE.



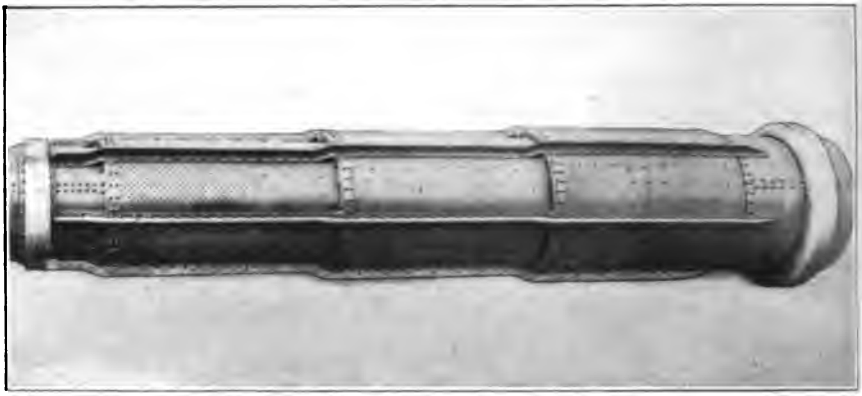
A. GRIZZLY OVER SAVE-ALL SLUICE; HAS CAST RACK FOR HOLDING MANGANESE-STEEL BARS.



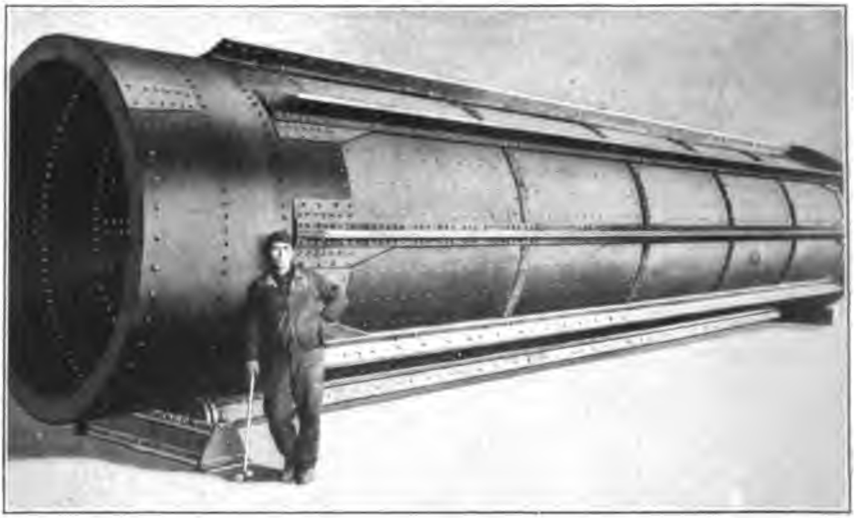
B. SCREEN ON UNION DREDGE SHOWING DRIVE AT UPPER END.



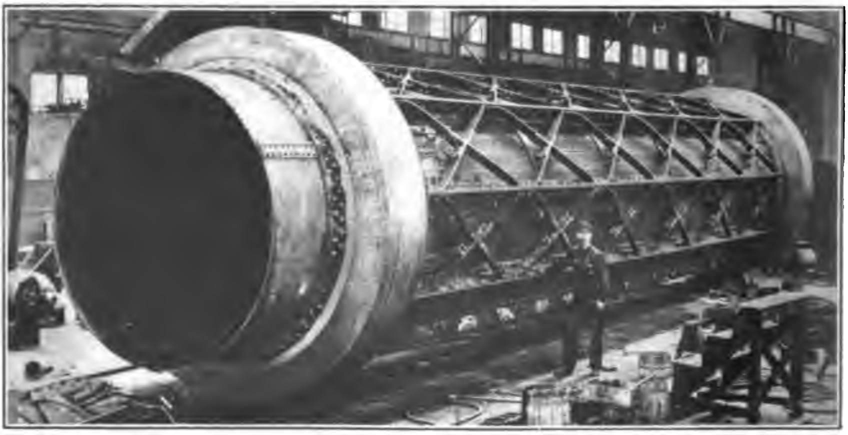
ONE-PIECE BUCKET IDLER.



A. REVOLVING SCREEN, SEGMENTAL STEP TYPE.



B. CYLINDRICAL SCREEN, 50½ FEET LONG AND 9 FEET IN DIAMETER FOR 15-FOOT DREDGE.



C. SCREEN FOR CONREY NO. 4 DREDGE. NOTE METHOD OF BRACING.

The grizzlies are usually protected by heavy cast-steel rails set about 18 inches apart, so that if the bucket line should break and the broken end go over the upper tumbler the grizzlies would not be damaged.

When it is necessary to dig the gravel to a depth of 50 feet or more a bucket idler (Pl. XL) should be placed to take a part of the weight of the return string of buckets. The idler is placed on the lower deck high enough so that the lips of a bucket will just clear the grizzly when the bucket is riding on the idler. This arrangement also affords a protection for the grizzly. A bucket idler is about 9 feet in diameter and the tread is shaped to conform to the edge of the lips of the buckets. The tread is shod with manganese-steel wearing plates. Idlers were formerly cast in four parts. The two spiders were cast separately and the rim was cast in two parts which were later bolted together. The shoulder of the rim against the ends of the spiders was also held with large bolts. The constant hammer of the buckets wore away the shoulder, when the bolts would not hold tight. At present the idler is cast in one piece, doing away with the difficulty mentioned.

The hubs of the idlers are long—in some idlers being more than 7 feet. Consequently it has been found difficult to keep the keyways from wearing loose, as the long hubs make difficult the use of taper keys. It is believed that this trouble will be overcome by splitting the hub the entire length and wedging it apart while boring it, and after the shaft has been put in place the wedges can be removed and the hub will bind on the key. The idler should not be pressed on the shaft. When repair of the idler is necessary it must be taken off the dredge through the well hole, and with the shaft in place it is too wide to go through the well hole. Hence the shaft must first be removed before the idler can be taken off the dredge. Similarly, in designing the idler, care must be taken to make it narrower than the well hole. In removing the idler it is lashed to the lower run of the buckets and the bucket line is revolved until the idler is landed on the bank.

SCREEN AND SCREEN DRIVE.

As the shaking screen is no longer used except in isolated dredging, only the revolving screen (Pl. XLI) is discussed here.

The size of the screen to be used depends upon the capacity of the buckets, the proportion of fine material, and the size of material that goes over the tables. The following table shows the sizes of screens with ⅜-inch or ½-inch perforations most commonly used for buckets of different capacities.

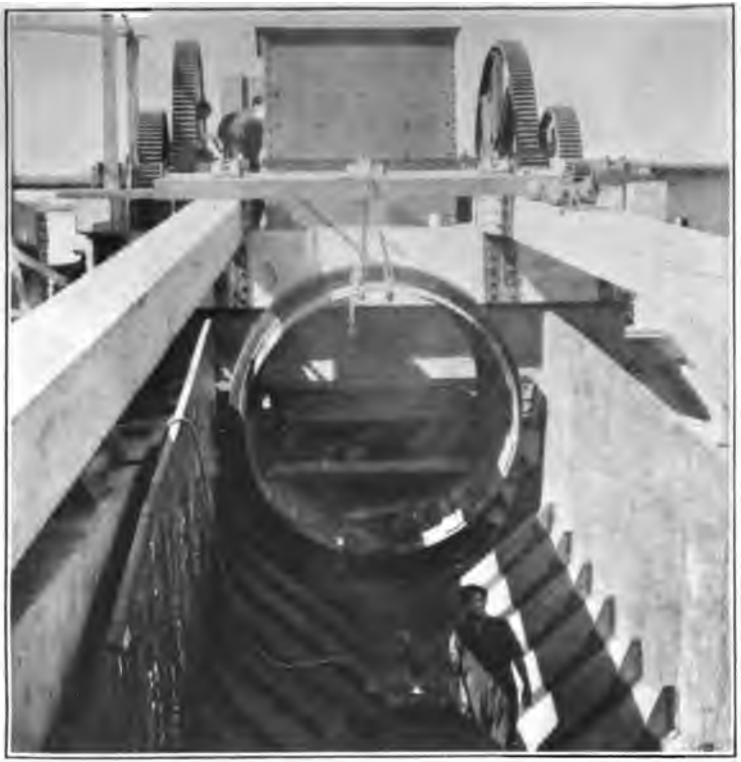
Sizes of screens used with different sizes of buckets.

Size of bucket.	Diameter of screen.	Total length of perforated plates.	Horsepower of drive motor.
Cubic feet.	*Feet.*	*Feet.*	
3	4	15	20
5	5	20	25
7	6	25	35
9	7	30	45
16	9	38	75

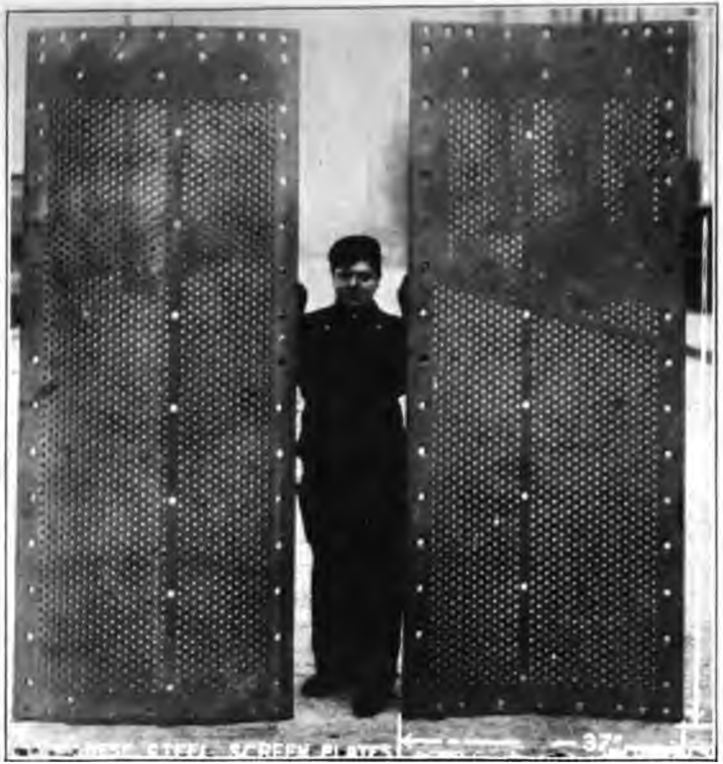
Screens are usually cylindrical, although some step screens are used. Each type has its advantages. Some of the advantages of the step screen are: (1) Weight is about 20 per cent less for equal capacity; (2) with the drive at the head end the weight is concentrated where it is needed for traction; (3) the same effect is obtained as when obstruction segments are used, but the drop with its attendant wear is eliminated; (4) telescopic screens allow the upper tumbler to be placed lower than with the cylindrical screens, as the tables at the rear end of the dredge can be placed higher. The advantages of the cylindrical screen are: (1) All of the screen plates are interchangeable and therefore a smaller number need be carried in stock; (2) it is less complicated; (3) it can be made more rigid; (4) a smaller proportion of the material goes on the upper tables, which is an advantage as with either type the upper tables have more than their share of material.

The screen structure should be designed so that it will take the torsional stress of the drive and also the bending movement without depending on the screen plates. This is sometimes accomplished by riveting braces between the longitudinal angles (Pl. XLI, *C*), but a simpler screen is obtained by so casting brackets on the tread rings that fit around the angles or bars as to take the shear from the rivets or bolts. Another method that accomplishes the desired result is to use sufficiently large gusset plates between the angles and the shell plates. All angles and plates should be held together with fitted bolts and spring washers instead of rivets, as the latter work loose too rapidly.

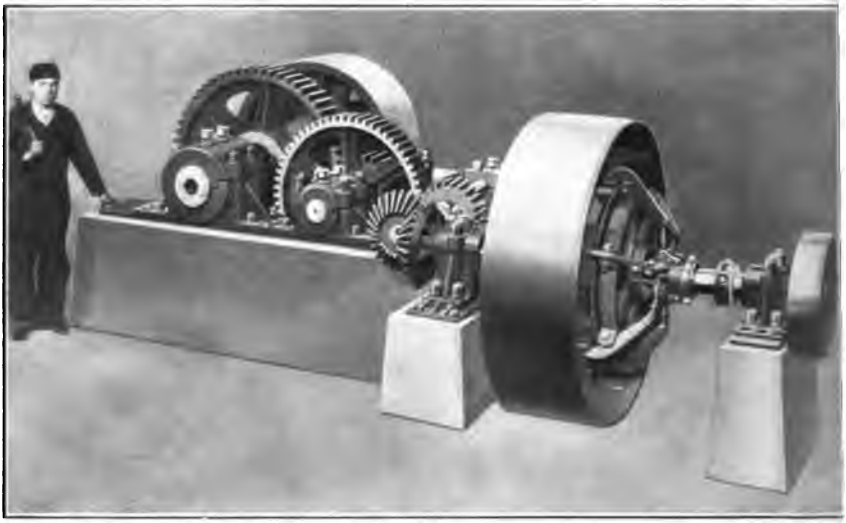
Screen plates are made either of high-carbon rolled plates with the holes drilled or of cast manganese steel. The holes should taper about one-eighth inch to every one-half inch of thickness of plate, and the small end should be placed inside the screen so that the holes will not clog. Longitudinal bars should be placed about 9 inches apart to carry the material with the screen as it revolves, thus spreading the gravel over a larger area. The bars cause the falling material to wear the screen plate more rapidly between the bars than directly under them. After such wear has developed the bars should be moved to cover the worn parts and expose the unworn parts.



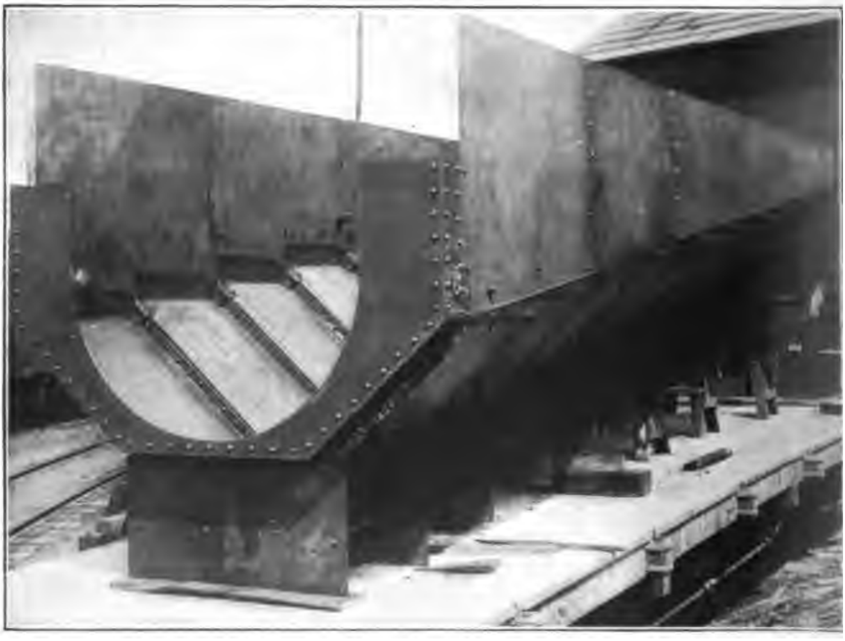
A. HOUSING AND DISTRIBUTOR FROM ABOVE, YUBA NO. 13 DREDGE.



B. MANGANESE-STEEL SCREEN PLATES, 90 INCHES LONG, 37 INCHES ACROSS, WITH 54-INCH RADIUS.



A. SCREEN DRIVE, SINGLE-ROLLER TYPE.



B. SCREEN HOUSING AND DISTRIBUTOR.

An obstruction segment is usually placed at the end of each screen plate. These segments are to keep the material from rushing through the screen too rapidly. The same result would be accomplished by reducing the pitch of the screen. One disadvantage of the segments is that the material is banked behind them and an uneven distribution along the screen results. Another objection is that there is excessive wear where the gravel drops over the segment. It is advisable to place one circle of bars at the upper end of the screen to retard the flow of the gravel as fed from the spout, also a set at the lower end to arrest the rapid flow of water to the conveyor belt.

The liners for the upper and lower shell plates should be bars about 6 inches wide by 1 inch thick rather than circular plates, as the former are more economical and easier to handle. Cast bars are superior as they can be made thicker where the greater wear comes.

For washing the gravel there are two methods, both of which have their advantages. In one method a large perforated pipe running the length of the screen is used. The diameter of perforations formerly used was about one-half inch, but this size proved too small as the holes often became clogged; now nozzles of 1 inch to 2 inches diameter spaced every 12 or 16 inches are used. It is impossible to support the pipe except at the two ends, and occasionally a pipe breaks. The other method is to place at each end of the screen a header with 2-inch nozzles, which direct the streams to the center of the screen. The advantage of the header is that it leaves the screen free from obstruction and more easily accessible. The nature of the gravel should determine which method of washing should be used.

The latest practice in driving the screen is to drive from the lower end principally because the driving machinery is then more accessible. The upper-end drive (Pl. XXXIX, *B*, and XLII, *A*) has, however, the advantage that the greater weight is concentrated at the upper end, thus producing better traction and less torsional strain.

It was formerly the custom to drive the screen from two rollers set at 45°. However, as the rollers did not wear down equally, there was a resultant slip, causing rapid wear. The newest dredges are designed with a single-drive roller (Pl. XLIII, *A*), placed directly under the center line of the dredge, or rather a few inches to one side of the center line. Plates XLII, *A*, and XLIII, *B* show two views of screen housing.

The idler rollers, or those places under the screen at the end opposite from the drive, are placed on a line 45° from the center line. By "cutting" these idler rollers or, in other words, shifting them out of line one way or the other the screen is kept from moving too far up or down. Unless these rollers are adjusted properly the thrust roller, which is usually placed at the lower end of the screen, is soon worn out. It is important that the tread rings be kept free of sand and grit. This is usually accomplished by splash rings, but it is well to have in addition a spray of water on the tread rings. Drive rollers and thrust rollers should be of special steel, either nickel-chrome or manganese. Tread rings should be of nickel-chrome.

Screens should have a peripheral speed of 150 feet per minute. The structure on which the screen is supported should be of steel as it is impossible to keep the bearings in line with a wooden structure.

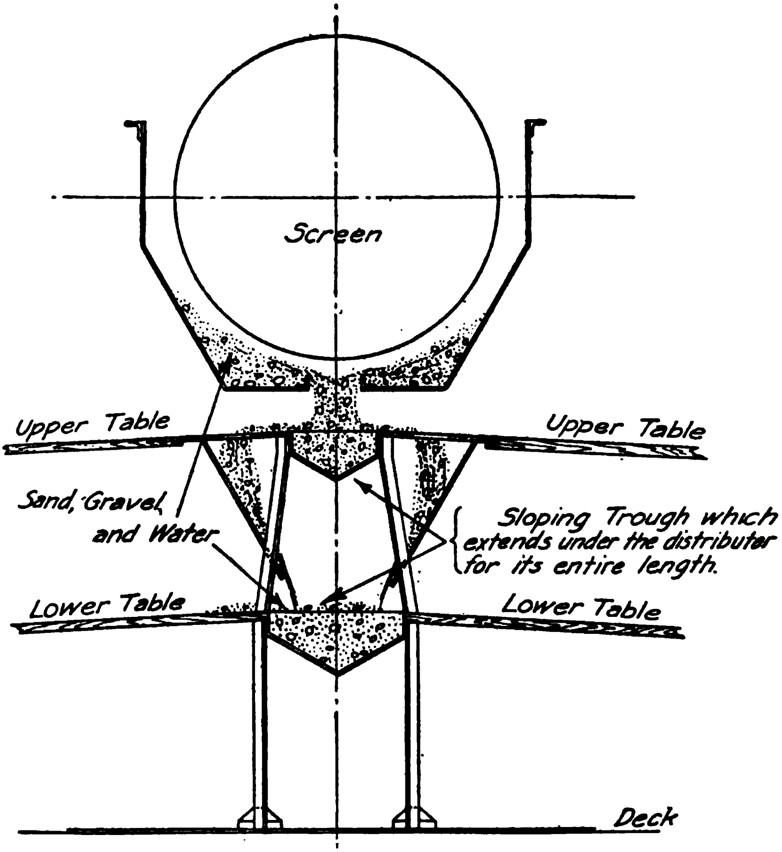
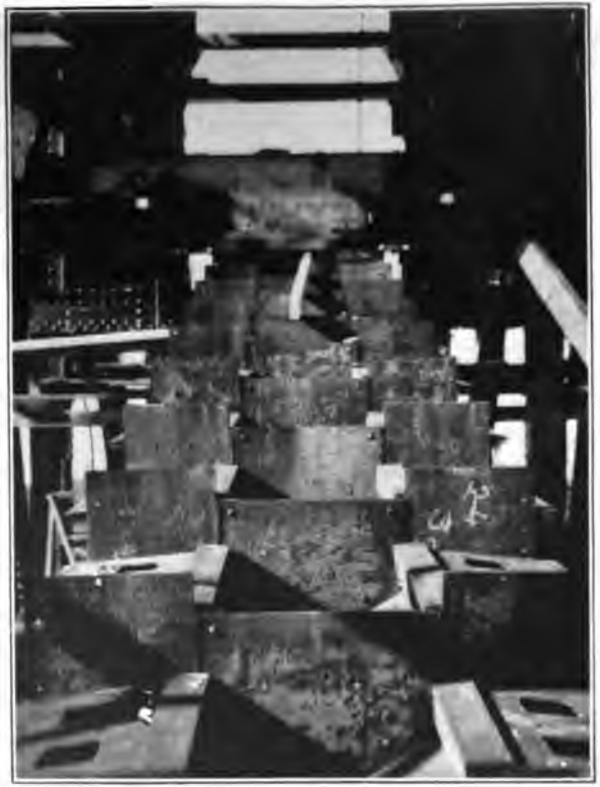
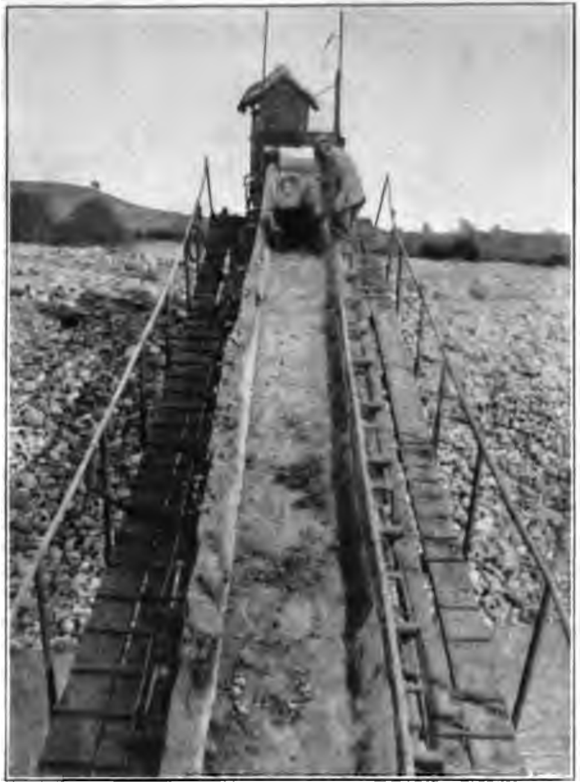


FIGURE 9.—Cross section of new type of distributor.

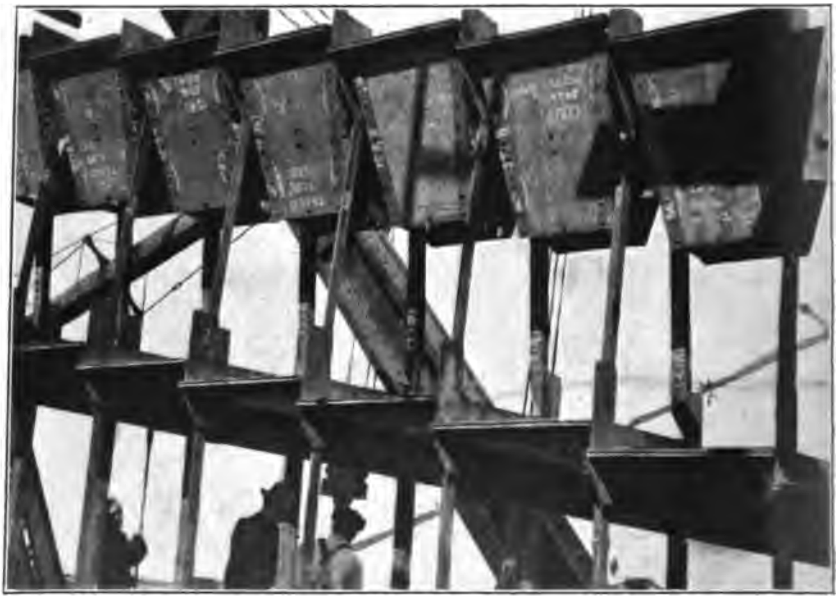
It is advisable to drive the screen with a variable-speed motor to eliminate as much jar as possible when starting. A silent chain drive is better than a belt drive. It is well to be able to reverse the direction of rotation of the screen as the wearing parts will last considerably longer, all holes and bars wearing according to the direction of rotation.



A. TOP OF DISTRIBUTOR.



B. FLAT TYPE OF STACKER BELT; NO TROUGHING IDLERS ARE USED.



A. SIDE VIEW OF DISTRIBUTOR, YUBA DREDGE. TABLES NOT IN PLACE.

B. SIDE VIEW OF DISTRIBUTOR. TABLES IN PLACE.

DISTRIBUTOR.

The object of the distributor is to catch the fine material and the water as they come from the screens and distribute them to the different sluices. A large proportion of the fine material is discharged at the upper end of the screen, and it is necessary to arrange the outlets of the distributor so that this material will be carried toward the lower end and in equal quantity to each of the tables.

Most of the material from revolving screens is discharged to one side of the center line, so that more goes to the tables on one side of the dredge than to those on the other side, unless the distributor is designed to equalize the amount fed to the tables.

Several types of distributors have been tried, but the one shown in Plate XLIV, *A*, was the standard type for some years. The bottom of this distributor was made of short eyebeams placed end to end, each about 6 inches lower than the one directly above. An opening was placed on each side with a sliding gate to regulate the feed. It was found impossible to get uniform feed to the different tables with this type, and another very serious objection was that it was very difficult to replace worn-out plates.

A new distributor (fig. 9) evolved by the Yuba Consolidated Goldfields has proved satisfactory. The bottom is separate from the sides, thus giving greater accessibility for repairs. It is a sloping trough with partitions placed every 2 or 3 feet. The trough fills with sand, which protects the steel bottom from wear. The material discharges over the sides of the trough onto the tables. When there are two decks of tables, openings are left at the head of the upper bank, and approximately half of the material drops to a similar distributing trough below. Plate XLV and figure 9 illustrate this feature.

STACKER.

At present the belt-conveyor tailing stacker is in general use, having practically superseded the bucket conveyor of the early dredges. There are not more than two or three dredges in the United States with bucket conveyors and the operators of these few dredges are the strongest advocates of the belt type. While this method of handling material was being developed by dredge operators, other industrial plants such as ore mills and cement works were proving the the belt-conveyor method of elevating and conveying broken rock or like substances to be much superior to any bucket system when the conditions were similar to those in dredging.

The belt stacker has the following advantages: (1) Lower cost of operation, as it does away with the innumerable pins and joints that are a source of wear and breakage, the breaking and bending of buckets, etc.; (2) less lost time owing to breakage; (3) less vibration, smoother running, and therefore less wear on the motor and the drive; (4) lighter weight; (5) with the belt stacker it is possible to dump the material at such height and distance behind the dredge as not to interfere with operations.

The stacking ladder consists of two parallel angle-iron trusses tied together at the bottom with braces, at the top with a continuous deck plate, and internally with cross braces. At the lower end the ladder is pivoted on a shaft placed as near as practicable to the receiving point.

It is important to have the drop from the screen chute to the conveyor placed as low as possible and it is necessary to place the pivot point near the receiving point so that the drop will not change as the ladder is raised and lowered.

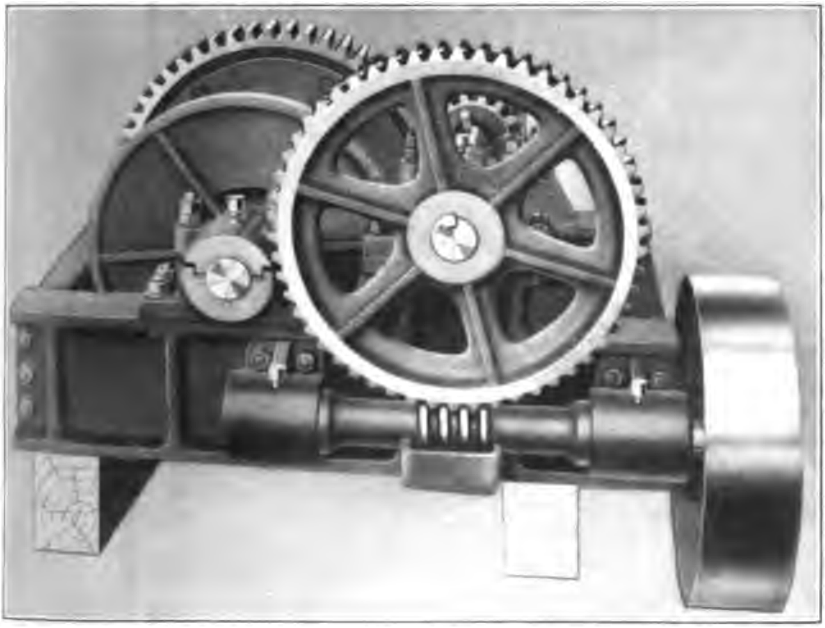
The outer end is supported and raised and lowered by wire rope from the stern gantry. A wire-rope sling, usually placed on the stacker, is arranged so that double ropes support the outer end and a single rope supports the center of the stacker ladder. It is best to do away as much as possible with steel rods and bars and to use wire rope, as the latter does not crystallize so rapidly and a careful examination of the rope will determine whether the strains are too great, but a steel rod gives no indication of weakness before an accident. Several stackers have dropped owing to the breaking of bars, but when wire-rope slings have been substituted the trouble has been eliminated.

The stacker is hoisted by a small winch (Pl. XLVI, *A*) and the drive is through a worm wheel. No brakes need be used to hold the stacker in place, as the teeth are so designed that the stacker will not unwind the drum of its own weight; the motor must be operated to change the position of the stacker.

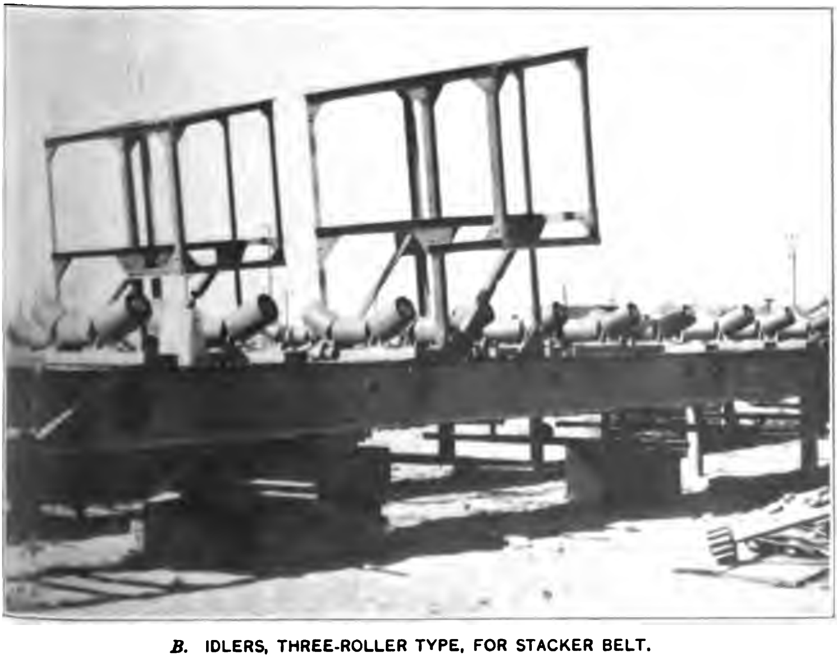
A recent innovation in the arrangement for distribution of the tailing from a dredge is shown in No. 16 dredge of the Yuba Goldfields. This dredge (Pl. XLVII) was designed with double-tailing stackers for reconstructing the channel of the Yuba River coincident with the recovery of gold from the gravel. With the exception of the stackers this dredge is practically the same as Yuba No. 15. The stackers are 225 feet long and the distance between the outer ends of the stackers is 416 feet.

BELTS.

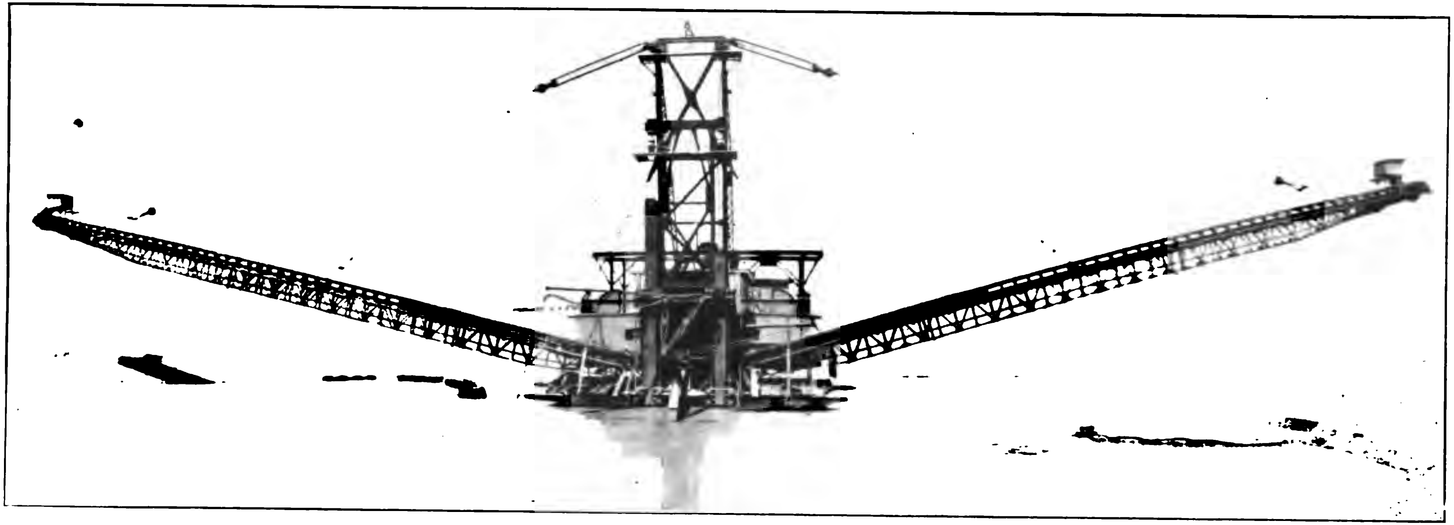
Practically all belting manufacturers have the duck used in their belting woven to specifications, some preferring a tight weave and others contending that a softer weave gives more flexibility and better wearing properties. One company states that the main body of the belt it manufactures is composed of duck of the best sea island cotton, and that experiments with different weights have shown that



A. STACKER WINCH, DOUBLED GEARED, FOR 9-FOOT DREDGE.



B. IDLERS, THREE-ROLLER TYPE, FOR STACKER BELT.



YUBA NO. 16 DREDGE, WITH DOUBLE TAILING STACKER.

a duck weighing approximately 32 ounces to the yard of 42-inch material gives the best satisfaction. Other firms use different weights of material.

Belts consist of three to eight plies of canvas, depending on the width and the length of the belt, covered on the carrying side and edges with one-sixteenth inch or one-eighth inch of rubber and on the other side with one thirty-second inch to one-sixteenth inch of rubber. The greatest wear occurs along a strip about 8 inches wide in the middle of the belt and in the past this part of the belt has often worn out while the remainder has remained in good condition.

Several methods of reinforcing the middle have been tried and many are in use. Figure 10 show the types now in most general use. Each of these has advantages and disadvantages. In figure 10, A shows the original construction, and B the first method of overcoming wear. Another serious difficulty experienced is that when the rubber is worn or punctured the water gets into the canvas which rapidly deteriorates, especially in cold climates where the moisture freezes and forces apart the canvas. The belt shown at C is known as the "Cleaveland construction." If the outer rubber is cut the damage does not reach to the inner layer of rubber and the efficiancy of the belt is preserved.

When a belt is not running true, owing to the idlers not being set correctly, the edges wear rapidly, hence it is well to continue the thick rubber well around the edges as shown at D.

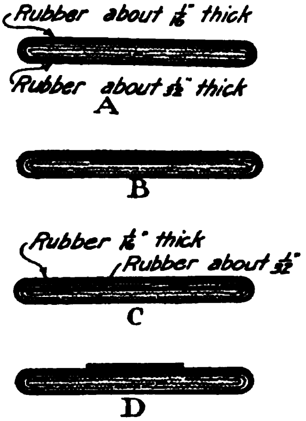


FIGURE 10.—Cross sections of four types of conveyor belts in general use. A, original type; B, first method of reinforcing center; C, "Cleaveland" construction; D, another method of reinforcement.

During the early years of gold dredging in this country, considerable difficulty was found in procuring a conveyor belt that would wear satisfactorily. Many different materials were tried, and some "patent" belts lasted only a few hours when subjected to the severe wear on a gold dredge. At first it seemed difficult for the manufacturers to turn out a uniform product and belts bought from the same firm and subjected to the same wear would occasionally give entirely different service. In those days it was customary to buy belts according to specifications, stating the number of plies and weight of duck used, the friction pull and the thickness of the rubber covering, etc. Gradually competition among the various belt-making firms caused a life guaranty of the belts to be given and it is upon this basis that belts are now purchased. For dredges that have been at work for some time, an average is taken of the life of belts previously used and in new fields some time basis is generally agreed upon by the operator and the saleman. This method has proved satisfactory to the dredge operators. When the life of the belt exceeds the guaranty a bonus is sometimes paid for the extra service obtained though this practice is not universal. Occasionally payment has been arranged on an operating-hour basis and again on the number of yards handled during the life of the belt. On one 6-foot dredge payment was made on the basis of 20 cents an hour for total operating hours.

The life of standard conveyor belts in California varies from 9 to 14 months. On one dredge three belts gave unusually uniform wear of 458, 468, and 485 days, respectively. A fourth belt lasted 256 days only and adjustment for this short life was made by the manufacturer.

The size of the conveyor belt is determined both by the quantity of material it has to carry and by the size of the bowlders. Therefore, it does not vary only with the size of the bucket. The following table gives the width of belt usually used with buckets of different sizes.

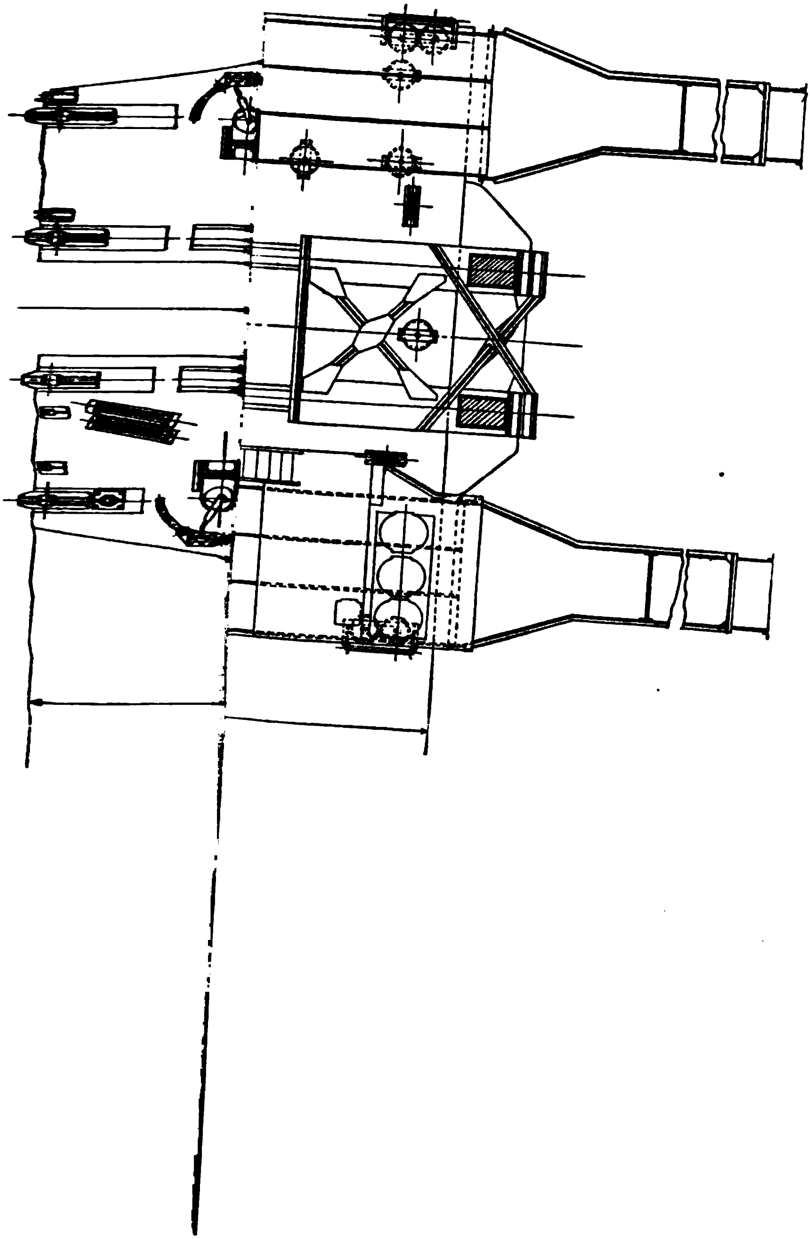
Width of conveyor belt used with different sizes of buckets.

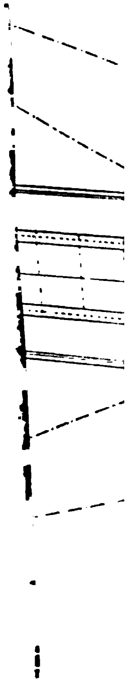
Capacity of bucket, cubic feet.	Width of conveyor belt, inches.
3	24
4	28
5	32
6	32
7	32
9	36
13	36
16	42

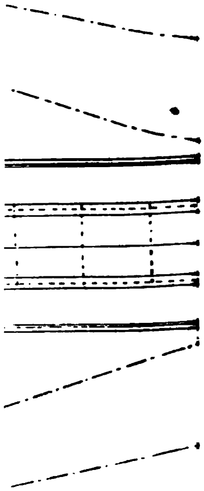
When the gravel is of an unusual character the size of belt is determined by the size of the largest bowlders, as the chute feeding the belt should be narrower than the belt. The widths of belts corresponding to the sizes of gravel are as follows:

Width of belt, inches.	Size of largest pieces.
20	7-inch cubes.
24	9-inch cubes.
30	12-inch cubes.
36	15-inch cubes.
42	18-inch cubes.

When the size of the largest pieces determines the width of the belt, the latter should be run at the lowest possible speed that will carry the material, so as to insure minimum wear. When the tailings are small it is best to use a small belt and to run it at a high speed, not exceeding 400 feet a minute.







The less the angle of incline of the stacker, the greater the capacity of a belt, according to the following table:

Relation of angle of incline to capacity of conveyor belt.

Angle of incline, degrees.	Per cent of capacity of horizontal conveyor.
Horizontal	100
6	99
8	98
12	97
15	96
16	95
18	93
20	90
22	86

This loss in carrying capacity is caused by the material slipping on the belt; similarly the wear on the belt increases with the angle of incline. The incline should never be more than 22°. The above table is for material comparatively dry, and the loss of capacity increases rapidly with the increase of water. It is therefore important that as little water as possible be allowed to go through the screen to the stacker, and the screen should be arranged to take care of this.

Plate XLIV, *B* (p. 76), shows a stacker belt in use.

The following table gives the number of plies for the different widths of belt and the diameter of the stacker pulleys that should be used. It will be noted that the sizes given are larger than those generally used. A larger pulley will not pull more than a small one, as the pulling power depends only on the arc of contact. However, the pull is increased by facing the driving pulley with rubber or leather.

Number of plies required for different widths of stacker pulleys.

Width of belt.	Number of plies.	Diameter of upper drum.	Diameter of lower drum.
Inches.		*Inches.*	*Inches.*
24	5	30	26
28	5	34	28
32	6	38	32
36	7	42	36
42	8	48	42

The upper and lower drums should be large enough to reduce to a mimimum the strain on the fibers of the belt. The thicker the belt the larger the drums should be. The longer the belt, the greater should be the number of plies, but the number of plies given in the above table will be sufficient for any length of stacker used on gold dredges.

The snubbing rollers should be eliminated whenever p should be made as large as practicable when their use is The idlers should be placed near enough together so tha not be too much sagging of the belt between idlers, as thi ternal wear (see Pl. XLVI, *B*, p. 78). For small belts the id erally placed about 4 feet between centers, and for large be placed 3½ feet apart. The idlers should not be supported planks, as is the custom, as these planks rot and do n bearings securely. They have to be replaced in time an cause the spilled gravel to collect around the idlers and rapidly, sometimes keeping the pulleys from revolving, belt has to slide over them.

The trouble mentioned could be avoided by doing aw flat deck plate that covers the stacker and substitutin plate, high at the center, which would allow the spill to r pond. This arrangement would eliminate much labor necessary in scraping off accumulated material.

The return idlers underneath the stacker are made s instead of using a number of short pulleys mounted on a done in the usual belt-conveyor installation, one long pu be used. The short pulleys present a number of edges likely to cut into the belt. The return idlers are placed 10 feet apart.

GUIDE ROLLERS.

Guide rollers should never be used to keep the belt fro off as their use injures the belt. If the head and tail pul line and the troughing idlers are set correctly the belt w the center.

Rollers with a carrying surface that approximates a sem generally accepted as the best type, although some operat the spool-shaped rollers, making a flatter belt. The spo roller has a long center roll with larger rolls on each end w up the belt at the edges. The capacity of a belt running shaped rolls is not so large for a given width, but such rolls advantage of distributing the wear over a greater proporti width of belt than do the troughing type of rollers.

The stacker belt (Pl. XLIV, *B*, p. 76) should always be dri the discharge end. Material conveyed on an inclined belt alw somewhat at each troughing roller. The tighter the belt th slips. When the belt is driven from the lower pulley the side of the belt is the slack side and as a consequence the be faster and the carrying capacity is decreased.

Formerly the driving motor was directly connected through a pinion to the countershaft. Even with a cut pinion the noise and wear were excessive, as maintanance of correct alignment was impossible. The next method of applying power from the motor was by belt, and most conveyors are at present so driven. The disadvantage of the belt drive is that the necessary exposure to the weather seriously affects the belt. The stacker belts on the latest dredges are driven by a silent chain which combines the advantages of the other two methods and does away with the disadvantages. The loss in time is much reduced over the other two methods. There is practically no loss of time from the chain drive if the chain is kept well lubricated and is replaced when worn.

WIRE ROPES.

The expense of upkeep of the wire ropes is necessarily large on a dredge but on many dredges much expense can be obviated by the selection of sheaves of proper size. The rapid wear of ropes running on small sheaves is shown by the difference in the life of the bow swing lines on the same dredge. The port bow line usually runs over two or three more sheaves than the starboard line as the winch is placed on the starboard side of the dredge, and the port line often lasts only one-half or one-third as long as the starboard line.

The diameter of the running sheave, if possible, should be at least forty times the diameter of the rope (Pl. XXXVIII, *C*, p. 72); thus, for a 1-inch rope a sheave having a diameter of 40 inches should be used. It is of course impracticable with some equipment to carry out this proportion. For example, shore sheaves of such a size would be too heavy for efficient handling. However, on many dredges on which large lines and small sheaves are used the life of rope would be lengthened if the rope were smaller. This factor is less important in standing ropes, or those used as guys. The strength of a rope where it bends around the sheave diminishes directly as the size of the sheave is decreased, that is, the bending stress around a 12-inch sheave is twice as much as around a 24-inch sheave

There is not much difference of opinion among operators regarding the most desirable types of rope to perform the different operations but they do differ as to what quality is advisable—when the most expensive ropes should be used, and when less expensive lines are economical. The following table shows average practice as to ropes, "plough steel" designating the best rope made by the different manufacturers.

Data on average practice as to wire ropes used on dredges.

Where used.	Grade.	Number of strands.	Number of wires.
Ladder hoist	Plough steel	6	19
Bow swing	do	6	19
Stern swing	Crucible cast steel	6	19
Spud hoist	do	6	19
Stacker hoist	do	6	19
Stacker sling	do	6	7
Bow gantry guys	Galvanized bridge cable.	6	(a)

[a] Wire center.

Almost as important as the selection of the proper kind of rope is the care given it in use. A proper lubricant, of which there are a number on the market, should be freely employed and will increase the life of the rope.

Manganese sheaves are easier on the ropes as they wear smoothly and in the grooves there is an absence of the ridges so common on cast-iron sheaves after they have become worn. Each sheave should be ground in the bore and a bronze or an iron bushing should be pressed in. Cast-steel sheaves are preferable if they are not subjected to wear.

The use of a socket on a rope in a permanent position is good practice provided the rope is properly put in place. The end of the wire should be spread out and washed with muriatic acid, then placed in the socket and zinc or spelter poured in. Lead or babbit should not be used nor should the wire be bent back. The object of using zinc or spelter is that either fuses to the wire, whereas lead depends upon the friction between it and the wires for holding power. Failure to fasten a rope properly in the socket has been the cause of several serious accidents.

Different makes of wire rope are used for dredging and a number of the rope manufacturers supply a special brand for the purpose, 8,000 to 11,000 feet of rope being required for one of the large dredges. The following list gives the ropes furnished by one company for use on the Conrey No. 4 dredge, a total of more than 10,000 feet being required.

'#
075'
19'
17
about
when
while

moment of D.F. = [illegible]

Total moment = 681000'#

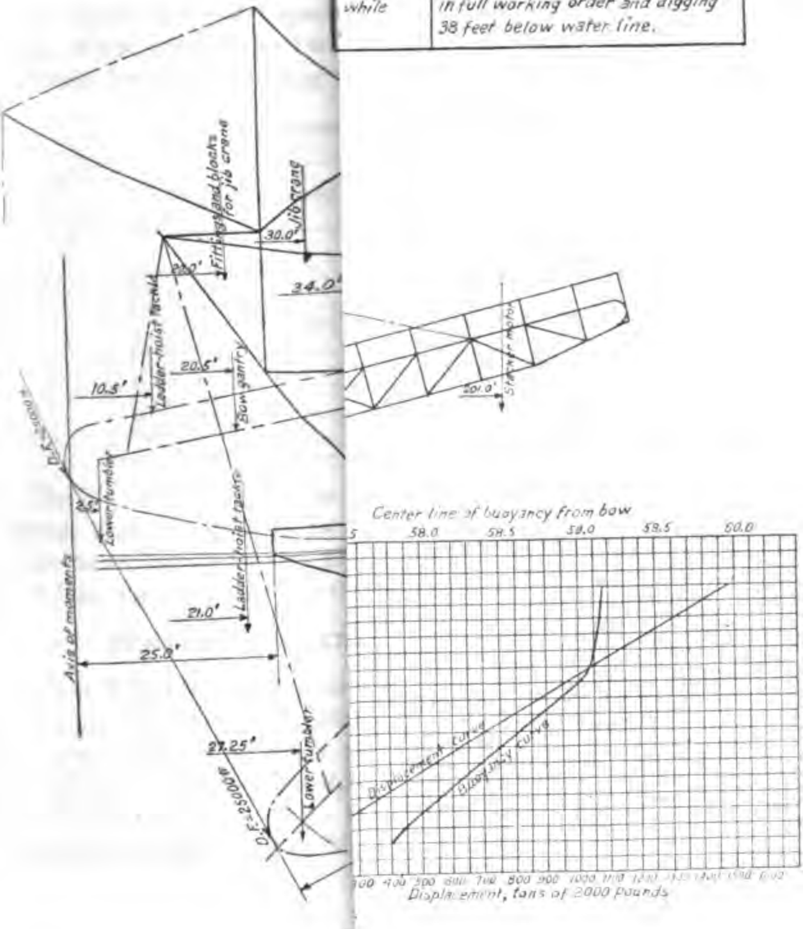
$t\text{(at bow)} = \frac{681000 \times 59.6}{1726955 \times 157.7} = .148$

$t\text{(at stern)} = .148 \times \frac{50.4}{59.6} = .125$

Draft (at bow) = 5.40 − .15 = 5.25

Draft (at stern) = 5.40 + .13 = 5.53

Note: The dredge is floating practically on even keel when in full working order and digging 38 feet below water line.



Sizes and uses of ropes used on Conrey No. 4 dredge.

Number of pieces.	Dimensions of rope.[a]	Where used.
2	2¼ by 19 by 195	Steam gantry guys.
2	2 by 19 by 375	Bow gantry guys.
1	2 by 19 by 200	Stacker suspension.
1	2 by 19 by 220	Do.
2	1½ by 19 by 1,100	Ladder hoist.
1	1⅜ by 19 by 46	Adjusting rope for stacker guys.
1	1¼ by 19 by 500	Straps for side swing lines.
1	1¼ by 19 by 650	Starboard speed line.
1	1¼ by 19 by 655	Port speed lines.
2	1¼ by 19 by 600	Bow swing lines.
1	1 by 19 by 440	Stacker hoist rope.
2	1 by 19 by 400	Stern lines.
1	1 by 19 by 420	Stacker hoist.
1	¾ by 19 by 190	Gang-plank hoist.
1	¾ by 19 by 510	Side guys for stacker.
4	¾ by 19 by 80	Sluice-pan guys.
4	¾ by 19 by 60	Do.
8	¾ by 19 by 43	Do.

[a] The first figures indicate the diameter in inches, the second figures the number of wires to the strand, and the last figures the length in feet.

As the lengths of ropes vary on different dredges the following table of the most important ropes on Yuba dredges is given, a total of about 10,000 feet being used for all purposes.

Data regarding most important ropes used on Yuba dredges.

Where rope is used.	Diameter of rope.	Number required.	Length of each.	Total length.
	Inches.		*Feet.*	*Feet.*
Bow gantry guys	2¼	2	239	478
Stern gantry guys	2	2	135	270
Tail-sluice support	¾	2	250	500
Foot-bridge suspension	⅝	2	250	500
Stacker sling	2	1	172	172
Stacker guys	1	1	120	240
Ladder hoist	1½	2	1,200	2,400
Stacker hoist	1	1	600	600
Speed hoist	1	2	600	1,200
Bow swing lines	1½	2	700	1,400
Stern swing lines	1	2	500	1,000

The life of rope depends on the quality, the use to which it is subjected, and the care it is given. The best brands of ladder-hoist ropes have lasted two years or over, whereas some rope in use for side lines has worn out in 90 days.

GENERAL DETAILS OF DREDGE CONSTRUCTION.

Plate XLVIII (p. 80) shows a plan and Plate XLIX an elevation of equipment on a 7½-foot dredge, showing the general arrangement of parts. Plate L shows a plan view and Plate LI a side view of the assembly of an 8-foot dredge. Plate LII shows a flotation diagram for a 5½-foot dredge. The table following presents general statistics regarding dredges used in Montana:

General statistics concerning dredges used in Montana.

Item.	Steam dredges.			Electric dredges.			
	Maggie Gibson.	Conrey No. 1.	Conrey No. 2.	Conrey No. 1.	Conrey No. 2.	Conrey No. 3.	Conrey No. 4.
Power used	Steam	Steam	Steam	Electricity	Electricity	Electricity	Electricity.
Date built	1896	1899	1901	1908	1908	1906	1911.
Type	Double-lift sluice	Single-lift sluice	Single-lift sluice	Single-lift table stacker.	Single-lift table stacker.	Single-lift table stacker.	Single-lift table stacker.
Anchorage	(a)	Lines	Lines	Spud	Spud	Lines	Spud.
Average monthly yardage worked, cubic yards.	27,000	41,360	52,520	96,270	62,709	82,415	300,000.
Hull dimensions:							
Draft	5 feet 3 inches	5 feet 4 inches	5 feet 9 inches	5 feet	5 feet 6 inches	6 feet	9 feet.
Length	110 feet	100 feet	110 feet	96 feet	102 feet	130 feet	150 feet.
Width	35 feet	44 feet	44 feet	44 feet	44 feet	48 feet	58 feet.
Depth	7 feet 6 inches	7 feet 6 inches	7 feet 6 inches	9 feet	9 feet	7 feet 11 inches	13 feet.
Pontoon dimensions:							
Length	30 feet	38 feet 6 inches	40 feet	None	None	50 feet	None.
Width	24 feet	35 feet	29 feet			40 feet	
Depth	4 feet	3 feet	3 feet 10 inches			3 feet 11 inches	
Stacker dimensions:							
Length	None	None	None	90 feet	85 feet	None	130 feet.
Width of belt				34 inches	34 inches		4 feet.
Motor drive							
Tailing height				Upper end 30 feet	Upper end 30 feet		Upper end 55 feet.
Dimensions of steel spuds:							
Length, feet	None	None	None	48	54	None	80.
Width, feet				3	3		4½.
Thickness, feet				2	2		3.
Weight, pounds				26,500	29,000		88,000.
Number				1	1		2.
Dimensions of wooden spuds:							
Length, feet	30	None	None	48	54	None	None.
Width, feet	1½			3	3		
Thickness, feet	4			2	2		
Number	2			1	1		
Horsepower:							
Bucket chain	45	60	73	100	100	150	550.
Trommel	5	6	8	30	30	10	100.
Pumps	40	79	147	160	160	175	285.
Winches	10	12	4	20	20	115	250.
Miscellaneous	None	19	16	70	70	10	50.
Total horsepower	100	176	248	380	380	460	1,235.
Ladder data:							
Length, feet	70	82	95	69	80	100½ [b]	116.
Type	Queen truss	Queen truss	Lattice girder	Lattice girder	Lattice girder	Lattice girder	Plate girder.

Depth of digging below water, feet.	29	35	40	30	35	45	55.
Dimensions of tumblers:							
Upper tumbler—							
Distance above deck	12 feet 7 inches	24 feet 6 inches	29 feet 3 inches	25 feet 9 inches	29 feet 3 inches	31 feet	35 feet 6 inches.
Weight, pounds	2,000	2,000	4,050	8,970	8,970	8,970	16,000 to 25,300.
Number of sides	5	5	5	6	6	6	6.
Shaft diameter, inches	9	11	12	16	16	16	25.
Lower tumbler—							
Weight, pounds	3,000	2,500 to 3,075	4,600	5,800	5,800	5,800	18,000 to 19,800.
Number of sides	6	6	6	6 to round	6 to round	6 to round	6 to round.
Shaft diameter, inches	6	6 to 8½	10	13¼	13¼	13¼	15⅜.
Bucket-chain data:							
Type	Open link	Open link	Open link	Close connected	Close connected	Open link, later changed to close connected.	Close connected.
Drive	Sprocket	Sprocket	Sprocket	Belt	Belt	Gear	Gear.
Weight, pounds	40,000	104,000	125,000	169,000	225,000	297,000	357,500 to 415,300
Capacity of buckets, cubic feet	5	5 to 7½	10	7½	7½	13 open, 9½ closed	16 to 17.
Number in chain	35	41	41	60	80	43 open, 80 closed	80.
Weight, pounds	750	750 to 1,500	1,850	2,594	2,594	2,900	4,468 to 5,191.
Pitch, inches	26	26 to 29⅜	31$\frac{5}{16}$	32¾	32¾	32¾	40.
Buckets dumped per minute	7	5 to 11	5 to 11	16	15	15	18 to 22.
Type	2-web	2-web	3-web	3-web	2-web	2-web	2-web.
Material	Steel	Steel	Steel	(c)	(c)	(c)	(c).
Hood and base, number of pieces	2	2	2	2	(d)	(d)	1.
Links—							
Weight, pounds	300	770	900	None		1,250	None.
Pitch, inches	26	29⅜	31$\frac{5}{16}$			32¾	
Number in chain	35	40	40		None	43	
Material	Steel	Steel	Steel			Steel	
Type	2-web	2-web	3-web			3-web	
Pins—							
Diameter, inches	3	3 to 4⅝	4⅞	4⅝	5	5	7 to 8.
Weight, pounds	50	50 to 130	155	168	184	184	520 to 495.
Material	Steel	Steel	Steel	Steel	(c)	(c)	(c).
Type	Double lug	Lock pin	Lock pin	Single lug	Single lug	(e)	Single lug.
Pin bushings—							
Type	Half circle	Full circle	Full circle	Full circle	Full circle	Full circle	Full circle.
Material	Manganese	Manganese	Manganese	Manganese	Manganese	Manganese	Manganese.
Trommel data:							
Length	14 feet	16 feet	15 feet 2 inches	35 feet	35 feet	18 feet	48 feet 6 inches.
Diameter, inches	48	46	48	51 to 72	51 to 72	61	96.
Type	Straight cylinder	Straight cylinder	Straight cylinder	Stepped cylinder	Stepped cylinder	Straight cylinder	Straight cylinder.
Drive	Sprocket	Sprocket	Sprocket and bevel gear.	Belt	Belt	Belt	(f).

a Originally spud; changed to lines.
b Changed to 116 feet.
c Steel; later changed to manganese.
d Steel buckets in two pieces; manganese in one.
e Lock pin; later changed to single lug.
f Gear; later changed to belt.

General statistics concerning dredges used in Montana—Continued.

Item.	Steam dredges.			Electric dredges.			
	Maggie Gibson.	Conrey No. 1.	Conrey No. 2.	Conrey No. 1.	Conrey No. 2.	Conrey No. 3.	Conrey No. 4.
Trommel data—Continued.							
Plates—							
Thickness, inches	3/8	3/4	3/4	5/8	5/8	5/8	1.
Material	Steel	Steel	Steel	(a)	(a)	(a)	(a).
Perforations	4 by 4 inches	4 by 4 inches	4 by 4 inches	½ to ¾ inch	½ to ¾ inch	4½ by 6 inches	½ to ¾ inch.
Dimensions of tables:							
Grade, per cent	None	None	None	12½	12½	12	12½.
Number				20	20	10	24.
Width, inches				30	30	30	30.
Length, feet				18	18	12	11 to 29¾.
Riffles (angles), inches				1½	1½	1½	1½.
Perforated plates						5/8-inch hole	
Pump data:							
Number	2	2	3	4	4	5	3.
Size and kind	4-inch water 12-inch gravel	10-inch water 14-inch water	10-inch water Two 14-inch water	3-inch water 8-inch water 12-inch water 6-inch sand	3-inch water 8-inch water 12-inch water 6-inch sand	3-inch water Two 10-inch water 12-inch water 14-inch water	4-inch water. 14-inch water. 16-inch water.
Make	Morris	Morris	McKay	Morris	Morris	Morris-Worthington.	Worthington.
Gallons per minute	4,000	6,500	8,100	6,700	6,700	12,500	12,000.
Main winch data:							
Length	12 feet	26 feet	26 feet	20 feet 1 inch	20 feet 1 inch	11 feet 10 inches	29 feet 10 inches.
Width	6 feet	16 feet	16 feet	4 feet 11 inches	4 feet 11 inches	8 feet 6 inches	7 feet 5 inches.
Diameter of drums, inches	16	16	16	16	16	30	24.
Number of drums	6	8	8	8	8	2	8.
Tail sluices:							
Grade, per cent	None	None	None	12½	12½	None	12½.
Number				2	2		12.
Length, feet				40	40		17 to 44.
Width, inches				48	48		48.
Riffles (angles), inches				1½	1½		1½.
Undercurrents:							
Grade, per cent	None	10	10	None	None	10	12½.
Number		1	1			2	12.
Length, feet		40	60			54	7½.
Width, inches		10	10			13	34.
Riffles—							
Material		Wood	Wood			Wood	Angles.
Size, inches		1 by 1½	1 by 1½			1 by 1½	1½.

Perforated plates:							
Width of slots, inches		1/16	1/16			1/16	½.*b*
Well save-all:							
Grade, per cent	None	8	8	8	8	8	8.
Number		1	1	1	1	1	1.
Length, feet		30	30	7½	7½	25⅛	18½.
Width, inches		12	12	18	18	18	18.
Upper flume:							
Grade, per cent	None	5	5	None	None	6	None.
Riffles—							
Material		Wood	Wood			Angles	
Size, inches		1¼ by 2	1¼ by 2			2 by 2½	
Length		21 feet	22 feet			23 feet 10 inches	
Width, inches		42	50			62	
Lower flume:							
Grade, per cent	5	5	5	None	None	6	None.
Riffles—							
Materials	Wood	Wood	Wood			Angles	
Size, inches	2 by 4	2 by 4	2 by 4			2 by 2½	
Length, feet	103	120	120			135	
Width, inches	38	52	52			66	
Total gold-saving area, square feet	320	656	692	1,231	1,231	1,264	3,000.

a Steel; later changed to manganese.

b Holes instead of slots.

SPECIAL STEELS FOR DREDGE PARTS.

PROPERTIES OF ALLOYS OF STEEL.

The gold-dredging industry has played an important part in the development of the different alloys of steel on account of the exceptionally severe service imposed. On the earlier dredges cast iron entered largely into the different machine parts, and the wearing parts were made of carbon steel. Cast iron has now given way to cast steel and all parts subjected to wear and those subjected to severe strains are made of one of the alloys. Thus, in order to design an efficient modern dredge, a thorough knowledge of steel and its properties is necessary.

MANGANESE STEEL.

Manganese steel has been the most important agent in bringing gold dredges to their present high efficiency. Forged or rolled manganese steel is little used on a dredge but there are many places where it could be used to advantage, especially in the shape of thin bars for lining riffles, etc. It has not yet been proven that forged manganese steel can take the place of cast manganese steel to any extent, so only the latter is here considered.

Manganese steel owes its importance to a combination of properties rather than to any one characteristic. It is not so hard as chilled iron; in fact, its ability to resist abrasion does not depend on its hardness alone but on its toughness and hardness and what is known as its ability to flow, and also on its ability to endure repeated distortions of the wearing surface without surface friction, which would result in constant loss of small particles. Its comparative hardness is shown by the scleroscope, cast steel being 30, manganese steel 50, and chilled cast iron 70.

Its property of flowing and its ductility must be guarded against when designing dredge parts; for instance, a bucket bushing must be made shorter than the back eye of the bucket, and clearance must be allowed all around wearing plates when there is a liability of their becoming wedged in place by undue expansion.

Manganese steel will withstand abrasion from heavy impacts much longer than will ordinary steel. Also, attrition caused by sliding material has less effect on manganese steel than on ordinary steel. Under certain conditions it does not resist the attrition of fine sands or slimes as well as other materials do; so manganese steel can not be said to be a panacea for all abrasive ills. It has high tensile strength—about 90,000 pounds per square inch for cast test pieces which have not been forged—but the elastic limit is low and not well defined, running about 40,000 to 45,000.

Partly because the metal is poured while very hot, it lends itself to the casting of intricate designs and thin sections, but exceptional care must be exercised in designing parts to avoid abrupt changes in the thickness of sections, for the castings are put through a severe heat treatment. The metal as cast is hard and brittle. Hence it is heated to about 1,700° F. and quickly plunged in cold water, which usually distorts the casting so that considerable restraightening is required. As manganese is a poor conductor of heat the sections of metal to be annealed can not be thick, not much more than 4 inches, as internal cracks appear in pieces with thicker sections.

Manganese-steel castings should never be heated for any purpose after they leave the foundry, for if they are heated to any other than the correct temperature they lose their valuable properties. Manganese steel differs from other steel in that the skin or outside is softer than the metal underneath, as it decarbonizes after annealing; and because manganese can be dented with a chisel it has sometimes incorrectly been thought inferior. The greatest drawback to a more general use of manganese steel is that it can not be machined with cutting tools, but such efficient grinding apparatus has been perfected that this objection is largely nullified. Formerly it was necessary to cast a soft-steel insert where boring was to be done, and this practice is still followed to some extent; however, it is usually better to grind the bore for all wheels. Manganese can readily be distinguished from other steels for, unlike them, it will not attract a magnet.

One of the peculiarities is that it wears much smoother than does ordinary steel, and its coefficient of friction is less. This may be an advantage or a disadvantage. For instance, on a dredge with a long bucket line if the heel plates are of manganese steel the buckets will sometimes slip to such an extent that undue vibration of the main drive will occur; such heel plates should be made of nickel-chrome steel or some other serviceable metal. It is desirable, however, that screen tread rollers and sheaves wear smooth, and this property is therefore advantageous. It is possible to cast manganese screen plates (Pl. XLII, *B*, p. 74) $\frac{3}{4}$ of an inch wide with $\frac{3}{8}$-inch holes. These wear several times longer than plates made of ordinary steel and are rapidly taking their place.

The best method of determining whether manganese is the proper material to use under certain conditions is to observe it in service. It has often proved unsuccessful where the conditions were thought to be ideal, and it has been found to suit admirably where little promise was held out for its success.

NICKEL STEEL.

Nickel steel takes the place of ordinary steel only when greater strength is required, or in a part that undergoes repeated shock, as the upper tumbler shaft, in which the stresses are repeatedly reversed.

The tensile strength of nickel-steel shafting is 90,000 to 100,000 pounds per square inch against 60,000 to 65,000 pounds per square inch for ordianry steel, and the elastic limit is much greater, provided the nickel content is 3 to 4 per cent, which is the usual percentage for shafting. Nickel-steel castings usually carry a smaller percentage.

The tensile strength does not represent the proper basis for comparison between the two steels, as the ability of nickel steel to withstand shock is much greater than the increased tensile strength over that of ordinary steel would seem to indicate.

When an ordinary steel shafting on a dredge is found too small in diameter it is advisable to substitute nickel-steel shafting of the same size, thus obviating any change of bearings, etc., that would be necessary if shafting of a larger diameter were used.

The shafts of the upper tumbler, the lower tumbler, and the ladder hoist are usually made of forged nickel steel. The only casting made of nickel steel is the upper tumbler when the shaft is cast integral with it. Nickel alone does not affect the hardness of steel to any great degree.

NICKEL-CHROME STEEL.

Chromium increases the hardness and wearing qualities of steel to a marked degree. When it is used in connection with nickel both strength and hardness are obtained, giving steel very useful for dredges. It makes an ideal material for gears performing severe service, and is extensively used for bull wheels and pinions, ladder hoist gears, etc. The teeth do not break nor wear so readily. If manganese wearing plates cause excessive slip, nickel-chrome plates can be substituted to advantage.

The hardness of nickel chrome can be controlled to a nicety by adding the correct percentage of chromium, but usually just enough is added so that the castings are machinable. As it is not necessary to put nickel chrome through such a severe heat treatment, it is practicable to make castings of this material where manganese castings could be made only with much difficulty. Round lower tumblers, buckets, ladder rollers, screen tires, and rollers give much better service when cast of nickel chrome than when made of ordinary steel.

HIGH-CARBON STEEL.

The tensile strength and hardness of steel increase with an increase in carbon, but the ability to withstand shock decreases. In "mild" steel castings the carbon runs between 0.18 and 0.25 per cent. When the carbon rises to 0.80 per cent, machining of the casting becomes difficult. For upper-tumbler castings, which are protected with manganese on nickel-chrome wearing plates, high-carbon steel is well suited, as there is slight chance of breakage and the faces hold up well under the plates. Most equipment for which high-carbon steel is used would have a longer life if made of nickel-chrome steel, but owing to the additional cost of nickel-chrome steel it is not always the more economical.

ELECTRICAL EQUIPMENT ON DREDGES.[a]

Electricity is used almost universally as motive power on gold dredges, except in isolated districts where electric current is not available and where the installation of hydroelectric or steam generating plants would be neither feasible nor economical.

Although many changes have been necessary in the type and arrangement of electric apparatus since electricity was first tried on dredges, the use of electricity has from the first proven its superiority to steam drive as regards cost of power, maintenance charge, and ease of control. The modern dredge is now supplied with heavy-duty motors that give efficient service and require minimum attention.

BUCKET-DRIVE MOTOR.

The design of the motor for driving the bucket line, for which power requirements vary through wide ranges and are at times excessive, is a real problem. When the size of buckets, the speed per minute, and the character of the ground to be handled are known, the motor requirements can be estimated closely for operations under ordinary conditions. However, should a very large bowlder or a reef of hard material be encountered the demand on the motor is immediately increased, and unless protective devices are arranged the motor will either stall or break some part of the bucket line or driving gear, depending on the torque that the motor is capable of developing and the strains that the buckets and gearing are capable of withstanding. Although it is hardly probable that the operator would allow the motor to remain stalled with full current on for a sufficient time to burn it out, should difficult digging of one sort or another—sticky clay, for instance—be encountered, the motor might be periodically so overloaded that the resulting high temperature would eventually break down the insulation and the motor would burn out. Burnouts have also occurred from the unsoldering of connections in the rotors by reason of excessive current caused by an overload.

The maximum reading of the power consumed on the main drive motors on the Yuba No. 14 dredge, while digging and raising ladder, was 652 horsepower; with the digging friction out, the maximum was 428 horsepower; and with buckets running empty 217 horsepower was required.

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[a] See also pp. 94, 101.

In addition to these requirements the bucket-drive motor must be capable of running the bucket line reversed, and also of slowly revolving the empty line for inspection and repairs.

These and other problems have been satisfactorily solved by electrical engineers, so that, generally speaking, by following standard specifications adopted by the foremost dredge manufacturers, anyone, although unfamiliar with the conditions surrounding electrical equipment on gold dredges, would probably get satisfactory results.

Motor economy is of importance, and motor efficiency is vital. Above all things it is necessary to keep the dredge running; a dredge is designed primarily as a money-making machine, and in order to get the best results there should be as little operating delay as possible. A dredge idle is making no money. It may be said that the loss of time incident to motor difficulties is not excessive. In extreme instances, or with new installations, it may run as high as several per cent of the total time. Motor trouble at one of the California dredging companies in 1914 caused only 26 hours' loss of time, or 1 per cent of the whole delay.

POWER CONSUMPTION BY ELECTRICAL EQUIPMENT.

The power consumed by the digging motor is usually 35 to 50 per cent of the total power consumption, and that of pump motors about 25 per cent, except when monitors are employed, when it may run as high as 40 per cent. Centrifugal pumps are used on account of the low head required and the large volume of water to be handled, and these are generally driven by direct-connected motors. The drive of the digging ladder is either directly by gear-connected motors or is by belt. Both of these drives work satisfactorily and each has its champions. Those favoring the direct gear believe that it shows lower operating and initial costs. In regard to the power consumed by these different methods comparisons are difficult, as power consumption on dredges varies greatly on the same dredge over different periods, according to the operating conditions encountered. Comparisons of working costs and power consumption of the Conrey dredges using the direct gear drive and some California dredges using the belt drive are shown by the following tables. It is difficult to make satisfactory comparisions between total working costs of dredges operating in different fields, and even more so to make comparisons of power consumption of the bucket drive, as there is where the greatest variance occurs. The following table[a] is interesting in its comparison of the amount of power consumed by the different Natoma dredges during one month's operation. As the operating time is fairly even, a study of the yardage in comparison with the power consumed illustrates the difference of the physical condition of the gravel worked.

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[a] Von Bernewitz, M. W., Dredging at Natoma, Cal.: Min. and Sci. Press, vol. 107, Dec. 27, 1913, p. 1017.

Power consumed by Natoma dredges during one month's operation.

Item.	Dredge No.									
	1	2	3	4	5	6	7	8	9	10
Bucket capacity, cubic feet	13.5	8.5	8.5	13.5	9	9	9	15	15	15
Height of bank being dredged:										
Above water, feet	4	3	6	2	12	6	20	10	3	16
Below water, feet	16	23	20	18	44	54	42	53	58	21
Daily operating time, hours	19	20	19	21	21	21	20	19	20.5	19
Gravel dredged, cubic yards	232,014	232,644	124,440	205,274	159,294	150,533	170,690	142,529	238,004	253,376
Power consumed, kilowatts	221,440	143,520	137,600	192,600	238,800	219,600	272,000	346,800	362,400	282,000
Power consumed per cubic yard, kilowatts	0.95	0.62	1.10	0.93	1.49	1.45	1.59	2.4	1.52	1.11

The power used by the No. 2 dredge was only 0.62 kilowatt as compared with 2.4 kilowatts a cubic yard for the No. 8 dredge. The total operating costs for these dredges over longer periods are given in a subsequent discussion of working costs. The following table, covering power consumption during an extended period, is furnished by Mr. Charles Kammerer, of the Conrey Placer Mining Co.

Power consumption of Conrey dredges from time of starting to July 31, 1915.

Item.	Dredge No.			
	1	2	3	4
Bucket capacity, cubic feet	7½	7½	9½	16
Total time operating, hours	44,950	41,754	50,867	23,520
Average depth, feet	22.1	28.1	40.8	45.3
Cubic yards dredged	7,890,000	6,268,000	8,646,000	10,025,000
Power consumed, kilowatt-hours per yard	1.55	1.89	1.73	1.36
Power cost per cubic yard, cents	1.27	1.55	1.48	1.09

Although dredges 1 and 2 were identical in size and equipment, it is evident that the operating conditions were different, for the power consumption on the one was 1.55 kilowatt-hours a cubic yard as compared with 1.89 kilowatt-hours on the other. The working conditions of the No. 3 and the No. 4 might be more nearly compared, but these dredges were different both in size and type, so that significant comparisons are again impossible, the No. 3 dredge being a flume dredge and the No. 4 a bucket-elevator stacker dredge with the standard type of tables and with the most powerful digging motor in use on a gold dredge.

The power consumed on the Dawson dredges of the Yukon Gold Co. was as follows during a three-year period, approximating closely 1 kilowatt-hour a cubic yard dredged:

Power consumption of Dawson dredges during three-year period.

Year.	Total power consumption.	Total yardage.	Power consumption a cubic yard.
	Kilowatt-hours.		*Kilowatt-hours.*
1913......	4,971,114	5,133,575	0.906
1914......	4,973,153	4,800,781	1.03
1915......	5,264,273	5,041,075	1.04

The varying amount of power consumed on the same dredge when digging ground of different textures is shown by the following table prepared from figures furnished by Mr. W. H. James, of Oroville, Cal.

Power consumed on a 6-foot dredge operating in the Oroville district, showing that difference in formation regulates power used.

MONTHLY FIGURES.

Cubic yards worked.	Power consumed.	Power consumed a cubic yard.	Nature of formation.
	Kilowatt-hours.	*Kilowatt-hours.*	
101,564	89,880	0.885	Sand filling.
91,822	77,040	.839	Do.
85,314	67,320	.789	Do.
68,577	86,640	1.263	Red soil and clay filling.
65,768	82,560	1.255	Do.
62,638	85,440	1.364	Do.
YEARLY FIGURES.			
874,845	882,837	1.01	Sandy filling.
710,257	956,050	1.34	Soil and clay filling.

Most of the California dredges, and, in fact, a large part of all dredges operating to-day have the bucket ladder belt-driven from a powerful motor on the main deck, and this method has proved satisfactory. Although it was found necessary after a smaller motor had failed to do the work satisfactorily to equip the bucket drive of the Conrey dredge with a 550-horsepower motor capable of 700 horsepower for continuous periods, there is no dredge in California requiring such a large motor on the digging end, and the conditions of operation for some of the California dredges are more difficult than those of Conrey No. 4. The Yuba No. 14 dredge is equipped with a 400-horsepower motor, and the large Natomas dredges Nos. 8, 9, and 10, which have been subjected to perhaps the hardest digging of any gold dredges in the world, have practically the same equipment. The power consumption is estimated on those dredges at 89 kilowatt-hours a cubic yard.

The average power consumption a cubic yard on the No. 14 dredge for a period of nearly two years was 1.45 kilowatt-hours. The 18-foot boat of the Boston & Idaho dredge with easy digging ground requires a belt-driven motor or main drive of only 300 horsepower, showing how the motor requirement varies under difficult digging conditions. The dredge was first designed with 14-foot buckets; a new line afterwards added had 18-foot buckets, but no change in motor drive was required.

The Yuba No. 15 dredge (Pl. LIII, *A*), designed to dig 82 feet below water level and equipped with 17-foot buckets, will have a 500-horsepower digging motor, which is 100 horsepower larger than that of the Yuba No. 14 dredge.

RECENT TYPE OF DIRECT-CONNECTED MOTOR DRIVE.

The following description of the bucket ladder drive at Ruby, Mont., illustrates the most recent practice of the direct gear-connected motor drive.

The most powerful digging motor in use for gold dredging is on the Conrey No. 4 dredge. This motor is of 550 horsepower, with a 100 per cent overload factor, and especially designed ventilation. The control arrangement is a radical departure from cast-iron grids, and consists of a water rheostat. The device as installed at Ruby differs from the Westinghouse water-rheostat control mainly in the manner of cooling of the electrolyte. In the latter type, coils of pipe are placed in the large electrolyte tank and cooling water is pumped through them, but in the Conrey installation the cooling coils are in the dredge pond, being fastened to the side of the dredge. Pipes connect the coils to the electrolyte tank, and a small pump constantly circulates the electrolyte through the coils. This arrangement insures greater cooling surface; also, there is no chance for the coils to become choked by the electrolyte, as occurred when this method was tried in another dredging field, where the water pumped from the dredge pond through the coils was muddy.

Previous to the construction of the Conrey No. 3 dredge, the bucket drive on the Ruby dredges was by sprocket chains, whereas California dredges then, as is now the general practice in that State, were driven by a rubber belt from motors placed on the main deck. When the No. 3 dredge was constructed, the direct connection of the digging motor to the bucket chain by cut-steel gearing was first used (Pl. LIV, *A*). To eliminate undue shocks and overloads on the motor, should the bucket chain run against large bowlders, or reefs in the bedrock, a large friction-disk clutch was put on the first countershaft from the motor, and a circuit breaker was placed on the power line of the digging motor. One of the essentials of perfect service from direct gearing is cut gears with perfect alignment. On the No. 3 such alignment was insured by building a large steel base to carry the digging motor and by using two counter shafts and an upper tumbler shaft. After seven years of continuous service, this direct drive is giving entire satisfaction. It has been used on the No. 4 dredge since the large motor was installed. Some of the large dredges constructed in other fields have adopted the Ruby practice of direct gear drive.

KIND OF ELECTRIC CURRENT USED.

The problem of control in large motors for digging service would be simpler if direct-current motors could be advantageously used. However, as the use of these motors is impracticable the control must be designed for alternating-current motors. One of the reasons is that alternating current is furnished by the power companies, as direct current can not be economically transmitted from distant generating plants. The use of direct-current motors, therefore, would require a motor-generator set to transform the current. The direct-current motors are larger, more complicated, and, besides being more expensive in first cost, require more skillful supervision. In this connection Mr. E. G. Thunen,[a] of Hammonton, Cal., makes the following statement:

> A comprehehensive discussion of the relative advantages and disadvantages of the several different direct-current systems as contrasted with the number of alternating-current systems and motors which are now available would be very long indeed. Probably the chief grounds for the discussion of the comparative advantages of direct-current and alternating-current power systems for dredging are the relative economies of power transmission and excellence of motor-performance characteristics, particularly as regards speed and torque. The high-voltage direct-current systems would certainly have good power-transmission economy, and the alternating-current series motor, brush-shifting polyphase series motor, and others can be built with speed-torque characteristics approaching those of the direct-current motors. Of all the motors, direct current or alternating current, the induction motor is the simplest, least liable to get out of order, and cheapest. Direct-current motors would be almost ideal for some of the drives on a dredge, especially the digging motor, if proper protective devices, both circuit breakers and slipping frictions, were employed.
>
> In the event of encountering rock reefs the importance of protective devices would be many times greater with direct-current bucket drive than with alternating-current drive, not only to protect the motor, but also for the protection of the bucket line and gearing. The reason for this is the very high torque which the direct-current motor is capable of developing at reduced speed, while the torque of the alternating-current motor is limited.
>
> Drum-type controllers are thought by many engineers to be sufficient for digging motors up to 200 horsepower. There are a number of digging motors of this size operating satisfactorily with drum controllers which have also been used on digging

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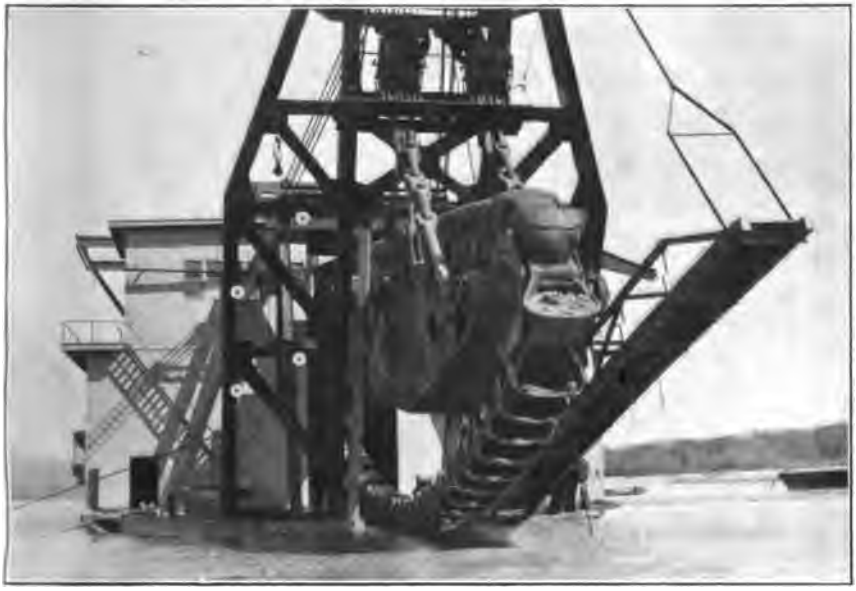
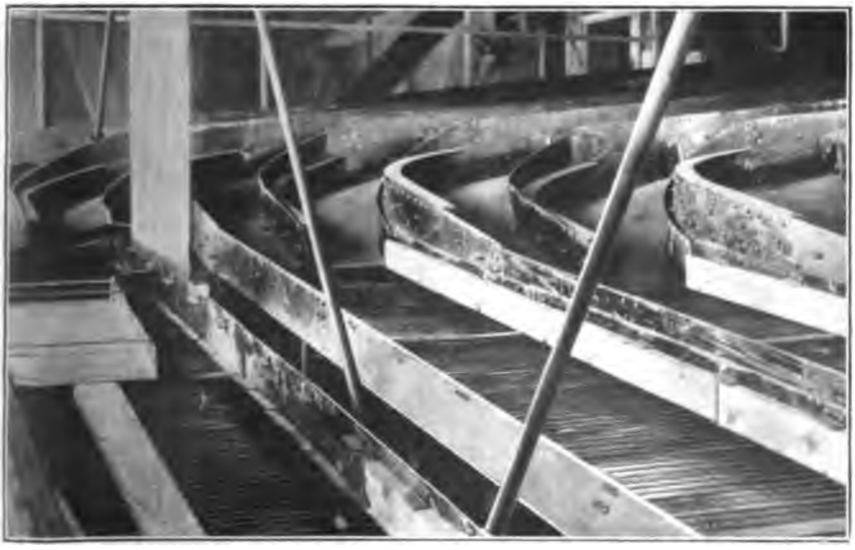
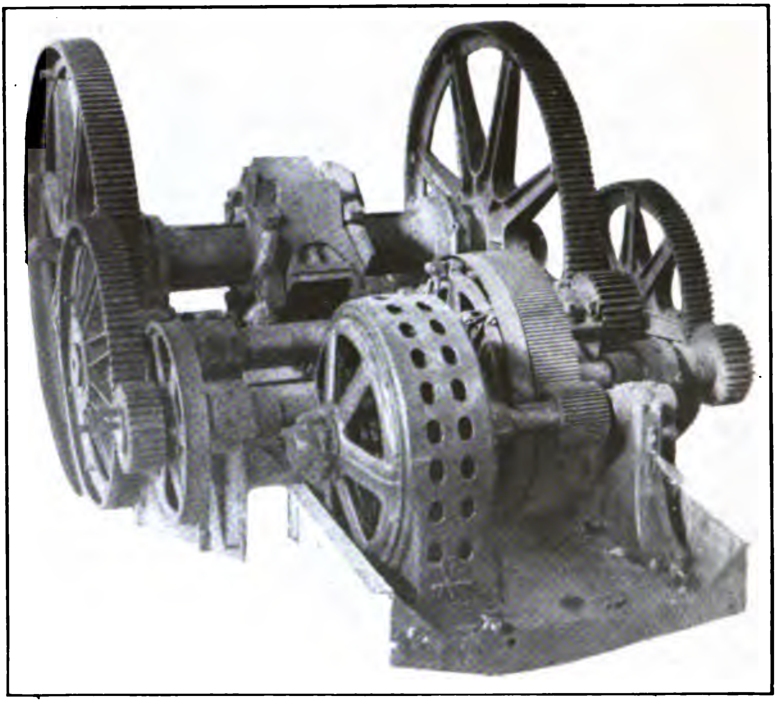
[a] Personal communication to author.



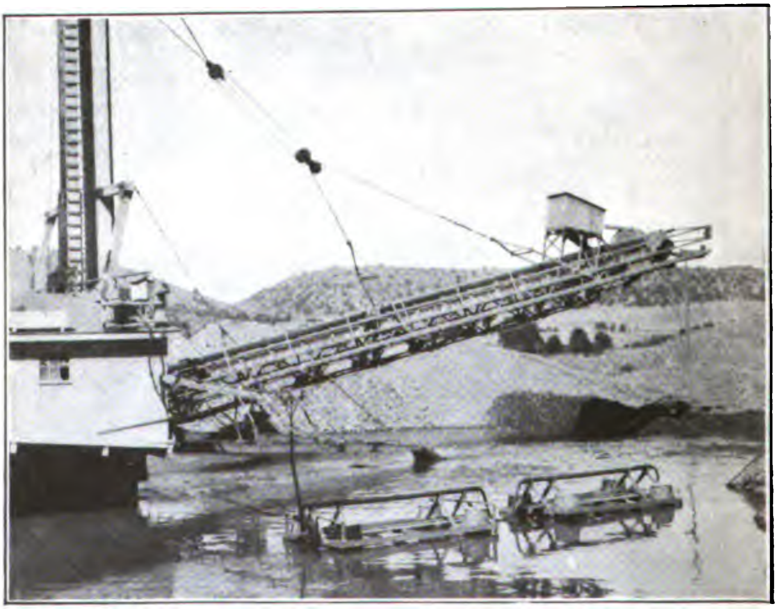
A. YUBA NO. 15 DREDGE, 17-FOOT BUCKETS.



B. GOLD-SAVING TABLES AND SLUICES, CONREY NO. 4 DREDGE.



A. DIGGING MOTOR OF CONREY NO. 3 DREDGE, WITH CUT-STEEL GEAR.



B. POWER-CABLE FLOATS, EL ORO DREDGE.

motors as large as 300 horsepower. For large motors, that is, those of over 200 horsepower, magnetic control or liquid rheostats are used, the latter being described above while the former are in general use in California; experiments made in that field with the liquid rheostat have not been satisfactory, perhaps for the reason mentioned above. Magnet-switch or grid control has been used satisfactorily on digging motors of the large California dredges operating under severe conditions, and no trouble has been experienced with same.

There is no doubt that a satisfactory combination of magnetic switch for the primary circuit of the motor, forward and reverse with overload relays, and of drum controller for cutting out secondary resistance, could be made for motors of 300 horsepower and over. This of course would do away with the automatic feature of uniform torque and current which, however, is not necessarily a disadvantage but is in fact a great advantage in certain cases.

The application of an ideal motor drive for the belt-conveyor tailing stacker is somewhat difficult. The requirement is severe for the reason that it is so often necessary to start with abnormal loads, and owing to the fact that the motor is generally placed at the far end of the stacker. The desire to run at reduced speeds under light load complicates the problem. A case in point is that of Yuba No. 14, where the stacker is 160 feet long; a high starting torque is required and it sometimes takes several minutes to bring the belt up to speed. Squirrel-cage motors with heavy-torque characteristics have been successfully used for this stacker drive, but some engineers prefer the slip-ring type, as a smaller motor can be used and they figure a consequent loss of power with the squirrel-cage type. The drive from the stacker-belt motor may be by silent chain belt, direct-gear, or ordinary belt. Each has its advocates and advantages. The stacker motor is small compared with the total motor equipment on a dredge, and a slight loss of efficiency is not as vital here as at the digging end. Revolving screens are motor-driven from either the upper or the lower end. Better practice seems to be from the lower end. On large dredges the screen motor must be designed to take a heavy starting load. When the screen is at rest and loaded with clay and rocks, as is often the case, considerable power is required to bring it up to speed. Some dredges are supplied with separate motors for ladder hoists. This is necessary when using a direct main-drive motor, and is also advantageous on other dredges, permitting greater flexibility of operation and consequent saving of time on account of being able to hoist or lower the ladder without interfering with the main-drive action. Electric current for dredge operation is usually transmitted from the main lines to a substation and from there transformed to the proper voltage for dredge use, from which it may be afterward stepped down on board the dredge. On the Yuba River the current is generally brought on board at 4,000 volts and stepped down to 440 for general use on the dredge. Most of the dredges of the Natomas Consolidated take the current aboard at 2,200 volts for use on all motors of 50 horsepower and over, whereas for smaller motors the current is stepped down to 440 volts as above.

VOLTAGE OF CURRENT USED ON DREDGES.

At Ruby, Mont., the current is transmitted at 46,000 volts to the substation, where it is stepped down to 2,200 volts, the voltage used aboard the dredges. Operating all motors at 2,200 volts makes it unnecessary to have transformers on board, and it is thought by some that, although the high voltage is more dangerous to workmen, there is less loss of power, and also less danger of fire, except on the all-steel dredge where the danger from fire is obviated. Using the higher voltage motors and dispensing with transformers reduces the weight of machinery on the hull. All motors on the large dredge of the Boston and Idaho Co. are designed for 2,200 volts. At Breckenridge the Tonopah Co. buys current at 13,000 volts, which is stepped down to 440 on all motors.

Thunen [a] states further—

> In selecting the electrical equipment for a dredge (or for any other purpose for that matter) it should be borne in mind that for the same economy of power transmission the cross section of line and power-cable conductors for the 2,000-volt system must be four times as great as that necessary for the 4,000-volt system. The other line construction materials—namely, insulators, poles, and cross arms—will be the same for both voltages.

Electric power is brought on board a dredge by means of armored insulated cables. Some operators use floats or small pontoons (Pl. LIV, *B*) to support the cable across the pond; others consider it better practice to carry the cable under water, because the insulation, if the floats are not well arranged and the cable gets wet and dry alternately, rots quickly unless thoroughly protected by rubber hose. In some places, as at Idaho City, Idaho, the current is taken on board the front end of the dredge and no floats are required; but winter conditions there are severe upon the cable insulation. A power cable supported on properly constructed floats, or pontoons between the shore and the dredge, is vastly easier to move than if unsupported. This is particularly noticeable on a large dredge digging wider than a single cut. The power cable for the large dredges weighs about 5 pounds a foot.

The table following shows the motor equipment on a number of different dredges:

Electrical motor equipment on dredges of Natomas Consolidated of California.

Name of dredge.	Size of buckets.	Average depth of cut.	Size of motors.				Pumps.		
			Bucket.	Swing.	Screen.	Stacker.	Group 1.	Group 2.	Group 3.
	Cu. ft.	*Feet.*	*Hp.*	*Hp.*	*Hp.*	*Hp.*	*Hp.*	*Hp.*	*Hp.*
Natoma No. 1	15	32	300	35	75	25	150	35	30
Natoma No. 2	8	35	200	50	50	35	75	35	10
Natoma No. 3	8	40	200	25	50	35	75	35	30
Natoma No. 4	15	42	300	30	75	(2) 25	150	100	50
Natoma No. 5	9	50	200	30	50	40	100	25	50
Natoma No. 6	9	50	200	30	30	50	100	50	40
Natoma No. 7	9	58	300	30	50	50	100	50	30
Natoma No. 8	15	60	400	75	75	60	150	75	50
Natoma No. 9	15	60	400	75	75	60	150	75	50
Natoma No. 10	15	60	400	75	75	60	150	75	50

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[a] Thunen, E. G., Personal communication.

The following description by Gardner and Shepard[a] of the electrical equipment of the Yuba No. 14 dredge will be of interest, as it represents the latest practice in the California field:

> The electrical equipment of the Yuba No. 14 is something of a departure from the equipment heretofore considered standard for California gold dredges. This departure from former practice was made in the light of experience gained in the operation of four dredges of the largest type, which were built by the Yuba Construction Co. for the Natomas Consolidated of California and one similar dredge built for the Yuba Consolidated Gold Fields. All electrical equipment was made by the General Electric Co.
>
> Power is supplied by the Pacific Gas & Electric Co. as a three-phase, 60-cycle, 4,000-volt current. The power is brought on board the boat through 750 feet of No. 0 B. & S. gage, three-conductor cable, each conductor being stranded and insulated with $\frac{3}{32}$-inch 30 per cent Para rubber, conductors twisted together, rounded out with jute, and covered with a layer of $\frac{1}{8}$-inch varnished cambric. Over this is a layer of jute, and the cable is armored with No. 10 B. W. G. armor put on with short pitch. This cable is insulated for 4,500 volts working pressure. The shore cable is brought aboard the dredge on pontoons and enters a switch house on the upper deck near the stern of the dredge, where there is installed a nonautomatic, 300-ampere, 7,500-volt oil switch with hand-operated remote control. The switch is mounted on pipe framework away from the panel carrying the operating lever. There are installed in this switch house two 4,400/110-volt, 200-watt potential transformers with fuses, and two 150-ampere current transformers. The secondaries of these transformers are connected to indicating instruments mounted in the winch room. From the switch house the current is fed through triple-conductor varnished cambric cable in conduit to the primary side of the main transformer. There is installed one 15-kilowatt, 4,000-volt primary, 230/115-volt secondary, oil-cooled transformer to supply lights. From the secondary side of the main transformers, various feeder circuits supply the different motors, triple-conductor varnished cambric being used in iron conduits.
>
> The control panel for the main-drive or digging motor, and for the winch motor, is placed in the winch room, as is also the instrument panel. The instrument panel contains the following instruments, which are supplied, as mentioned above, from the instrument transformers placed in the entrance switch house: One 5-ampere alternating-current ammeter with 150-ampere scale, one 175-volt alternating-current voltmeter with 8-point potential receptacle and plug, one alternating-current polyphase indicating wattmeter with 1,200-kilowatt scale. The readings of these instruments give the total input to the dredge.
>
> The double-circuit control panel for the main-drive and winch motors has the following equipment: One 60-ampere alternating-current ammeter (winch motor); one 5-ampere alternating-current ammeter, 800-ampere scale (main-drive mortor); one 200-ampere automatic oil switch with double-coil series trip (winch motor); one 600-volt, 800-ampere automatic oil switch with double-coil trip (main-drive motor); and two 800-ampere current transformers in main-drive motor circuit.
>
> The panels for the pump motors are placed on the lower deck, the starting compensators for the pump motors being mounted alongside the panels. These compensators are all provided with inverse time-limit relays. The pump-motor panels consist of the following: Two 3-phase 2-circuit motor panels, each panel mounting two 500-volt, 200-ampere lever switches controlling the high pressure, the low pressure, the 6-inch two-stage, and the vertical pumps.
>
> The stacker and screen motors are controlled from independent panels in the stern of the dredge, each panel having the following equipment: One 200-ampere automatic oil switch, with 2-coil series trip, and a mounting on the front panel for one

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[a] Gardner, W. H., and Shepard, W. M., The largest electrically operated gold dredge: Min. and Sci., Press, vol. 108, June 27, 1914, pp. 1055–1056.

reversible type controller. The oil-switch operating lever is directly above the controller. These motors are of the slip-ring, variable-speed type, provided with resistance for continuous operation at 50 per cent to full speed, and with reversible controllers. They operate at 600 revolutions per minute, the screen motor being 75 horsepower and the stacker motor 60 horsepower. In most of the former gold dredges these motors were of the constant-speed squirrel-cage type. However, in the more recent large dredges, owing to difficulties sometimes experienced with squirrel-cage motors on account of the heavy starting conditions, and the desirability at times of running for short periods at reduced speeds, slip-ring motors, with resistances for 2-minute starting duty, were used. In heavy work, however, these were found too light, and for Yuba No. 14, resistances for continuous operation were installed and reversible controllers used. There is also provided, for the repair shop on board the dredge, a 2-horsepower, 180-revolutions per-minute, three-phase motor with pulley and base.

All of the panels are natural black slate mounted on pipe supports.

The digging or main drive of the dredge is a 400-horsepower, 514-revolution-per-minute, 3-phase, 60-cycle, 440-volt, slip-ring, variable-speed, induction motor, with three bearings, pulley, and sliding rails, with a master controller and contactor panels, and is provided with resistance good for continuous operation at all speeds, from 50 per cent normal to normal.

The contactor equipment is provided with current-limiting relays, which limit the maximum load that the motor can take, and so protect both the motor itself, and the complete digging mechanism that it drives. This is of considerable importance in keeping down the cost of repairs, and preventing loss of time incident to repairs. The digging mechanism is subject at times to sudden and excessive loads, and without the current-limiting feature provided by the contactor control, the only protection would be given by the overload oil switch. Owing to the annoyance of having this switch tripping out frequently, it is usually set so high as to afford little protection. The current-limiting relays also protect the motor and digging mechanism when starting. The winchman can throw the master controller to the full "on" position, and the motor will come up to speed at a predetermined rate, which will not impose excessive stress on any part of the mechanism.

Another motor operates the winch, raises the spuds, and swings the boat. This motor is a 35-horsepower, 600-revolution-per-minute, 3-phase, 60-cycle, slip-ring, variable-speed induction motor, with a controller and resistance for continuous operation at 50 per cent to full speed. The motor is provided with pulley and base. In a few instances on smaller dredges, motors of intermittent rating were furnished for operating the winch. These motors, however, have not been found suitable for this service, as the winch operates almost continuously.

All the pump motors are squirrel-cage motors, the high-pressure pump having a 150-horsepower,600-revolution-per-minute motor, the low-pressure pump, a 75-horsepower, 600-revolution-per-minute motor, the 6-inch two-stage pump, a 50-horsepower, 1,200-revolution-per-minute motor, and the vertical pump a 10-horsepower vertical motor. These motors are all direct connected to the pumps, and with the exception of the 10-horsepower motor, are all provided with welded end-ring construction in the rotor.

Practically the same conditions governing the electrical equipment on No. 14 obtain on No. 15 dredge, except that the latter dredge is designed for deeper digging. In addition to the larger (500-horsepower) digging motor on the No. 15 dredge, there will be a separate motor drive for raising and lowering the stacker, an additional hold pump (two hold pumps on No. 15), direct-current control for the main-drive contactor panel and switch arrangements to start the pump motors from the middle point of the main transformers in place of using compensators. The transformers will also be a little larger than those on No. 14.

Plates LV[a] and LVI[b] and figures 11[c], 12, 13, and 14[d], which include views of different dredges using electrical equipment show the placing of the motors.

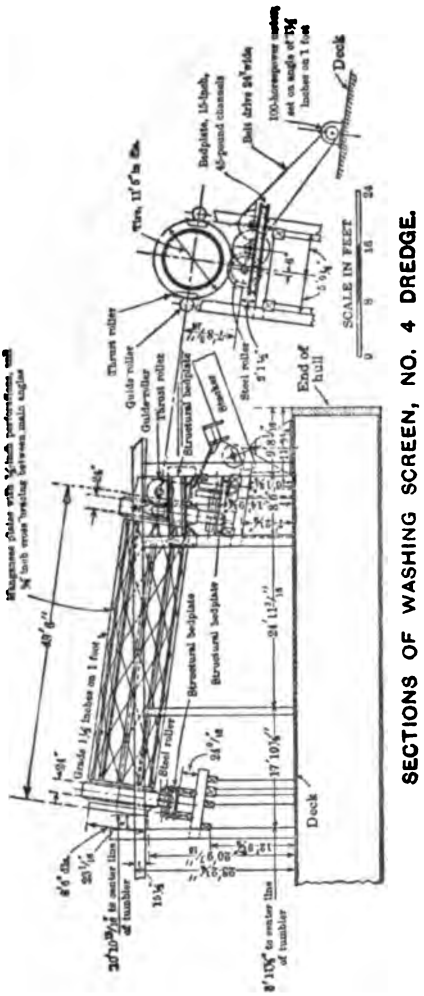
SECTIONS OF WASHING SCREEN, NO. 4 DREDGE.
Tire, 11'6" in dia.
Bedplate, 15-inch, 45-pound channels
Belt drive 24" wide
100-horsepower motor set on angle of 7¼ inches on 1 foot
Deck
Thrust roller
Guide roller
Guide roller
Thrust roller
Structural bedplate
Bracket
Steel roller
5'1½"
5'9¼"
End of hull
SCALE IN FEET
0
8
16
24
Manganese plates with ½-inch perforations, all ¾" inch cross-bracing between main angles
43'6"
Grade 1½ inches on 1 foot
Steel roller
24 9/16
Structural bedplate
Structural bedplate
24'11 1/16"
17'10⅞"
Deck
8'6" dia.
20 1/16
15⅞
20'10 9/16" to center line of tumbler
8'10⅞" to center line of tumbler

FIGURE 11.—Sections of washing screen of Conrey No. 4 dredge, showing placing of 100-horsepower driving motor.

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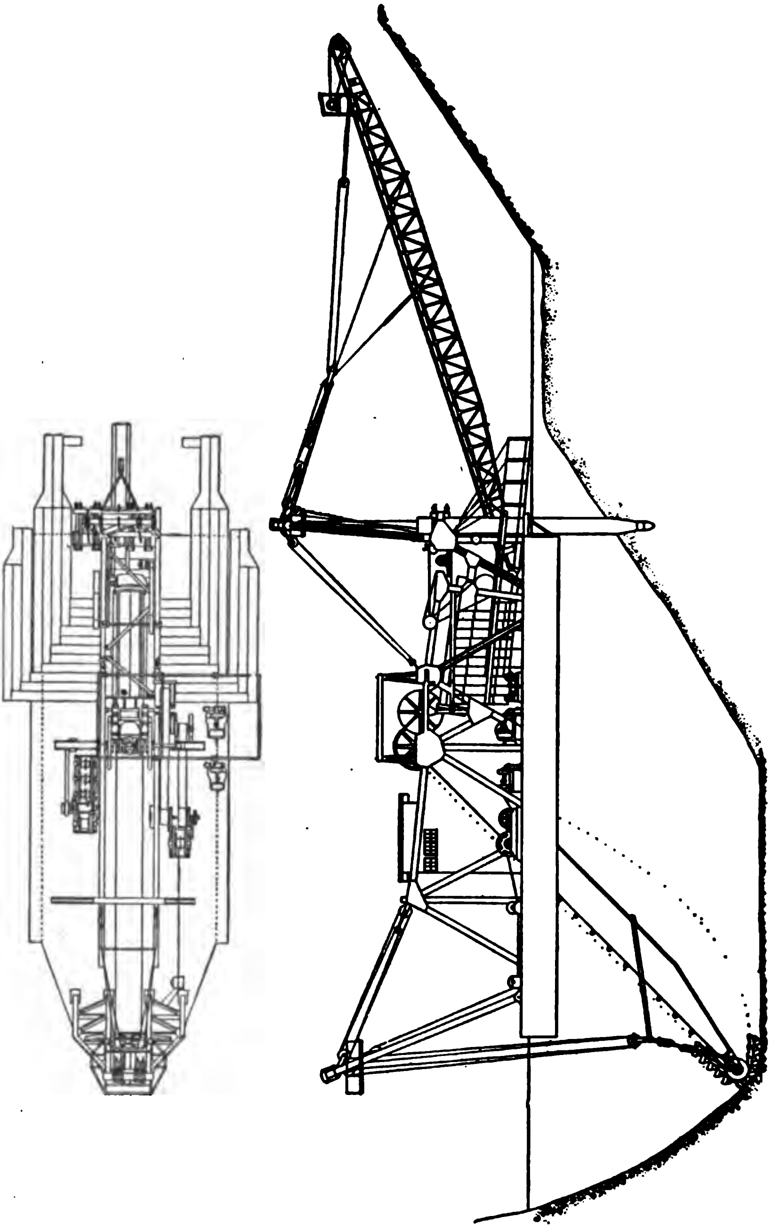
[a] Jennings, Hennen, The history and development of gold dredging in Montana: Bull. 121, Bureau of Mines, 1916, p. 18.

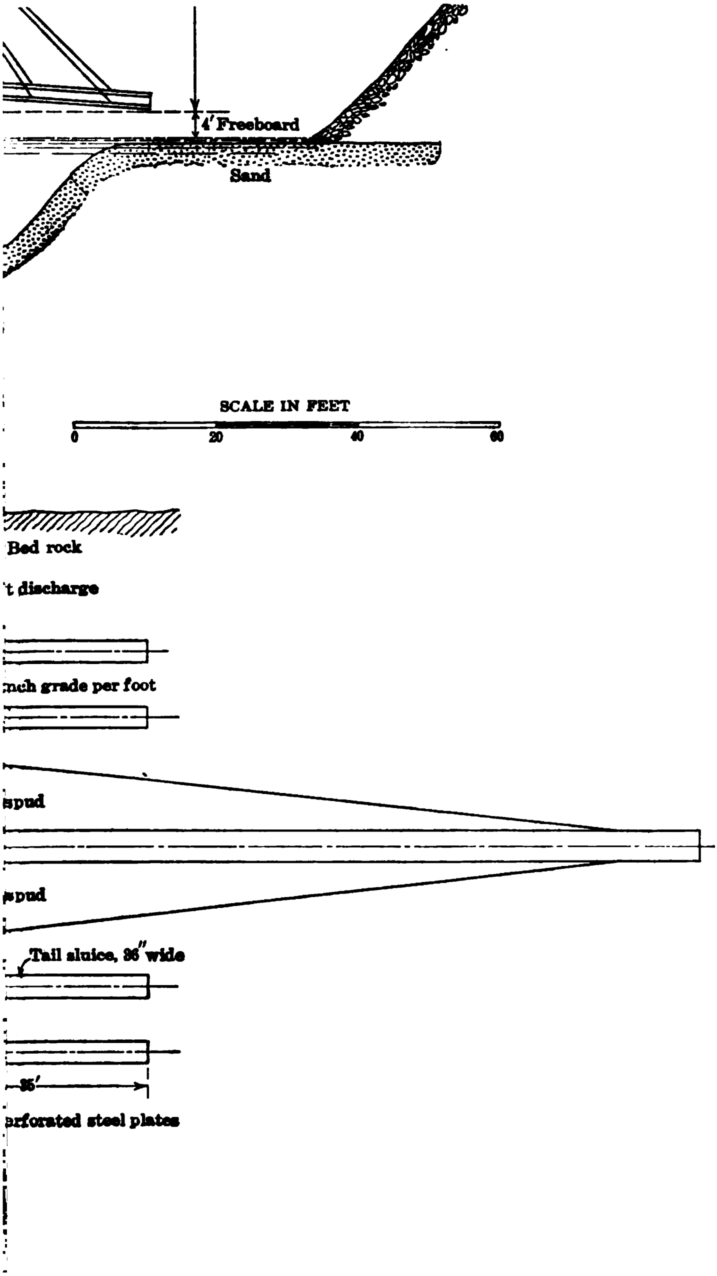
[b] Jennings, Hennen, work quoted, p. 16.

[c] Jennings, Hennen, work quoted, p. 18.

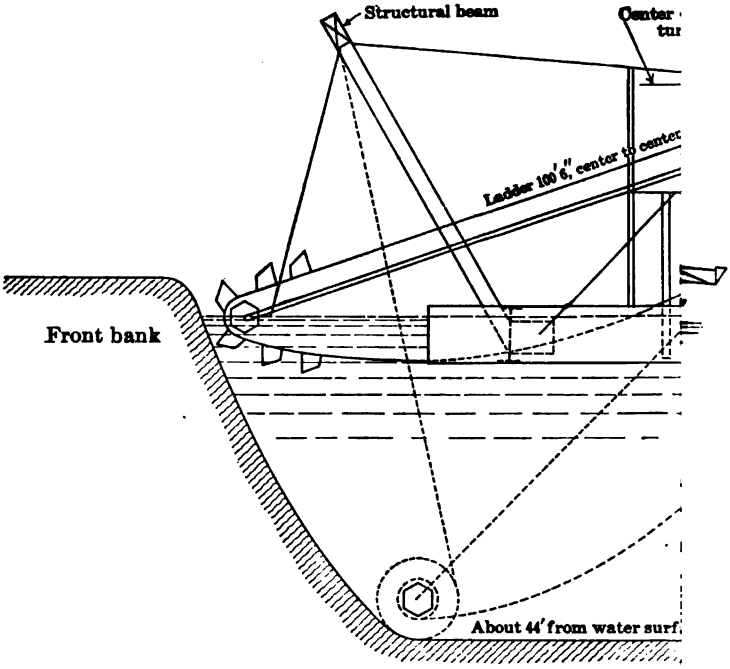
[d] Janin, Charles, Review of gold-dredging in 1912: Min. and Sci. Press, vol. 106, Jan. 4, 1913, p. 47.

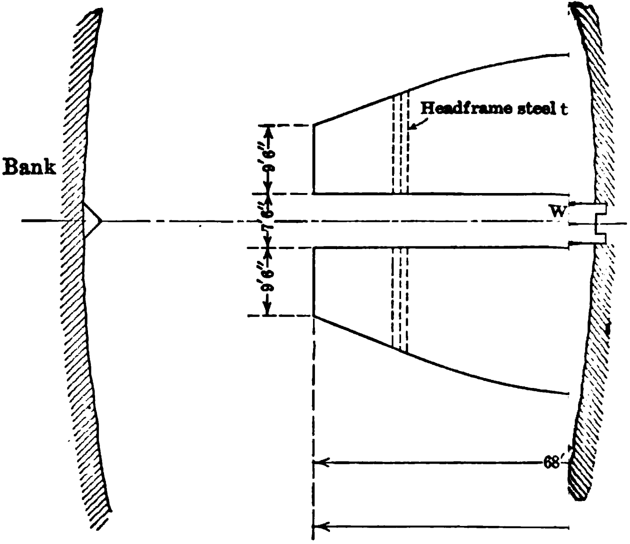


0
Bed rock
t discharge
nch grade p
spud
spud
Tail sluic
35'
erforated st



4' Freeboard
Sand
SCALE IN FEET
0
20
40
60
Bed rock
't discharge
nch grade per foot
spud
spud
Tail sluice, 36" wide
35'
rforated steel plates


Structural beam
Center
tur
Ladder 100'6", center to center
Front bank
About 44' from water surf


Bank
Headframe steel t
9'6"
7'6"
9'6"
W
68'

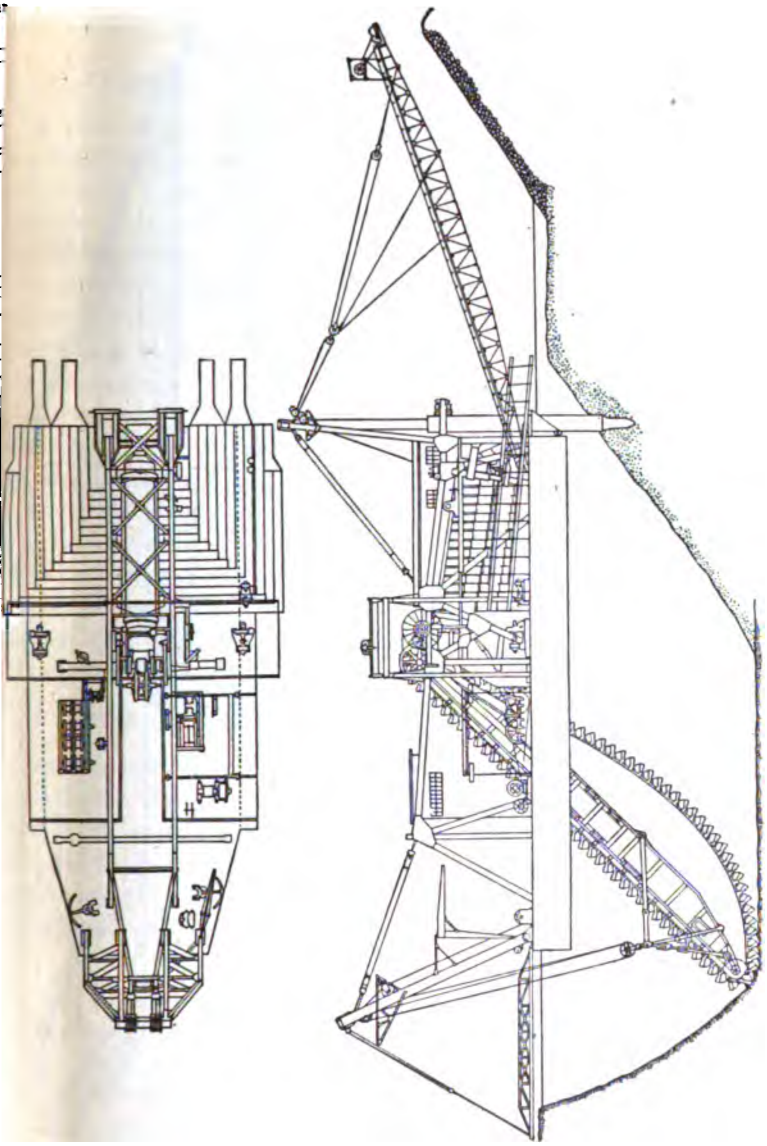


FIGURE 13.—Plan and elevation of 17 cubic foot Yuba dredge electrically operated, digging 80 feet below water level.

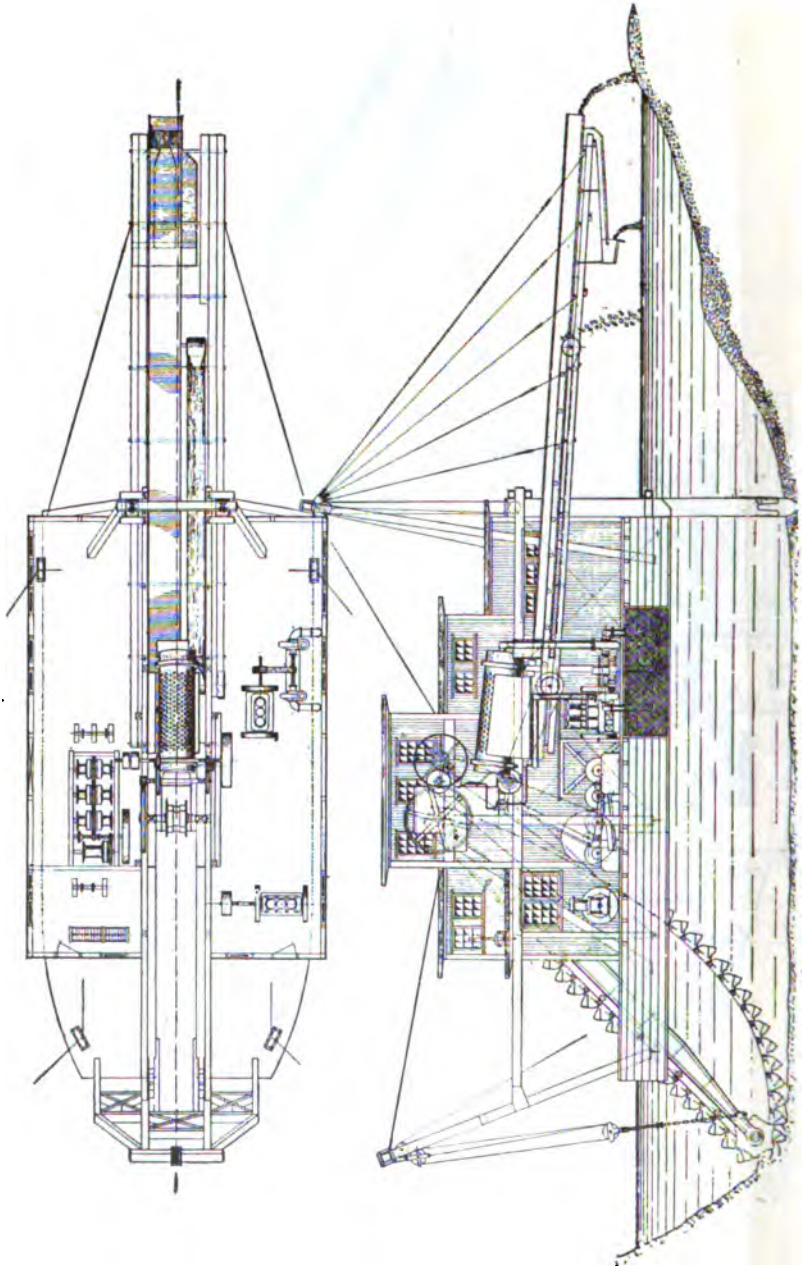


FIGURE 14.—Plan and elevation of flume-type Alaskan dredge with revolving screen and with distillate engines.

REBUILT DREDGES.

FEASIBILITY OF MOVING DREDGES OR MACHINERY.

A matter of considerable interest at present is the moving of machinery from dredges that have worked out the areas for which they were built or have been dismantled and replaced by other dredges. Areas that a few years ago were deemed too small for profitable dredging are being considered on the basis of modern practice and with the idea of reconstructing a used dredge. The machinery of some of these dredges that have been or are to be dismantled is in good condition and fit for many years of service; and on properties not too difficult of access it can be refitted to new hulls and practically new dredges built, in some instances at less than 50 per cent of the original cost. These rebuilt dredges may not always be adapted to handle the gravel at as low operating cost as might be attained with new dredges, but the smaller expense of installation will prove a large factor in their selection and use. The machinery of two dredges that proved unprofitable at Golden, Colo., was moved to new hulls, one dredge being rebuilt in the Breckenridge district in the same State and the other at Jenny Lind, Cal., and both dredges yielded a profit in their new locations.

EXAMPLES OF SECOND USE OF DREDGES OR MACHINERY.

Dredge machinery from dismantled hulls in California has been moved to Alaska, and a number of dredges have been dismantled and rebuilt on other areas in the same or in a different State. Several of these dredges were financial failures in the first locality and proved successful afterwards, as were the Golden dredges, but with one dredge at least poor judgment seemed to follow its career.

TIN CUP DREDGE.

This was the Tin Cup, a 3-foot dredge, originally built near Shoshone, Wyo., at a cost of approximately $49,000. The recovery from the ground for which it was built was less than 1 cent a yard. The dredge was afterwards purchased for about $11,000 and moved to Gunnison County, Colo., at a total cost stated to be approximately $65,000, or more than a similar dredge would have cost new. The second field of operations was seemingly no better chosen than the first, as the dredge was sold at a sheriff's sale in 1915.

SCOTT RIVER DREDGE.

Examples of a more favorable kind may be mentioned. The Scott River dredge, first operated at Callahan, Siskiyou County, Cal., was unprofitable, and it was shut down in May, 1910, after it had worked a little less than two years. This dredge, which was equipped with 7½ cubic foot buckets, was purchased by the Alta Bert Gold Dredging Co., and the machinery moved to Trinity County, Cal. The total cost of the rebuilt dredge was approximately $90,000, including purchase price of machinery, freight charges, cost of hauling 28 miles at 1 cent a pound, and cost of building a new modern hull and a power line 5 miles long. This dredge has been working successfully for several years.

BUTTE DREDGE.

The Butte dredge, which was equipped with 3½ cubic foot buckets, was built and put in operation at Oroville, Cal., in November, 1902, at a cost $52,000, and was dismantled in July, 1910, after handling nearly 3,250,000 cubic yards. The machinery, valued at $6,000, was moved to Jenny Lind, Calaveras County. The cost of the rebuilt dredge, including all charges, is given below. It will be noticed that the cost of new machinery, which consisted principally of new buckets, was $9,446.

Cost of rebuilt Butte dredge.

Estimating and traveling expenses		$237.23
Stationery, drafting, etc		905.80
Superintendence		1,910.00
Hull and superstructure:		
Lumber		4,655.61
Hull irons and spikes		1,005.41
Labor		4,650.95
Paints and materials		347.15
Hardware and sundries		1,212.73
Machinery installation:		
Repairs	$910.28	
Freight, hauling, etc	3,012.44	
Labor and sundry materials	5,033.00	
Electric wiring and equipment	1,014.32	
		9,969.94
Total cost of moving and rebuilding		25,356.52
Outside expense:		
Camp	$727.24	
Water plant	771.71	
Power line, transformers, etc	2,673.50	
Changes in construction	1,900.22	
		6,072.67
		9,446.40
Value assessed to old machinery		6,000.00
Total cost of rebuilding dredge		46,875.59

EL ORO NO. 1 DREDGE.

The machinery of the El Oro No. 1 dredge, a 5-foot dredge, was moved from Oroville, Cal., to a new location at Yreka, Cal., where the dredge was rebuilt, after being redesigned. The cost of moving the machinery and rebuilding the dredge was approximately $55,000, the original company carrying out the work of rebuilding and operation. Other companies operating dredges in California have moved machinery from their dismantled dredges and advantageously reconstructed the dredge in a new locality. One of these dredges, the Pacific No. 1, was moved from Oroville to new ground on the American River, the total time required from commencing to dismantle the dredge until the rebuilt dredge started operations being just four months, which was rapid work.

DREDGE NEAR LEADVILLE, COLO.

A rebuilt dredge which cost more than was first anticipated but which has been a success is that known as the Magpie, on the Derry Ranch, near Leadville, Colo., originally built in South Dakota at a cost of approximately $105,000 ready to start. The dredge was unsuccessful and ceased operations. It was purchased by the New York Engineering Co., which had arranged to dredge the Derry Ranch property on a royalty basis. Dismantling of the dredge required three weeks. It was moved to the Derry Ranch, where construction commenced May 20, 1915. The dredge began operations October 10, 1915. Many important changes were made in construction, the hull was lengthened about 10 feet, and adjustments were made in the machinery to meet these conditions. The total cost of building the new hull and reconstructing the entire dredge, including cost of machinery, which was purchased at a low figure, is stated to have been approximately $100,000.

CASTLE CREEK DREDGE.

The Castle Creek dredge, built in 1912 at Mystic, S. Dak., did not prove the success anticipated and was moved to John Day, Oreg., and rebuilt for the Empire Mining Co. The rebuilding of this dredge was done under contract by the Stearns Roger Manufacturing Co., of Denver, the original constructors of the dredge; and the total cost of the dredge rebuilt ready for operation at John Day was figured at considerably less than the cost of a new dredge. At the time of writing, complete details were not available, but it is of interest to note that dismantling the dredge machinery and the power house and loading the equipment on cars at Mystic cost $4,500. The framed timber of the front, center, and rear gantries, the main trusses, and the screen frame, amounting to nearly 45,000 feet, was taken, and more lumber would undoubtedly have been saved except for the high freight rate to the new locality.

FEATURES TO BE CONSIDERED IN REMOVAL OF DREDGING EQUIPMENT.

If the machinery is in good condition, the company owning it can afford to consider rebuilding the dredge in a new locality, as a substantial saving can be made as compared with the cost of a new dredge. Where it is necessary to purchase the machinery of an old dredge, except in exceptional circumstances and at a reasonable price, it is questionable how great a saving can be made.

Among the points other than the condition of the machinery that are to be considered but are liable to be overlooked in valuing an old dredge, are the costs of dismantling and hauling to the nearest railroad; also parts of the superstructure of some dredges can be used to good advantage if the material is in good condition and freight charges not too high. The hull of a dredge is of no value unless the dredge can be moved bodily—that is, can dig its way to a new location. This was done by the Pennsylvania dredge at Oroville, Cal.

OTHER EXAMPLES OF MOVING DREDGES.

The Pennsylvania, after the operating company's property had been worked out, was purchased by a group of Oroville dredge men for $7,500 and paid for by a stock interest in a new company they then formed. The dredge dug its way to the new ground through a distance of 1 mile, the greater part of which was dredge tailing, in three months. About five weeks was required to overhaul the dredge before operations were begun on the new property. This dredge, although an old machine, having been first put in operation in November, 1902, will with care last the life of the property, and is already making a substantial profit for the new owners.[a] The advantages in this case were exceptional; an area of 40 acres of known value as dredging ground that the owner had for years refused to sell was purchased on satisfactory terms by experienced dredge men, who also had the opportunity to buy the dredge just as the ground for which it had been built was worked out.

Dredges built in Alaska which proved unprofitable in the original localities have been moved to new areas. One of these, the Osborne Creek dredge, was first built in lower Dry Creek at Nome. The ground had not been prospected before the dredge was built and when the dredge was started the gravel was found to be of too low grade for profitable dredging. The dredge was afterwards moved to Osborne Creek, about 12 miles distant. The machinery was first removed and the hull sawed in half and hauled in two sections. Each section weighed 55 tons and was drawn by 66 horses.

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[a] Since this was written the dredge has worked out the property and has again been sold.

In 1912 the No. 7 dredge of the Yukon Gold Co. was moved at considerable expense from Dawson, where it had worked out the property for which it was built, to Flat Creek, where it has since operated successfully. Other dredges of the Yukon company are being moved to Idaho from Bonanza Creek in the Klondike.

The following tables show the approximate maximum values given to different used dredge parts in California; the first covers a 5-foot dredge and a 5½-foot dredge, and the second a 9-foot dredge in excellent condition, with a new manganese-steel bucket line. It is thought that the figures may be of assistance to persons desiring estimates under like conditions, but it should be remembered that the purchase of second-hand dredge machinery requires considerable study.

Approximate value of second-hand parts of two dredges used in California.

Name of part.	5-cubic-foot close-connected remodeled dredge built in 1911.		5½-cubic-foot close connected remodeled dredge.	
	Condition.	Approximate value.	Condition.	Approximate value.
	Per cent.	*Dollars.*	*Per cent.*	*Dollars.*
Buckets	Poor.			6,000
Pins				
Upper tumbler		300	New.	800
Lower tumbler	Poor.			
Ladder			Good.	2,000
Ladder end casings	O. K.	50		250
Ladder suspension shape	O. K.	50		75
Ladder rollers and bearings				300
Bull wheel	Fair.	150		500
Intermediate shafts and bearings		250		500
Pulley		125		
Hopper	([a])	100		200
Screens	25	250	25	500
Screen drive	50	200		250
Ladder hoist		500	Poor.	
Main winch		900		
Operating levers	50	200		200
Ladder suspension tackle		500		500
Stacker suspension tackle		100		150
Spuds	None.		Good.	750
Deck sheaves	50	100		100
Distribution and gold-saving tables	20	100		100
Save-all		100		75
Spud casing	None.			100
Tail sluices				
Stacker chute				
Riffles				
Stacker ladder	75	500	25	600
Stacker rollers	None.		None.	
Stacker gearing		50		75
Bow gantry cap		100		125
Stern gantry cap		75		100
Pumps and piping	50	400		500
Ropes and bolts	20	75		75
Electric motors		1,500		2,000
Electric wiring		50		100
Tools and sundries		300		400
Hull and housing		50		100
Total		7,175		17,425
Approximate weight of all parts listed ... tons	50		150	
Cost of hauling to railroad	[b] $5	[c] $250	[b] $4	[c] $600
Cost of dismantling		$750		$1,500

[a] Part of hopper in fair condition. [b] Per ton. [c] Total.

Approximate present value of machinery and equipment on a 9-foot dredge built to dig 55 feet below water level.

Name of part.	Weight.	Cost new.	Salvage value.	Present value.
	Pounds.		*Per cent.*	
Buckets, 2 pieces	[a] 79	[b] $17,380.00	75	$13,035.00
Buckets, 3 pieces	[a] 16	[c] 3,456.00		Scrap.
Bucket pins	[a] 95	[d] 2,470.00	25	617.50
Bucket bushings	[a] 190	[e] 566.20		Scrap.
Digging ladder	150,000	9,000.00	50	4,500.00
Ladder rollers, complete	[a] 15	[f] 1,320.00	25	330.00
Ladder suspension, bars and blocks	45,000	4,275.00	50	2,137.50
Lower tumbler and shaft		1,750.00		Scrap.
Upper-tumbler body		1,000.00		Scrap.
Upper-tumbler shaft		1,000.00	75	750.00
Main drive, complete	45,200	4,300.00	50	2,150.00
Bucket ladder, complete	10,500	1,325.00	50	662.50
L. H. winch, complete	42,000	5,040.00	60	3,024.00
S. L. winch, complete	40,000	4,800.00	60	2,880.00
Main hopper	15,000			Scrap.
Revolving screen, complete	47,700	2,860.00	50	1,430.00
Revolving-screen drive	12,600	1,300.00	50	650.00
Distributor	36,000	2,160.00	40	864.00
Stacker ladder	46,000	2,400.00	50	1,200.00
Stacker-hoist winch	3,200	400.00	60	240.00
Stacker suspension, complete	5,000	500.00	60	300.00
Stacker drive, complete	16,500	1,450.00	50	725.00
Stacker belt				Scrap.
1 spud[g]	62,000	3,100.00	50	1,550.00
1 spud suspension	4,500	450.00	60	270.00
Deck depression and fairlead sheaves	29,500	2,500.00	50	1,250.00
Operating levers	11,500	1,150.00	50	575.00
Motors		8,000.00	65	5,200.00
12-inch high and low pressure pump	11,000	1,300.00	50	650.00
5-inch hopper pump		500.00		Scrap.
Total				44,990.50

[a] Number.
[b] At $220 each.
[c] At $216 each.
[d] At $26 a pound.
[e] At $2.98 each.
[f] At $88 each.
[g] One spud was valueless, being cracked in several places.

OPERATION OF GOLD DREDGES.

The material following is intended to describe dredging operations in a general way, although elsewhere in this bulletin will be found material that could properly be classed under this head.

The principal factor in operating a gold dredge is to maintain the highest possible proportion of operating time; that is, to keep the dredge digging. As mentioned elsewhere, a dredge idle is making no money, but fixed and interest charges still go on. A few years ago an operating time of 75 per cent was considered a high average. The average operating time of the modern dredge has increased to 80 or 85 per cent, mainly because of improvements in machinery and design and general strengthening of dredge parts that are subjected to great strain, although the weight of the machinery and the first cost of the dredge have greatly increased. The modern gold dredge has increased in weight from the early boats with a total of 300 tons of machinery and hull to the complete 16-foot dredge which will dig 80 feet below the water level and which weighs 2,250 tons.

ESSENTIALS OF DREDGE OPERATION.

With a dredge suitably equipped, and placed, the essentials of operation may be summarized as follows:

(1) Outline the course with due regard to elevation of surface.

(2) Secure side lines to as long head lines as practicable.

(3) Step to insure a full bucket only.

(4) Run buckets at full speed in free ground; run more slowly in hard ground.

(5) Swing fast for a shallow drop and run slowly with deeper drop. The proper drop can be determined only on the ground.

(6) Cleaning of bedrock depends on situation of pay ground and careful cleaning of bedrock, digging well into it, may or may not be advisable.

(7) Motors should be in charge of an experienced electrician. On small dredges conveniently situated, an electrician can come at regular intervals and on special calls. The regular oiler can look after the oiling of motors daily.

(8) Keep repair parts in liberal supply, and of the best material.

AMPLE SPARE PARTS SHOULD BE AVAILABLE.

Perry[a] points out that in order to make a success of dredging operations far from sources of supply, the equipment must be complete and of the best possible kind. A single breakdown or loss of a part that can not be readily replaced may mean the loss of the season's work, so that the supply of extra parts and the shop equipment for repairs should be much greater than is ordinarily required. An ample supply of spare parts is advisable in all gold dredging, as can be illustrated by a case in point. During 1915 and 1916 the Natomas Co. operated under contract the 9-foot dredge of the Wilkes-Barre Co. This dredge was similar in design to several Natomas dredges, and the Natomas Co., with its complete organization and shop equipment, could work to far better advantage than a company operating a single dredge could possibly have done. For instance, when there were breakdowns of the Wilkes-Barre dredge, the repair gang would be rushed to it as though it were a Natomas dredge; and when in the fall of 1915 the bucket line began to go to pieces all at once, the Natomas Co. was able to supply from its large stock on hand enough extra buckets to keep the dredge running until the new buckets arrived, whereas if a company with a single dredge had been operating, a shutdown of two months or more would have probably resulted while new buckets were awaited.

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[a] Perry, O. B., Development of dredging in the Yukon territory: Trans. Canadian Min. Inst., vol. 18, 1915, p. 30.

Most dredges built in accordance with California practice are now equipped with digging spuds, in place of the head lines used on the early dredges. In easily dug light gravel, as that worked by the 16-foot Boston & Idaho dredge at Idaho City, the dredge was operated on head lines, and it was not necessary to furnish spuds, though the dredge had been designed for spuds and spud casings had been supplied when the dredge was built. Head lines have sometimes replaced spuds successfully when the dredge hull has become too old to withstand the strain of digging against the spud; and during the repairing of a broken spud, digging has sometimes been continued by rigging up a temporary head line.

DIGGING ON SPUDS.

The digging face is arc shaped; sometimes a single cut is made and at other times double cuts are carried. In carrying the excavation forward, the dredge is swung from one side of the cut to the other by side lines. Figure 15 shows a method of dredging to leave a channel. The spuds are raised or lowered by wire cables running through large sheaves on the rear gantry to the spud drums on the main winch. The digging spud is generally heavier than the stepping spud and is made of steel. Formerly the stepping spud was made of wood. The practice on large modern dredges is to have both spuds of steel and of similar weight and design; thus should the digging spud break the dredge can work on the other. The advantage is apparent.

To move the dredge forward, the spud is raised and the line on one side of the dredge is hauled in while the other side line is played out, the second spud being used as a pivot on which the dredge is moved. The first spud is then dropped, the second one raised, and the operation repeated. With each swing the dredge is stepped ahead 2 feet or more, depending on the width between spuds. For the new cut the total distance of moving the dredge is 5 to 8 or 10 feet. On the Yuba dredges the spuds are placed about 14 feet apart and it is necessary to swing through an angle of only about 40° to make a 6-foot step. The lower tumbler is usually greased during this operation, which requires about 12 minutes.

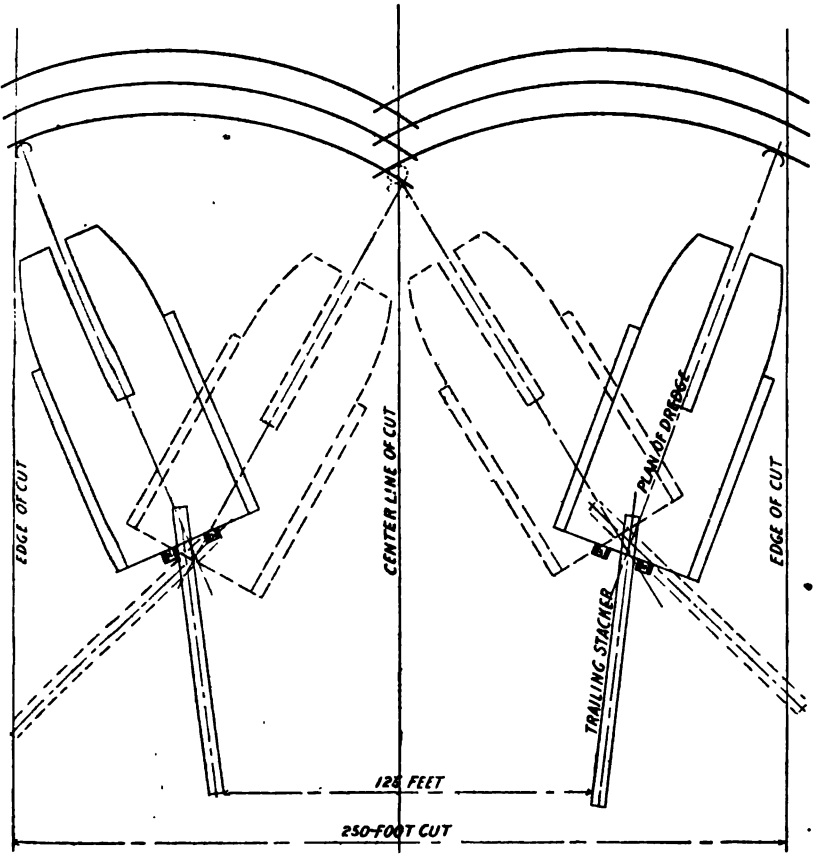

FIGURE 15.—Method of operating gold dredge to leave a channel.

At the Derry Farm dredge near Leadville, Colo., the method was to lift both spuds and to pull with the stern lines until the dredge was in the desired position, when the spuds were lowered. It was thought that a saving of nearly 10 minutes each time the dredge was moved resulted by adopting this method.

DUTIES AND REQUIREMENTS OF WINCHMEN.

In running a dredge care is taken to prevent the tailings from the sluices and stacker from so filling the dredge pond as to cause the boat to ground when it is being moved. At times it is difficult to retain sufficient flotation space, and an inexperienced winchman may ground the boat if gravel considerably deeper than usual is dredged, and if the tailing stacker is too short to dispose properly of the tailing, or if the fines are greatly in excess of the coarser material and the sluices are short the excess of tailing may ground the boat. A sand pump to dispose of fine material is occasionally necessary, and simple elevator buckets to deliver the fines from the side sluices to the conveyor belt were designed by Walter Johnson for a dredge in Alaska at which a large force of men were formerly required to shovel back the fines in order to prevent the boat from grounding on them. This plan resulted in a great saving of labor and increased the yardage.

The manipulation of the bucket line often accounts for a considerable difference in yardage handled and in the recovery of the gold content of the gravel.

The position of winchman is the most important on the dredge, and the yardage dug depends almost entirely on the winchman's knowledge of his work and the character of the gravel in different parts of the cut. It is important that the winchman be kept interested in his work for if he is lax in operating the bucket line considerable loss will result.

REGULATING SPEED OF BUCKET SWING.

On every electric dredge a wattmeter or similar recording meter, connected to the digging motor, should be placed so as to be easily seen by the winchman as a guide in his digging. It will tell him when his motor is pulling its regular load and will warn him of any unusual stress in the motor such as might be caused by running into a cave-in of gravel below the surface of the water which may not be visible above the water line. It will also warn him when his buckets are running out of gravel, and will help in many ways to maintain smooth operation.

In some ground it is possible to devise a constant speed of swing for the bucket winches, instead of letting the winchman wait for empty buckets to come up before swinging. Here a wattmeter is particularly useful, as it will show a decrease in load the instant the buckets are out of gravel. However, when bedrock is being cleaned, there is some question as to just how far a constant speed can be devised, but familiarity with the ground enables the careful operator to judge the proper time for moving the buckets to a new point of attack. To mention an instance, Walter Johnson was able to increase the yardage at the Bangor Creek dredge on the Seward Peninsula by 300 yards a day after a careful study of the conditions and after determining a proper speed of bucket swing. Similar efficiency has been obtained by the large California companies as a result of careful study, and the yardage has been considerably increased by having the buckets take smaller cuts and increasing the speed of swing. (See Pls. LVII and LVIII.)

In this connection Summers[a] makes the following statement:

> In the earlier days of dredging the rule was to drop deep and it was necessary to use the slow gear to pull the buckets through the gravel. It was found that this was wrong, as it put an unnecessary strain on bow lines, winches, and ladders, and in fact on every part of the machinery used in digging and elevating the gravel. The old cry in dredging was for more yardage, and as soon as a winchman had his digging ladder down to its limit he would swing to the other side of the cut and step up. In using low gear on the swinging it is possible to pull the buckets through the gravel faster than they can pick it all up, and in that way gravel was left in the bottom of the pond.
>
> As the buckets will hold a limited amount of gravel it is necessary to use only power enough to fill them, and this is most economically done by taking light drops and swinging on high gear. In this way it is impossible to pull through a lot of gravel on the bottom of the cut as the fast gear is not strong enough, and when it is stalled it is necessary to raise up enough to allow it to swing freely. In using the older method the power consumption was much higher than necessary and the results were poorer on account of more time lost owing to broken bow lines, buckets slipping off the lower tumbler, broken bucket lines, wear on bucket pins and bushings, and repairs incidental to the increased power strains.

On some dredges the superintendent has found it advisable to take the low-speed gear off the ladder winch, as the winchman when meeting an obstruction in hoisting would change to low speed and the resultant powerful pull would at times cause a break in some part of the machinery. Without the low-speed gear, he is obliged to relieve strain by reversing.

In this connection Summers[a] says:

> In stepping ahead on the smaller boats a 5½-foot step is usually about the best length in ordinary gravel. In using a much smaller step the boat is liable to surge back and forth. This is caused by the bucket engaging the gravel and passing beyond the point of its strain before the next bucket following engages the gravel. As soon as the bucket engages the gravel, it forces the boat back hard against the digging spud; then when the bucket has cut into the gravel beyond the point of its greatest strain, the boat will surge ahead until the following bucket will hit the gravel, and then the boat is forced back against the spud again. This condition continues unless a step is taken that is about twice the length of the pitch of the buckets; this results in two buckets engaging the gravel at all times. Speed of the bucket line is varied somewhat by conditions but ordinarily 60 feet per minute is the usual rate of travel. Side-swing speed and depth of drop also vary. Side swing is usually about 30 feet per minute and the depth of drop about 8 or 9 inches in solid gravel. Recent practice is to drop light and swing on high gear.

Regarding the causes of delays to dredging Jennings[b] presents the following table covering the work of dredges of the Conrey Placer Mining Co. in the Ruby, Mont., district:

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[a] Summers, M. L., personal communication.

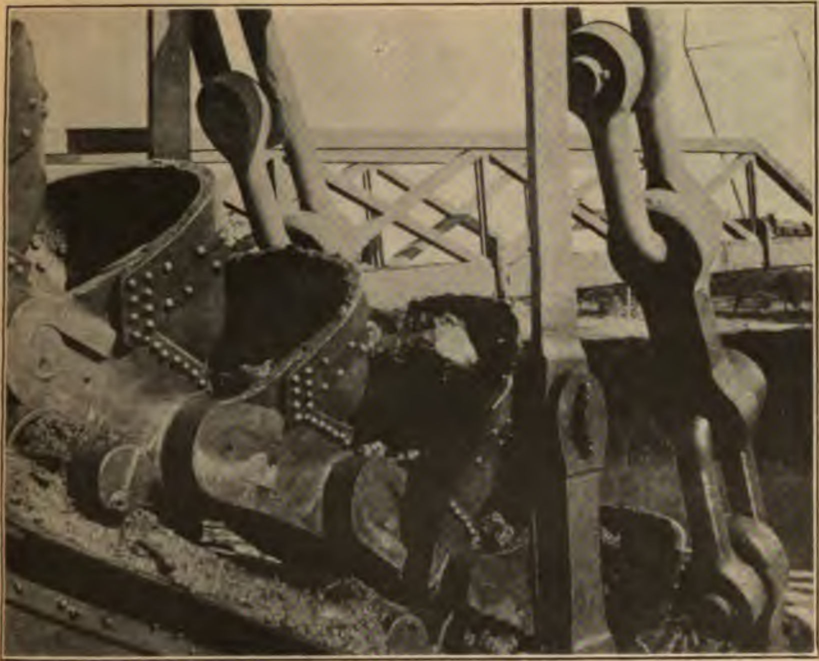
[b] Jennings, Hennen, History and development of gold dredging in Montana: Bull. 121, Bureau of Mines, 1916, p. 28.

Statement of running time and delays, Conrey Placer Mining Co., Dredges Nos. 1, 2, 3, and 4, Jan. 1 to July 31, 1915.

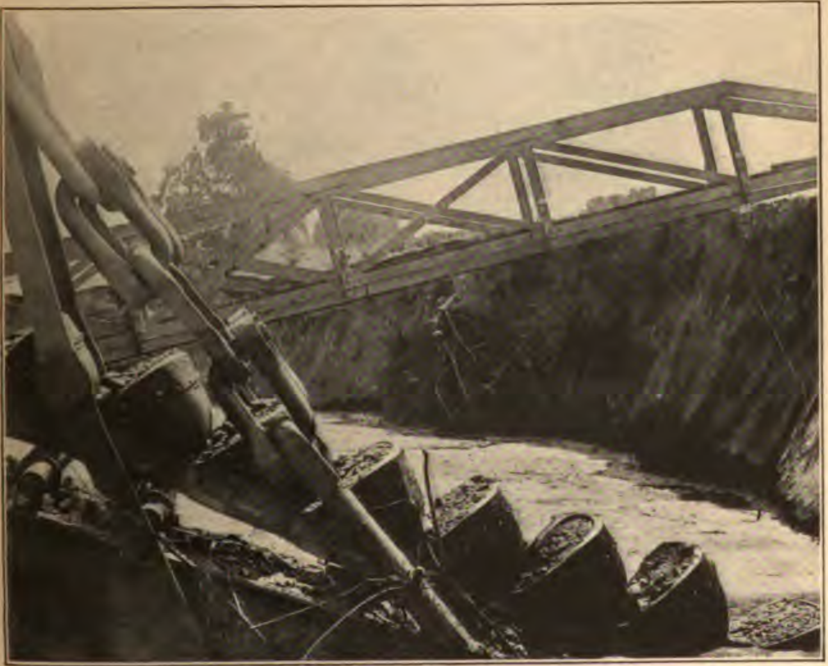
Cause of delay.	Time, per cent.			
	Dredge No. 1.	Dredge No. 2.	Dredge No. 3.	Dredge No. 4.
Clean-up	1.56	1.36	0.87	1.60
Moving ahead	3.89	6.33	2.75	3.80
Lines and spuds	3.42	3.01	2.26	2.95
Winches	.40	.41	.61	.13
Flume	....	....	.48	....
Ladder	.02	.14	.23	1.07
Tumblers	1.02	1.58	.70	5.91
Hopper	.16	1.24	....	.87
Grizzly	3.99	3.25	3.03	3.41
Bucket chain and drive	.62	3.05	2.52	2.27
Pumps and piping	.67	2.05	.38	.49
Electrical appliances	1.36	3.13	1.13	1.09
Heating system	....	....	....	.03
Débris	3.66	2.68	.36	.11
Power off	.68	.66	.53	.57
Stacker	.85	1.48	....	.41
Tables and sluices	.01	.39	.07	.97
Well	.02	.08	.18	.85
Gang plank	....	.06	....	.02
Unloading coal	.09	.15	.02	....
July 4	.71	.78	.78	.71
Hull and housing	.07	1.00	....	.01
Low water	.03	.38	....	....
Winter conditions	....	....	42.46	....
Funeral, President J. J. Myers	.01	.01	01	.01
Miscellaneous	.02	.01	.02	....
Total delays	23.26	33.23	59.38	27.28
Total running time	76.74	66.77	40.62	72.72
Total time	100.00	100.00	100.00	100.00

PRACTICE AT RUBY, MONT.

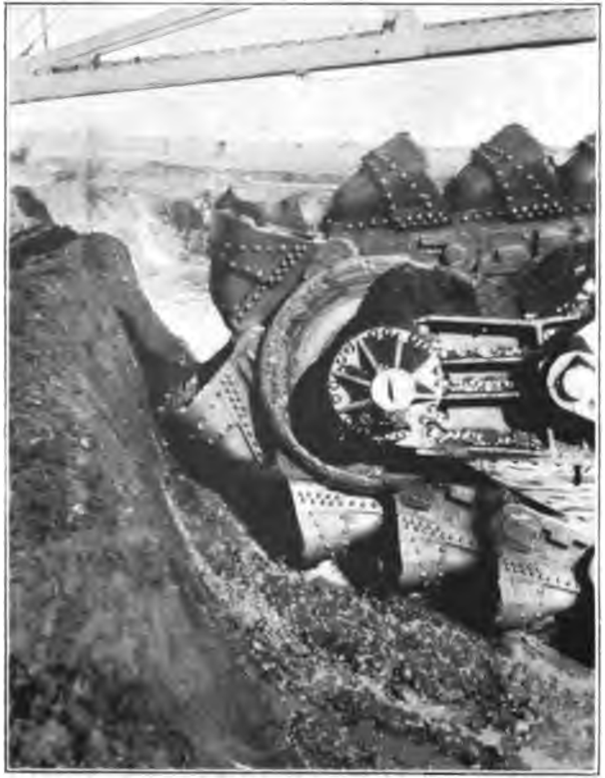
At Ruby, Mont., where the big No. 4 dredge is working there is about 16 feet of overburden and loose gravel. In digging this the bucket chain is run at the limit of 22 buckets per minute and the buckets can be readily filled at that speed. Below 16 feet the gravel begins to get very tight and is mixed with considerable clay, and the buckets are not filled when running at a speed of 22 buckets a minute; hence the speed is reduced to 20 buckets a minute; and near bedrock where the gravel is still tighter the buckets are run as slowly as 18 a minute, making an average of about 20 buckets a minute. It is found that it is better to reduce the speed of the buckets in hard digging and to fill them than it is to keep up the speed and to fill the buckets only half full, as the yardage handled is just as great, if not greater, with considerably less wear on the buckets and tumblers. It is necessary to dig up 6 to 12 inches of bedrock, depending largely on the character of the bedrock. If the bedrock is uneven it is necessary to dig deeper in the higher spots in order to be sure to get all the gravel in the depressions. In the Yukon along the Klondike River the gold in places extends into the bedrock for 2 or 3 feet whereas the upper gravels contain little or no gold. The bedrock is a soft decomposed mica schist and is easy to dig.



A. PARTLY FILLED BUCKETS, HARD GROUND.



B. FULL BUCKETS, LOOSE GROUND.



A. DIGGING HARD CEMENTED GRAVEL.

Note shape of face of cut and buckling of bucket line under tumbler.

B. LARGE BOWLDERS DUG AT NATOMA.

BUCKET-CLEANING METHODS.

The Natoma No. 8 dredge has worked in ground composed largely of sticky mud which can not be easily dug or readily sumped from the buckets. In addition the water in the pond becomes so muddy that it blocks the pumps, adds to the cost of operation and repairs, and reduces yardage materially. To overcome the sticking of clay to the buckets, an additional high-pressure pump has been placed on the dredge to deliver water through two nozzles, which are directed into the bucket as it is turning over the hopper. This arrangement proves of considerable benefit, and although it requires 250 additional horsepower it shows a net saving per cubic yard. A comparison of yardage records and power costs in 1913, before the change, with those in 1914, after the change, is particularly interesting.

Yardage record and operating costs per cubic yard, Natoma No. 8 dredge.

Year.	Time dredging.	Depth dug.	Material dug.	Material dug per hour.	Costs per cubic yard, cents.							
					Labor	Material.	Power.	Water.	Total.	Repairs.	General taxes, insurance.	Total.
	Hours.	*Feet.*	*Cubic yds.*	*Cu. yds.*								
1913........	6,616	62	1,807,235	271	1.01	23	1.49	14	2.87	2.81	86	6.34
1914........	6,370	67	2,053,568	322	.84	15	1.32	10	2.41	4.01	72	7.14

In 1914, as compared with 1913, the dredge dug 50 yards more per dredging hour, and consumed less power, with a running expense of 56 cents less per cubic yard, though the total expenses increased by 0.80 cent over those of 1913 on account of the necessary improvements, changes, and repairs.

On the hopper of the Viloro dredge, Oroville, to cut the clay from the buckets it was necessary to install two 1½-inch nozzles, fed by a two-stage Byron-Jackson pump delivering 1,200 gallons a minute and driven by a 100-horsepower motor. To prevent splashing the hopper was built up on the sides and covered. A part of the ground worked by the Viloro dredge contained 5 feet of indurated clay above the pay gravel, and underlying 5 feet of top soil. To facilitate dredging, the top soil was first removed by the bucket. Auger holes 8 feet apart in lines 5 feet from each other were then put down in the clay and blasted. The clay which prior to blasting was difficult to handle could then be easily dug. The cost of the extra work included the time of three men at $2.50 a day and three to five boxes of powder at $4.50, or an average of about $25 a day extra, but the extra expense was more than justified by the increased yardage handled. When the Yuba dredge was handling a stratum of clay that contained no gold, at a depth of 45 feet to 60 feet below water, an iron plate was temporarily used to block the hopper and the material was dumped into the well hole and not sent to the screen and tables.

In digging stiff clay, the dredge master generally finds it good practice to take light cuts, as buckets dump cleaner than if deep cuts are taken, when the buckets clog and the clay may also get lodged in the hopper.

When the Isabel dredge was handling a larger quantity of barren clay in Calaveras County, Cal., an improvised clay cut-out worked satisfactorily. It is described by Winston.[a]

The principle of the cut-out is shown in figure 16. A deflecting plate, *a*, operated by a lever, *d*, causes all fines from the shaking screen with which the dredge is equipped to pass either to the gold tables or, when the operator so desires, to a separate clay sluice, which extends a considerable distance beyond the stern of the dredge.

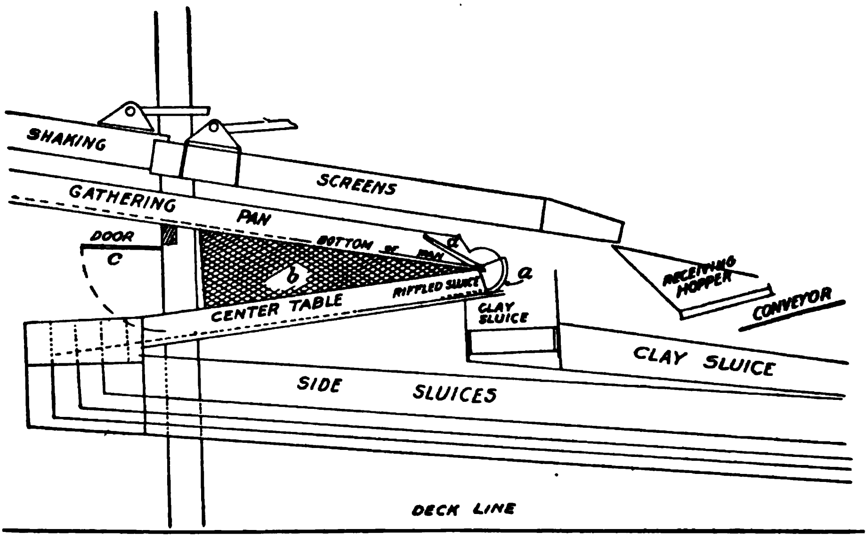


FIGURE 16.—Clay cut-out device for gold dredge.

Large lumps of clay of course are not broken at the screens, and pass directly to the tailing conveyor. The device shown, with properly placed hopper sprays for cleaning clay from the dumping buckets, has obviated most of the delay and reduced the gold losses due to clay robbing the sluices.

USE OF MONITORS AND OF BLASTING IN CEMENTED GRAVEL.

In handling hard, compact gravel, monitors were placed on the front of some dredges, especially in the Folsom district. Although these were of some use in washing down high banks of gravel of a medium compactness, they were of little use against cemented gravel. A number of years ago at Oroville blasting was tried with considerable success in handling cemented gravel, but the extra cost was 2 to 5 cents a cubic yard. The method of operation was to drill holes 5 to 15 feet into bedrock and placed 25 to 250 feet apart. Charges of 50 to 120 pounds of 40 per cent dynamite were used in the holes, the charges being fired either by electricity or by fuse. Blasting in front of the Natomas dredges, which were dredging in hard cemented material, did not give very satisfactory results, as the gravel broke into large chunks which went through the screen without disintegrating.

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[a] Winston, W. B., Clay cut-out, Isabel dredge: Min. and Sci. Press, vol. 101, Dec. 24, 1910, pp. 838–839.

LIFE OF HULL.

The life of a wooden hull in California is generally estimated at 10 years for small dredges and 7 for large ones, and a steel hull for either size is expected to last 10 years. Wooden dredge hulls have lasted longer than the period mentioned, as, for example the old Continental, which lasted more than 10 years, the Pennsylvania, which is still working, though built more than 13 years ago, and the Viloro hull, which was 11 years old when the dredge was burned in October, 1914. At Breckenridge, Colo., the hull on the No. 3 dredge is 15 years old and is seemingly still in good condition. On account of climatic conditions there has been no evidence of dry rot, and when it was necessary to overhaul the hull about the stern no evidence of dry rot was found. This condition is not usually found in dredge hulls. A dredge hull is ventilated by means of blowers at the stern of the boat, which force fresh air through the hull, the air coming out through the ventilators on the bow. Such ventilation is essential for dredges operating in a warm or humid climate, and neglect to ventilate the hull properly has caused serious damage on more than one dredge. The general practice is occasionally to paint the interior of the wooden hull with hot crude oil, which considerably preserves the life of the timbers.

On dredges operating in cold climates, a steam-heating plant is arranged either by having the boiler placed in the hull or on the main deck, and by having coils of steam pipe placed around the interior of the dredge and at suitable places in the housing and stacker.

Few dredges work throughout the winter in regions where that season is extremely cold. As exceptions may be mentioned Dredges Nos. 1, 2, and 4 of the Conrey Placer Mining Co., at Ruby, Mont., where heating by steam pipes is employed. The steam circulates under the gold tables and along the stacker, which is supplied with a canvas housing. Electric heaters are used on some dredges with success. Miles[a] describes the method of operating the Boston &

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[a] Miles, J. H., Winter dredging in Idaho: Min. and Sci. Press, vol. 108, Mar. 14, 1914, pp. 455–456.

Idaho dredge in Idaho during the coldest weather. It was afterwards found advisable not to operate during the winter months.

REPAIRING AND MAINTAINING HULLS.

Dredge hulls frequently require repairs, either near the stern where the spud strain develops, or near the bow when the weight and strain on the front gantry causes a springing of the hull and starts leaks. On one dredge the hull was repaired by pouring a rich mixture of cement between a bulkhead formed by the side planking of the dredge and timbers placed on the inside of the hull frame. Generally the hull is repaired by building in solid with new lumber against the inside shell of the boat, though at times it is necessary to pump all the water from the pond, and make extensive repairs on the hull, especially if the dredge has capsized.

Steel hulls also have given trouble at spud casings and made dry docking and extensive repairs necessary. Consequently, these parts have been strengthened and heavy spud-casing springs have been placed both on the hull and on a level with the upper deck, to take off much of the strain caused by pressure against the spuds during digging.

On nearly all dredges, arrangements are made to handle any leaks that may develop in the hull, by having either separate emergency pumps or pump suctions attached or attachable to the monitor pump. Some operators have a float so arranged that a bell will notify them if water rises on the hull, whereas others trust to regular investigations. On the smaller dredges a hand pump on the bow generally handles all ordinary seepage, although there is also an emergency pump for use if a leak develops.

On some of the older dredges continual pumping is required in order to handle the water coming in the hull, and a number of dredges have capsized owing to leaks caused by sudden strains to the hull planks. Although all operators appreciate the need of properly ventilating the hull, some believe that it is unnecessary to pay much attention to the dredge housing, as wood housing will ordinarily outlast a hull two to one.

Most operators take pride in the appearance of their boats, and paint the housing at least once every two years; others, especially on headline boats, or boats having small flotation and on those handling considerable clay, feel that it is useless to try to keep the dredges clean. Opinions regarding the care given machinery also differ, and one successful dredge operator feels that any unnecessary care given to the machinery merely to improve its appearance without adding to the operating efficiency of the dredge, is useless. The larger companies make it a rule to keep the dredge and machinery as clean and neat as possible. The men take a much keener interest in the work if everything is kept clean and tidy. Some companies have notices in conspicuous places. "A place for everything and everything in its place." After a breakdown, there is always more time lost if there is a mess of grease and grime that has to be removed before the work can be properly done. Most men have an aversion to greasy muck, and dread starting a repair job in the midst of a lot of unnecessary dirt.

HANDLING OF LARGE BOWLDERS.

The handling of large bowlders in dredging is always somewhat difficult. The method used on the dredge of the Andrada mines (Ltd.), in Portuguese East Africa is worthy of mention. When a bowlder too large to be lifted is encountered, it is usually either passed over or is pushed to one side; but in ground containing many medium-sized bowlders, the bowlders must be raised in order to recover the mineral content of the ground, especially when the bowlders are near bedrock. In 4⅔ months, from February 10, 1914, to the end of June of the same year, the Andrada dredge, a close connected 7½-cubic foot boat, treated 547,700 cubic yards in a ground full of bowlders and of clay, and picked up 4,400 bowlders weighing on an average of more than a ton each.

Elimination of bowlders from the buckets is the most delicate part of the work. The ordinary method in dredging is to stop the bucket chain, and, after tying the bowler with slings, to raise it with chain blocks suspended to a moving crane, and to set it on the deck. When the bowlder is very big and is squeezed in between two buckets, this operation requires at the least 10 minutes. By this method the Andrada dredge has pulled up more than 200 bowlders of an average weight of one ton, and the time lost has averaged just about 10 minutes a bowlder.

In February, however, the number of bowlders found became so large, that the removing of them by crane, operated at first by hand and later by winch, caused such loss of time that the actual dredging fell to about 50 per cent of the capacity of the boat. At that time cracking the bowlders in the buckets by blasting, and discharging the pieces through the screen and stacker was tried. A small dynamite cartridge was placed on the bowlder, covered with clay, and fired. The bowlder was not broken in small pieces, which flew and injured the dredge, but was only cracked and caused to settle in the buckets.

During March, 1914, the Andrada dredge picked up 1,510 bowlders, fired 1,544 dynamite cartridges, and lost only about 1,800 minutes, or, say, 60 minutes a day, or 1.2 minutes a bowlder. During the month the dredge treated 131,000 cubic yards of ground averaging 15 feet deep. In four months the chain of 62 buckets withstood 4,600 blastings (nearly 72 blasts per bucket), and only 5 buckets were pierced in the hood. The holes then made were only about 5 inches diameter and were easily patched and did not interfere with the working of the buckets.

The blasting of bowlders in buckets can be undertaken without danger, provided care is taken as to the quantity and placing of the dynamite used and the buckets are strong enough.

NIGHT LIGHTS.

In working at night, either arc lamps or clusters of incandescent lamps are used, the arc light being the more economical. In addition, it is good practice to provide the dredge with a searchlight placed on the front gantry and operated from the pilot house. This searchlight facilitates the work of the winchman and furnishes light for shore work, such as moving power cables and shifting side lines. After accidents to the electrical equipment the large acetylene lights have been found useful in dredging operations.

METHOD OF MAKING REPAIRS.

In California the small dredging companies run the dredges continually until these require a complete overhauling, when they are shut down for 10 days or more while the necessary repairs are made (Pls. LIX, *A*, and LX, *A*). The large companies, and some of the small operators have a separate repair gang which is always engaged on the boat during clean-up periods, when minor repairs, such as changing bucket lips and tumbler bearing plates are made. Extensive overhauling is often started during clean-up periods, on the principle that repairs should be made as soon as needed. The repair crew of the large California companies consists of eight or nine men, who soon gain a knowledge of all the different methods of repairing used on the different dredges. The maintenance of a crew of trained repair men greatly shortens the time lost on shutdowns. The dredge masters are encouraged to try new methods for hastening any detail of repairing or operation, and in that way there is an incentive for every man to try to improve on the methods in use.

In the northern fields, as pointed out by Perry,[a] all the heavy work is done during the winter or closed season. In the autumn, when work is suspended, the dredges are stripped of their bucket lines and conveyors. The buckets, screens, pumps, and all parts subject to heavy wear are taken to the shops and thoroughly overhauled. The dredges are then put in as good order as possible, with a view of running them continuously throughout the season of approximately four months. During the operating season all repairs upon the dredges are minimized. For bucket-line repairs the whole line is

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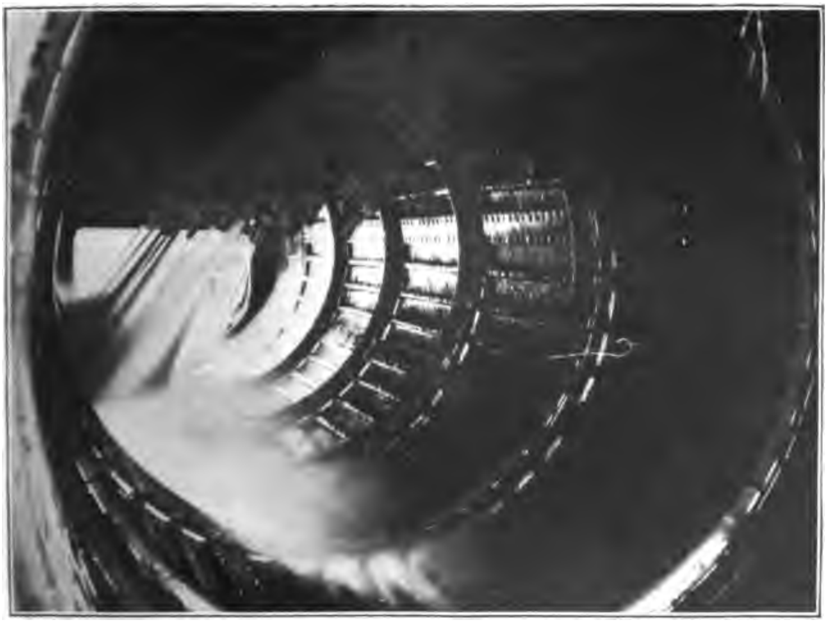
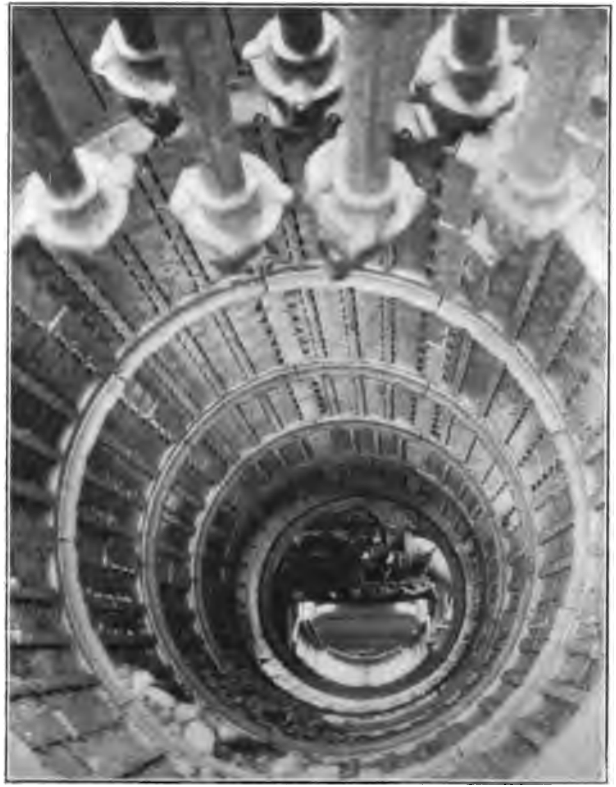
[a] Perry, O. B., Development of dredging in the Yukon Territory: Trans. Canadian Min. Inst., vol. 18, 1915, pp. 32–34.

A. TROUBLE WITH THE BUCKET LINE.



B. WATER NOZZLES IN PIPE IN INTERIOR OF REVOLVING SCREEN.

A. CHANGING BUCKETS.



B. INTERIOR OF A REVOLVING SCREEN, SHOWING END PLACING OF WATER NOZZLES.

sometimes removed and is replaced by a line that has been entirely overhauled, the other line being repaired for the next replacement. This arrangement illustrates the advantage to an operating company of having all boats of the same size with interchangeable parts. The repair crews of some of the large companies when renewing bucket lips use a portable air compressor which they take to the different dredges. This method is much quicker and cheaper than taking the buckets to the repair shop.

Formerly at the Natomas Consolidated there were several different sizes of dredges, each dredge working under different conditions; and nearly every dredge even of the same capacity was built differently and by different companies. Consequently, it was necessary to carry a large stock of supplies. Under new management this disadvantage is being adjusted, and any dredges rebuilt are made in two capacities only, machinery parts being made interchangeable for all dredges of the same capacity. This plan has resulted in an appreciable reduction in working cost.

Broken or badly worn buckets are ordinarily changed as soon as possible or at the first clean-up period. If new parts are not available, or if for other reasons delay seems inadvisable, the line may continue running even with a number of broken buckets. The method of changing buckets varies in different fields. On some dredges the old line is first removed entirely, and the new one added in sections which are first connected on the bank, and are hauled to the ladder by cables from the main winch. Sometimes the buckets are replaced one by one, new buckets being transferred to and from shore by cable. The practice at the Yuba Consolidated is to change buckets in sets of six or seven; that is, the extra buckets are taken to the dredge and replace the same number of broken or worn ones in the line.

On the No. 14 dredge the dredge crew directed by Mr. Williams, the dredgemaster, took off and replaced six 16-foot buckets in 40 minutes. It required 12 minutes only for putting on the new buckets. One hundred and eight 7½-foot buckets were taken out and replaced in 11 hours and 15 minutes. The Yuba company handles buckets by a special small hoist operated by a water wheel fed by a pump, and a small hydraulic hoist is attached to the front gantry to handle bucket pins. Swinging rams, hung on either side of the ladder on the front of the dredge, are used to knock out old bucket pins and drive in new ones.

USE OF DEPTH INDICATOR CHART.

On some dredges a time chart is so arranged by connections to the digging ladder that when the ladder is raised or lowered the depth of digging is indicated (fig. 17).

On the Conrey dredges an indicator is arranged on the upper tumbler shaft automatically to keep count of the number of buckets dumping.

Figure 18, reproduced from a sketch furnished by Charles Kammerer, illustrates the method of ascertaining the yardage dug. It also shows the method developed at Ruby of returning the bowstring lines to a sheave on the lower end of the ladder, before passing to the swing winch, thus taking much of the strain from the bow of the hull. (See chapter on "Operating Cost.")

On many dredges working in a known channel, and making only one cut, panning is practiced to determine the gold content of the ground. Sometimes a small section of the gold table is cleaned daily, and the course of the dredge is changed as results indicate advisable. At Breckenridge a short distance on either side of the richer section being dredged, the gold content falls below operation costs, but in order to insure working to the extreme limit of the good ground, the cut is extended slightly beyond the known limit of the channel.

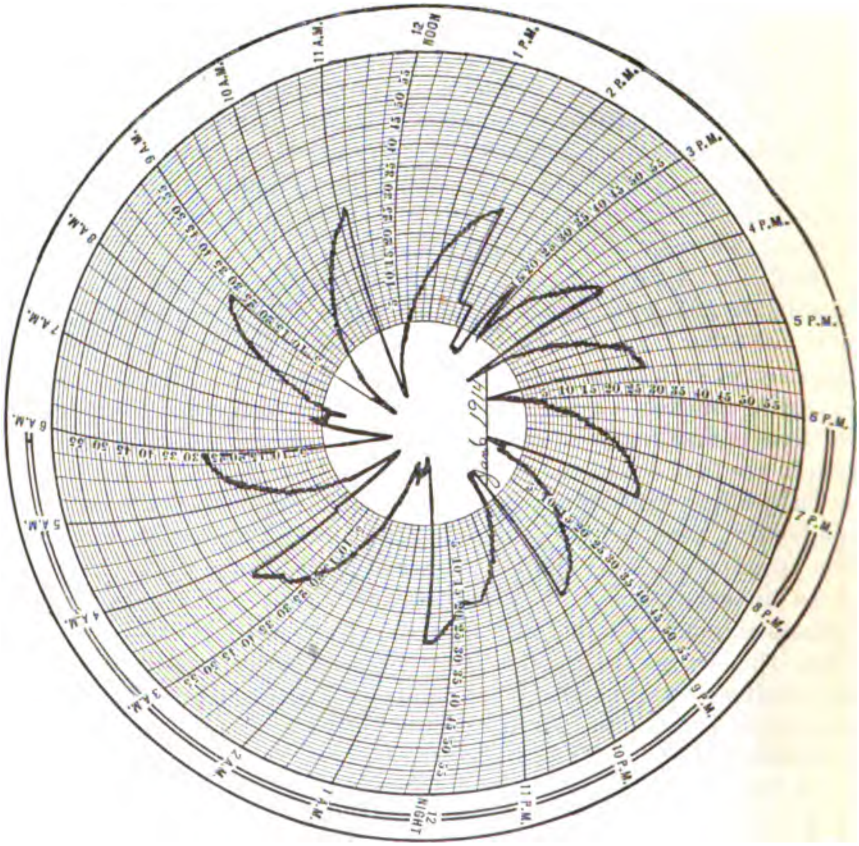


FIGURE 17.—Specimen chart showing record of depth dug by gold dredge.

EXTENT AND CAUSES OF LOST TIME.

When the dredge is in operation, all time not actually spent in digging is generally considered as time lost. Some operators do not consider stepping ahead as lost time, although time spent in clean-ups is nearly always so regarded. The greatest proportion of lost time is due to accidents, delays, and repairs to the digging equipment, including the changing of buckets and the renewing of bucket tips, tumbler wearing plates, etc., but changing of the cushion type upper tumbler plates formerly used took five men 10 hours. The present type of upper tumbler obviates most of this loss of time.

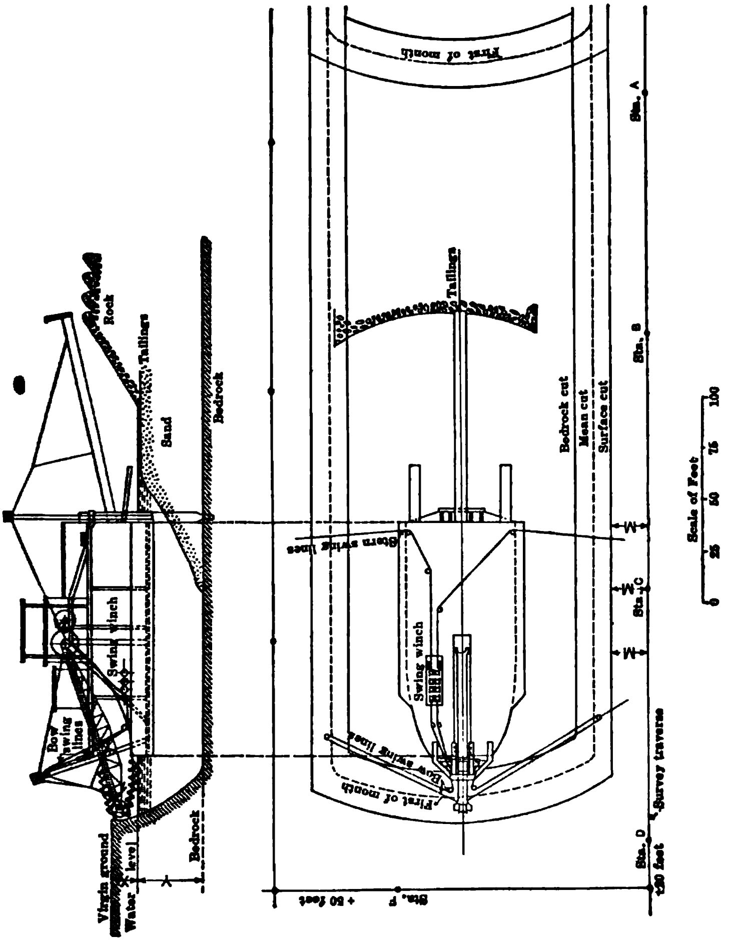


FIGURE 18.—Plan and elevation of 9-foot dredge, showing swing winch and swing lines; also method of calculating yardage handled.

The shutting off of power because of storms and other causes outside the dredge master's control sometimes causes considerable loss of time, though by most operators this is not considered loss of operating time. They merely deduct this time from the total time the dredge was in operation. Other parts of the dredge that require frequent attention and are responsible for loss of time are the screen and the tailing stacker, and also the pumps, particularly when the pond water is muddy.

The table following shows the percentage of lost time attributed to various parts of the machinery in 1914 on dredges 3 and 4 operated by the Marysville Gold Dredging Co. near Marysville, Cal. Loss of time on other dredges is shown in different tables given in the section on working costs. The total lost time on all Natomas Consolidated dredges for 1914 averaged 18 per cent; on all dredges of the Yuba Consolidated Gold Fields, 14 per cent; and on all dredges of the Marysville Gold Dredging Co., 15 per cent of the possible time.

Delays in dredging referred to causes, dredges 3 and 4, Marysville Gold Dredging Co., California, 1914.

Cause of delay.	Per cent of total delay.
Stepping up	13.6
Stacker	4.2
Clean up	11.7
Motors	1.0
Lower tumbler	8.2
Upper tumbler	4.4
Lines	5.1
Pumps	3.1
Screen	7.8
Buckets	8.1
Power	6.6
Rollers	4.6
Sand	.2
Winches	2.0
Upper tumbler drive	2.9
Miscellaneous	16.5

The following notice regarding the recording of delays is issued by the Marysville Gold Dredging Co.:

NOTICE.

In order that reports may be uniform, please be guided by the following when making record of delays on winchman's shift report:

Stepping up.—Include also moving dredge to new position.

Stacker.—Includes stacker belt, stacker drive (not motor trouble), idlers, guy ropes, drums, etc.

Clean-up.—Time actually engaged in clean-up.

Motors.—This refers to all motors on dredge.

Lower tumbler.—Refers to time lost oiling and any repairs to lower tumbler, or lower tumbler bearings or bushings.

Upper tumbler.—Includes all parts of upper tumbler and bearings of same, and upper tumbler shaft.

Lines.—Refers to ladder lines, swinging lines, or stern lines.

Pumps.—All delays caused by pumps (not pump motors), including also screen around intake.

Screen.—Delays on account renewing or repairing revolving screen, screen drive, rollers, bearings, driving gears, spray pipe, etc.

Buckets.—Time lost repairing or replacing; also when off tumblers, unless clearly caused by defective tumbler.

Power.—Delays caused by power being off or low voltage.

Rollers.—Refers to ladder rollers, bearings, chairs, bushings, oiling, etc.

Winches.—Refers to both ladder hoist winch and main winch for swinging lines.

Bucket drive.—Refers to all gears, shafts, bearings, etc., in connection with driving upper tumbler, including magnetic clutch.

Electric equipment and power lines.—Refers to power cables, power lines, switches and switch boards, controllers, transformers, wiring, and all electric equipment except motors.

Spud.—All delays caused by spuds, springs, spud housing, lines, etc.

Hopper.—Delays caused by clogging of hopper or any repairs to hopper.

Ladder.—Digging ladder only; cleaning, repairing; includes ladder arms and chafing beams.

Tables.—Includes also sluices, distributors, and save-all.

Miscellaneous.—All other causes of delays not covered by the above headings, giving cause.

In making bucket-line report, when giving number on base from which a pin has been removed, or put in, use the number on base in which the lug fits.

The following table[a] showing segregated causes of lost time on a small dredge in Alaska is of interest, as it is not only typical of the work of a single dredge in an isolated field, but of a modern, steam-driven dredge operating under severe conditions in the northern field. Out of a total possible dredging time of 3,144 hours, 805 hours, or 25.7 per cent, was lost for various causes.

Causes of lost time on an isolated gold dredge in Alaska.

Cause.	Time lost. Hrs.	Time lost. Min.
Bucket line[b]	19	39
Lower tumbler		45
Upper tumbler	1	10
Main gearing	103	46
Digging ladder	24	25
Hoppers	33	40
Save-all	20	15
Screen	61	19
Stacker	75	54
Flumes, etc	27	31
Pumps, screen	70	37
Winches	60	53
Lines	64	5
Spuds	16	15
Digging engine	4	20
Pump engine	18	55
Oiling	6	31
Stepping	141	5
Clean-ups	35	10
Rocks	13	14
Fuel	2	0
Light plant		13
Frozen ground	99	45
Miscellaneous	144	24
Total	805	1
Thawing (included in "Miscellaneous")	79	5
Bringing water to pond (included in "Miscellaneous")	29	0

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[a] From report of Gerald Hutton, dredgemaster for C. J. Berry Dredging Co., 1915.

[b] Most of the lost time was caused by the bucket line coming off the lower tumbler during heavy side feeding.

The table following shows causes of delay and percentage of time lost in operation of dredges in Montana:

Causes of delay on dredges Nos. 1, 2, 3, and 4, of the Conrey Placer Mining Co. and Poor Farm Placer Mining Co., 1914.

Cause of delay.	Time lost on dredge No.—								Percentage of time lost on dredge No.—			
	1.		2.		3.		4.		1.	2.	3.	4.
	Hrs.	*M.*	*Hrs.*	*M.*	*Hrs.*	*M.*	*Hrs.*	*M.*				
Clean-up	131	50	119	15	97	30	153	20	1.50	1.36	1.11	1.75
Moving ahead	367	5	448	55	250		299	55	4.19	5.13	2.85	3.42
Lines and spuds	260	10	176		278	20	212	15	2.97	2.01	3.18	2.42
Winches	98	50	69	55	180	40	17	5	1.13	.80	2.06	.19
Flume	....		....		147	35	....		....	....	1.68	....
Ladder	11	45	71	05	34	25	453	15	.14	.81	.39	5.18
Tumblers	86	35	184	55	166	55	313	55	.99	2.11	1.91	3.58
Hopper	4	10	33	40	43	30	73	50	.05	.38	.50	.84
Grizzly	212	25	161	25	·279	55	115	20	2.43	1.84	3.20	1.32
Bucket chain and drive	95		277	55	92	45	275	35	1.09	3.17	1.07	3.15
Pumps and piping	61	35	47	15	55	5	27	25	.70	.54	.63	.31
Electrical appliances	43	55	72	35	284	20	337	10	.50	.83	3.25	3.85
Heating system		10		15		40		15	....	....	....	....
Débris	212	55	108	35	219	30		55	2.43	1.24	2.51	.01
Power off	23	05	39	45	21	40	19	10	.27	.46	.25	.22
Stacker	186	05	90	55	....		63	55	2.13	1.04	....	.73
Tables and sluices	1	00	7	55		20	69	35	.01	.09	....	.79
Well	13	30	3	15	8	10	50	55	.15	.04	.09	.58
Gang plank	1	30	....		....			20	.01	....	....	....
Unloading coal	8	10	16	25	17		....		.09	.19	.19	....
July 4	32		40		40		40		.36	.46	.46	.46
Christmas	32		40		40		40		.36	.46	.46	.46
Scow	1	30	....		....		....		.01	....	....	....
Hull and housing	4	30	4	50	....		....		.05	.05	....	....
Low water	....		....		3		....		....	....	.03	....
Winter conditions	....		....		1,024		....		....	....	11.69	....
Raising No. 1 dredge	192	50	....		....		....		2.20	....	....	....
Headframe	....		68	30	....		....		....	.78	....	....
Miscellaneous	....			40	6	30	....		....	....	.07	....
Total delays	2,082	35	2,084		3,291	50	2,564	10	23.76	23.79	37.58	29.26
Total running time	6,677	25	6,676		5,468	10	6,195	50	76.24	76.21	62.42	70.74
Total time	8,760		8,760		8,760		8,760		100	100	100	100

WATER CONSUMPTION IN GOLD DREDGING.

The quantity of water required in dredging varies with the size and type of dredge and the character of the material to be handled. The fresh water that runs into the dredging pond does not by any means indicate the quantity supplied to the gold-saving tables by the pumps, as the water from the dredge pond is constantly reused. The quantity of fresh water obtained from outside sources depends on the nature of the material to be washed and on the water level on the ground being dredged. The quantity of water that is sent to the gold-saving tables is regulated by the dredge pumps. The following calculations assume that the pumps were running at full capacity, and do not take into consideration excess water lifted by the dredge buckets. On a large dredge working in easily washed gravel there are two 14-inch pumps which, when the dredge is at work, furnish water as follows: High-pressure pump, 5,500 gallons per minute; low-pressure pump, 5, 500 gallons per minute, or a total of 660,000 gallons an hour. On the basis of a running time of 22 hours and a daily capacity of 10,000 yards, these figures would indicate approximately 10 cubic yards of material washed per inch of water, and about 1,400 gallons to a cubic yard of material dug and washed in the screen.

The percentage of fine material in undersize that goes to the tables from the screen varies from 35 to 60 per cent of the total material handled, so at this stage of the operations there is more water used per cubic yard of material washed than is shown by figures based on the total yardage dug.

As a rule dredges working river bottoms or owning a gravity water supply take no account of the fresh water used, but on dredges for which water must be purchased the quantity used is recorded by a gage. Figures taken from the report of the Natomas Consolidated for 1913 indicate that there was supplied an average of 100 inches of water per dredge for the Folsom dredges.

Dredges working in Calaveras County pump water from the river, the only cost being pumping charges. The quantity of fresh water supplied for the 5-foot boats is estimated to be 50 to 70 inches. At Bohanan Bar near Salmon, Idaho, the manager calculated that when the dredge was working upstream on a 4 per cent grade, 350 inches of water was required from outside sources to supply fresh water and to maintain flotation of the dredge.

On dredges like the Natoma Nos. 9 and 11, which work much sticky clay, in addition to the pumps supplying water to the screens and the tables, the quantity of wash water is increased. There are two nozzles discharging water at high pressure (910 pounds) into the buckets as they dump into the hopper. These pumps would require an increased water supply of about 50 per cent over that for the other dredges. The yardage of these dredges averages about 350 yards per hour; hence the water furnished the screen would be about 2,800 gallons per cubic yard of material dug.

Plate LIX, *B* (p. 124), shows a revolving screen washed with water from nozzles attached to a pipe passing through the screen, and Plate LX, *B* (p. 124), shows a revolving screen with nozzles at each end for supplying water.

On the 5-foot dredges there are generally provided two 8-inch pumps, which supply about 3,200 gallons of water per minute. Based on a working time of 22 hours a day, this would be equivalent to 284 inches of water. Assuming 3,000 cubic yards dredged a day, the number of gallons of water used per cubic yard would be 1,400. The water consumption stated in cubic yards dredged per inch of water used is approximately 10.6.

On the small Alaska dredges a 10-inch pump works under a head of 45 to 50 feet, furnishing about 3,000 gallons a minute. Based on a 22-hour day and an average of 2,000 cubic yards dredged daily, this figures 266 inches of water, with 7.5 cubic yards to the inch and nearly 2,000 gallons per cubic yard. On the small flume type of dredge, with 2½-cubic-foot buckets, which has become so popular in Alaska, one 14-inch pump under the low head of about 22 feet is used. This delivers about 5,500 gallons per minute and on the basis mentioned above gives 488 inches of water or 4 cubic yards to the inch, and 3,600 gallons per cubic yard. This type of dredge requires more water than the screen dredge on account of the method of washing the material that dumps from the buckets into a straight flume instead of into the hopper supplying the screen.

On the 9-foot flume dredge of the Conrey company there are two 10-inch pumps and one 12-inch and one 14-inch pump, which supply the screen and the sluices with about 12,500 gallons per minute. As the average capacity of the dredge has been 161 cubic yards per hour, 4,800 gallons per cubic yard has seemingly been required, though the actual amount may have been less, as the pumps may not at times have been worked to full capacity.

On the Conrey No. 4 dredge the pumps furnish more than 12,000 gallons a minute, and the capacity of the dredge has averaged 420 cubic yards an hour, or 7 cubic yards a minute, since starting, indicating that 1,700 gallons of water has been pumped per cubic yard dug.

On the Boston & Idaho dredge there are three 14-inch pumps, which, when handling the maximum capacity of gravel, furnish nearly 20,000 gallons of water a minute. As more than 700 cubic yards an hour has at times been dug by this dredge, this dredge will also use approximately 1,700 gallons of water per cubic yard.

ACCIDENT PREVENTION ON GOLD DREDGES.

"Safety-first" rules for the prevention of accidents should be in force on gold dredges as in any well-conducted machine shop or other industrial plant. They will increase the efficiency of the men about the dredge as well as to save them from accidents resulting from their own carelessness.

All belt railings, gear guards and gear railings on dredges and in machine shops should be so arranged as to make it difficult for the men to step between the top and bottom of a running belt or through the slow-running bull wheels. The form of these guards will vary according to the arrangement of the drives and according to the construction of the machinery to be guarded, suiting the climatic conditions under which the dredge is to work.

The general safety orders of the industrial accident commission of California prescribe the following:

> All gears, where exposed to contact, must be entirely inclosed, or equipped with side flanges extending beyond the root of the teeth.
>
> All spoke gears and open-web gears which are over 18 inches in diameter, where exposed to contact, must be entirely inclosed. On large gears, such as those in heavy shears or punches, the guard must be such as to cover them to a height of 7 feet above the floor.
>
> Where it is clearly impracticable to cover gears, as described above, a boxed frame of metal or wood must be installed, completely shutting off the machinery gears.
>
> All gear guards must be kept in place while the machinery is in operation.
>
> All belts, ropes, or chains driving machinery or shafting, and all secondary belts, ropes, or chains where exposed to contact, must be guarded. In all cases the point where the belt, rope, or chain runs onto the pulley, sheave, or sprocket, if within 7 feet of the floor or platform, must be guarded.
>
> Exception: Belts which are so small or so slow moving that they are not in any way a source of danger.
>
> All horizontal belts, ropes, or chains driving machinery or shafting, 7 feet or less above the floor or platform, where exposed to contact, must be guarded. All overhead belts 6 inches or more in width and over 7 feet from floor or platform, must be guarded underneath and on sides unless so guarded that persons can not pass under them. All chain or rope drives over 7 feet from the floor or platform must be guarded in like manner to belts over 6 inches in width. In all cases the guard should cover the outer faces of the two pulleys or sheaves and extend upward to such a point, and be attached in such a way that in case the belt, chain, or rope breaks the guard will withstand the whipping force.
>
> Vertical and inclined belts must be guarded substantially as follows: If the guard must be less than 15 inches from the belt, with a complete inclosure of wood or metal to a height of 6 feet above the floor. If the guard can be placed with at least 15 inches clearance from the belt, with a two-rail railing at least 3½ feet high.

Oiling facilities and convenience in repairing should receive special consideration with a view to minimizing the chances of accidents to the machinery itself, as well as to the crew. Except for motor and pump bearings, practically all the bearings on a gold dredge are lubricated with grease, and therefore must be filled either every shift or every day. As there are a rather large number of bearings, the grease cups and grease-gun openings should be placed conveniently so that the oiler may grease as many bearings as possible while the dredge is running, without exposing himself to any danger. If the grease cups are placed some distance from the bearing proper, larger pipes should be used so as to decrease the friction.

The lower tumbler may be greased by means of mechanically operating grease pumps driven from the upper tumbler shaft, and equipped with a pressure gage in the pilot house to indicate that they are working properly. The passage for the grease may be drilled through the whole length of the lower ladder end casting to which it may be connected by a 1¼-inch pipe running under and outside the ladder flange. On the upper end, this pipe is connected by means of a metal house, to the pump. Each side of the lower tumbler is lubricated independently.

The ladder roller bearings have to be greased separately with a grease pump. A considerable saving in weight can be effected by making the barrel of the grease pump of aluminum. The oiler has to walk up and down the inclined ladder on the ladder flange, and he will readily appreciate the difference in weight.

Platforms should be provided on both front and rear gantries for oiling the ladder-hoist sheaves, the spud sheaves, and the stacker sheaves. If a double ladder hanging is used, there should also be a platform for the lower sheave blocks. The platforms should be made strong enough so that they may be utilized in repairing the parts mentioned. All stairways should be provided with smooth railings. Platforms should have both railings and toe boards wherever practicable. All sheave blocks should be equipped with separate oil holes and grease cups for each sheave.

The hubs of operating levers, as well as the shaft bearings for lever connections, should be provided with oil holes or grease cups. Proper use of them will prevent the sticking of levers, caused by the cutting or rusting of shafts and bearings, and will also make the handling of the levers less arduous.

Efficient oiling arrangements not only add to the safety of the oiler, but increase the life of bearings and shafts, and result in better running time.

Drip pans should be arranged underneath the gear trains and bearings to catch excess grease, to help to keep the dredge clean, and to prevent accumulation of grease on the deck and floors.

For bucket-line repairs, the ladder should be provided with cleats or equivalent members, to which cables may be hitched. They should be so arranged that a hitch may easily be made around them.

Keys for connecting the hubs of revolving or swinging members to shafts should, if possible, be of the head type; if this is impracticable, arrangement should be made for backing out the keys. For the flange couplings of direct-connected centrifugal pumps, the Woodruff keys for hubs and shafts seem to be preferable to the ordinary taper keys for insuring easy disconnection of the coupling when ring members of the lamp shaft have to be renewed and are too small to slip over the couplings.

The seats of hubs having a press or shrink fit on a shaft should be made about one-sixteenth inch larger in diameter than the remainder of the shaft, for convenience in backing the gears or pulleys connected to the shaft.

Belt speeds on dredges, especially on the first countershaft on the bucket-drive motor, should be kept below 4,000 feet per minute, as at a higher speed the problem of shaft and pulley balance enters, and bursting of clutches, owing to centrifugal and other forces, is possible.

Electric wiring should, in general, be installed according to the fire underwriters' code. Wires should be run in metal conduits. For a 2,200-volt circuit, lead-covered cables, with all joints carefully soldered, are recommended. Outlets close to the motor terminals should be arranged to guard against mechanical injury. A slot or hole in the conduit at the lowest point of the circuit will prevent accumulation of water from any source. The wires to the controllers of the variable-speed motors should preferably be run in conduits to prevent fouling with operating levers below the floor of the pilot house.

Rubber matting on floors below the switchboard and below motor starters has proved of advantage. Discarded conveyer belt may be used for the purpose. Care should be taken to see that gravel and grit are not allowed to be tramped into the rubber mat, as a small puncture will allow a ground to be made. Mats held together with wooden pins are recommended as best, but they must be kept dry at all times when in use. It is suggested that a wooden mat be placed on top of a rubber matting. To prevent stumbling, the two ends and the side away from the switchboard should be beveled. Rubber gloves are valuable for the crew for handling shore cable and extension pole lines.

As climatic conditions and operating requirements influence the most desirable construction of electric motors no detailed remarks need be made about them except that they should be fully guarded by railing or wire netting.

If transformers are used on board the dredge, they should be housed as independently as possible from the rest of the dredge. On a wooden hull dredge, the transformer house should be lined with asbestos and galvanized-iron sheathing. The switchboards should be equipped with all the safety and protective devices used on any first-class electrical installation.

Flanges on water piping should be made of material to withstand vibration strains set up in digging.

For pump-discharge valves and for all other gate valves used on dredges, outside spindle valves or valves equipped with indicators are desirable, as they give instant information as to the amount of valve opening. Foot valves can be made so that the valve seats are accessible and so that the valve flaps may be replaced without hoisting the suction out of the water.

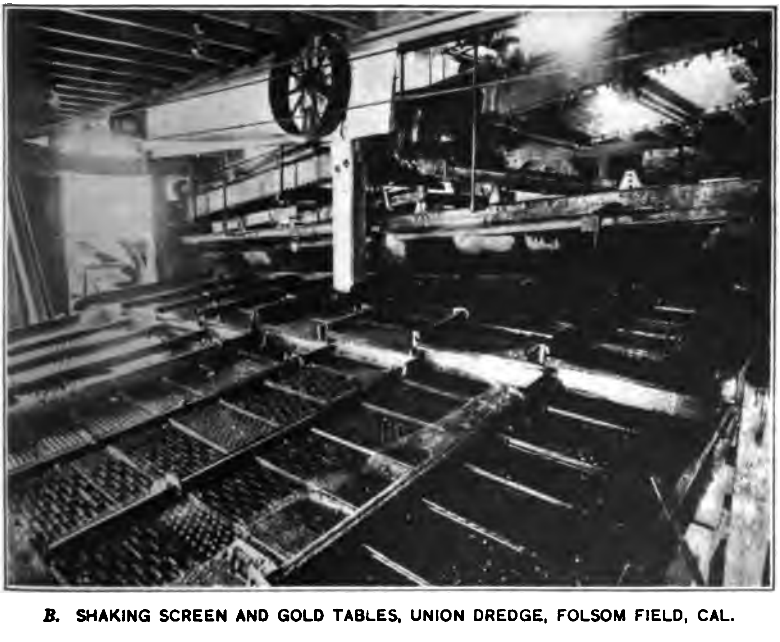
Although the underwriters' rules specify fire pumps on board of dredges, these pumps are generally useless as they are driven by the same power that operates the dredges and the usual cause of fire is through some disarrangement of the electric current, which of course puts all the motors out of commission. It would be better policy to have an auxiliary fire apparatus consisting of a quick-starting gasoline engine which could be started at a moment's notice, attached to a fire pump with 150 feet of hose, and placed on a barge alongside the dredge. This would be a precautionary measure that would appeal to insurance companies and result in a reduction in rates.

In addition to having a number of effective hand extinguishers at readily accessible places, a large chemical extinguisher on wheels with a capacity of 20 gallons or more, with hose connected, should be supplied. The first cost of such an extinguisher is small, and several disastrous fires could have been averted had the dredges been supplied with them.

The cause of the Viloro dredge (Pl. LXI, *A*) being lost by fire was thought to have been a breakdown or short circuit in the main bank of transformers, as flames suddenly burst through the transformer housing. Three men were on shift and two of them must have seen the flames at about the same time and rushed for the fire apparatus. One of the men who was on the lower deck made an effort to start the high-pressure pump that was connected to the fire mains, only to find that the power was off owing to the panel at the substation being thrown out. Thus a first-class pumping plant, attached to a separate set of transformers, proved inadequate in time of need.

The lack of cleanliness and the careless handling of oil and other inflammable material about dredges can not be too severely condemned. It is better to be safe than sorry, and a little attention

A. WRECK OF VILORO DREDGE.



B. SHAKING SCREEN AND GOLD TABLES, UNION DREDGE, FOLSOM FIELD, CAL.

to some of these rules is certainly justified. Oil may be readily removed from floors by sprinkling them with air-slaked lime and then brushing them with a stiff brush.

It is well to have sand within easy reach to use in overcoming electrical disturbances at transformers, and the judicious use of asbestos board or other fireproof material at exposed places is advisable. In addition, on wooden dredges, a few dollars can be advantageously spent in liquid stone or other fireproof paint for exposed woodwork. The new steel dredges are regarded as fireproof, and no fire insurance is carried on them.

The foregoing general suggestions are offered for consideration. The application of such measures is the business of the dredge builder. The instruction of the dredge crew as to their respective duties regarding operation, use, and care of tools and machinery, inspection of hoisting and swing lines, proper anchoring of the swing lines, and the like is part of the dredge master's task.

GOLD-SAVING APPLIANCES.

The gold-saving appliances of a modern dredge consist of a revolving screen to disintegrate and classify the dredged material, a distributor, and a series of tables and sluices fitted with riffles. Formerly the shaking screen had a number of advocates, but all dredges built in recent years have been equipped with the revolving screen which has proved its superiority as a disintegrator and classifier, especially when the material is at all difficult to wash.

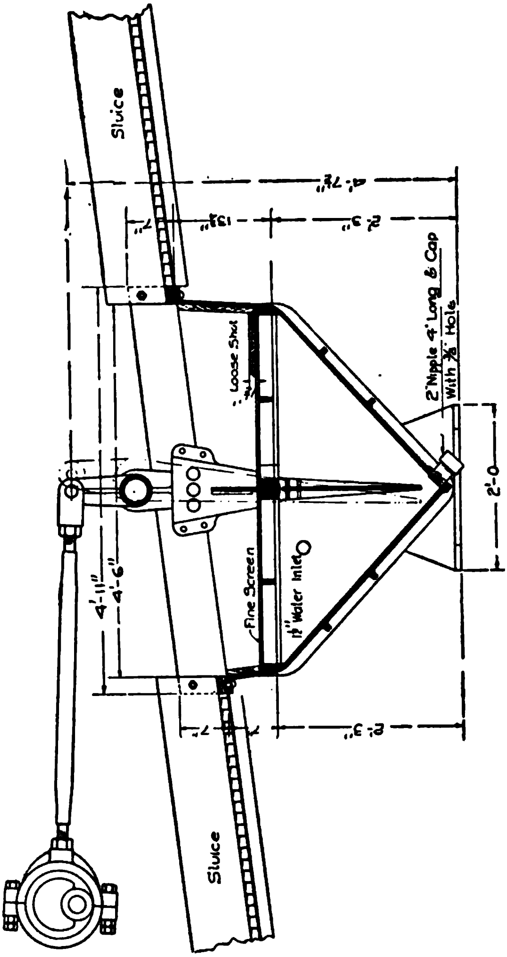


FIGURE 19.—Neill jig and eccentric drive.

With a few exceptions, the metallurgy of dredging is simple, and until recently few changes have been made in the gold-saving arrangements of a dredge during the last six or eight years. Although considerable attention has been given to increasing the yardage handled by the dredge and to increasing the operating time, it was thought that additional table area was sufficient for all metallurgical requirements. In the early days of dredging experiments were made to ascertain the loss of gold in dredge tailings, but it was difficult to obtain a representative sample. Although most of the tests showed a loss of some gold, the loss on the best managed dredges was generally estimated to be not more than 5 per cent of the total content for the ground handled, and many of the operators were content to let well enough alone. Present-day tests have demonstrated that there has been a considerable gold loss at times, and far more attention is being given to improving the percentage of gold recovery. The most important change that has been made to this end is the use of jigs to replace or supplement tables, and the grinding of jig concentrates in order to brighten the rusty gold so that it will amalgamate.

USE OF JIGS.

According to the author's recollection, the use of jigs on the gold dredge was first suggested by Robert H. Richards,[a] following experiments made by the United States Geological Survey at Portland, Oreg. It remained for J. W. Neill to first try out this idea on a working scale. Mr. Neill describes[b] his tests with the jigs on the Yosemite dredge in California. (See figs. 19, 20, and 21.) Subsequent experiments with one of these jigs were made on Dredge No. 7 of the Natomas Consolidated Co. It was found that a considerable quantity of gold failed to amalgamate and was lost in the tailing. A number of changes and improvements were made in the jig, and it has to-day proved a decided advantage in saving fine gold and gold that will not easily amalgamate.

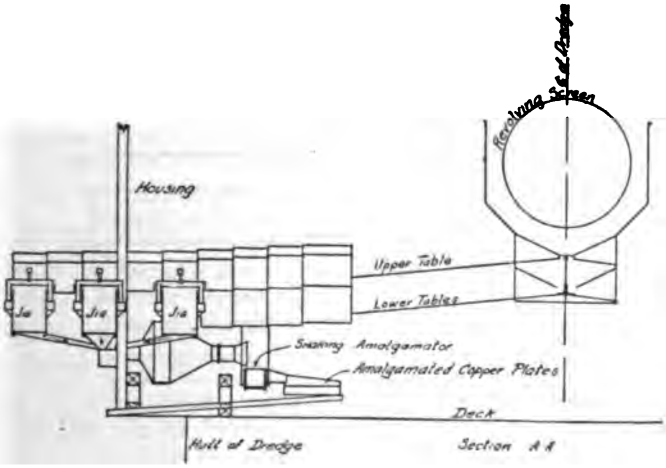


FIGURE 20.—Cross section showing arrangement of jigs, Hardinge mill, and amalgamator, Yosemite dredge.

GOLD-SAVING EQUIPMENT ON TWO NATOMA DREDGES.

The installation on the No. 7 dredge consists of 8 jigs and two 4½-foot Hardinge pebble mills. Four of the jigs and one of the mills were placed on each side of the dredge. On the No. 10 dredge 10 jigs are used—5 on each side. The jigs are placed in the first four sluices of the gold tables leading from the upper, or forward, end of the screen, and the mills are placed on the main deck, so that the concentrates from the jigs will run by gravity to the feed box of the mill.

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[a] Richards, R. H., Utilization of black sand: Eng. and Min. Jour., vol. 83, June 29, 1907, pp. 1251-1252.

[b] Neill, J. W., Application of jigs to gold dredging: Min. and Sci. Press, vol. 109, Nov. 28, 1914, pp. 839-842.

The results of several tests made at different times indicate that the two mills handle approximately 75 to 80 tons of concentrates a day, on the basis of a daily average running time of 20 hours. The sluices in which the jigs are placed recover approximately 75 or 80 per cent of the gold that is recovered on the dredge.

The value of the concentrates from the jigs varies from 75 cents to about $1.50 a ton, from about 6 to 12 per cent of the gold cleaned up on the tables. Tests made for comparative purposes indicate that not less than 75 per cent of the gold passing through the mill is recovered by the jigs and mills.

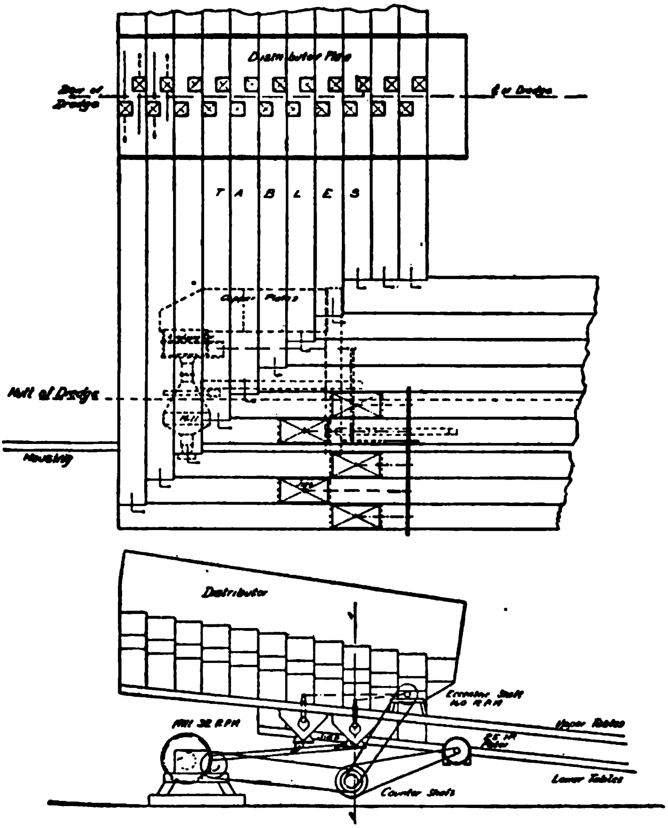

FIGURE 21.—Plan and elevation of tables on Yosemite dredge, showing position and drive of jigs and mill.

Several tests to determine the saving of gold after it had passed through the mill showed that the losses between the heads or concentrates from the jigs and the tails, after the material had passed through the mill and over the plates, were from traces to 5 cents per ton. The valuable material lost was in the form of minute particles of gold amalgam and floured quicksilver.

The following figures regarding the Natoma dredges were furnished by Griffin:[a]

Each bank of jigs and one mill are driven by a 25-horsepower motor. The total power consumed is about 36 horsepower, divided about 18 horsepower to the mills and 18 horsepower to jigs.

The labor cost averages about $110 a month; material, about $25; and power, about $120; total, $255.

We have jig and mill equipment on dredges Nos. 7, 8, 9 and 10 and a jig without mill on No. 4. My data are complete on only dredges Nos. 7 and 10, results from which follow:

Gold recovery from mills and jigs of Natoma Nos. 7 and 10 dredges.

Dredge No.	Period days.	Cubic yards dredged.	Gross clean-up.	Value per cubic yard.	Mill and jig clean-up.	Value per cubic yard.	Mill and jig clean-up, per cent of total.
				Cents.		*Cents.*	
7	176	842,014	$108,153	12.84	$12,879	1.52	11.90
10	256	1,973,661	215,444	10.92	15,736	.80	7.30

The recovery percentage from Nos. 8 and 9 is somewhat lower than from No. 10 and will average about 5½. The recovery on No. 4 is so low that I am considering removing the jigs unless there is a change for the better in the near future.

CHARACTER OF GOLD-SAVING TABLES.

Gold-saving tables are made either in single or double banks. Single-bank tables are used on dredges up to 7½ feet, but in the larger boats the riffle area obtainable on a single set of tables has generally been found insufficient, and to meet the requirements of the increased yardage a second set of tables was arranged, thus doubling the riffle area. The double-deck table was developed in 1908, and the use of four tail sluices of different lengths made it possible to distribute the gravel over a greater area and to discard the sand pump which was always a source of trouble.

The single-bank tables are usually made of wood and the double-bank tables of steel. The tables are supplied with some form of the so-called Hungarian riffles, either of wood shod with an iron bar on top, or of angle iron, or of cast bars of manganese or other steel. The angle-iron shape is still considered by many as the most effective, for the water passing over the riffles forms an eddy under the projecting leg which facilitates catching the gold. Amalgam or quicksilver riffles are set in each sluice and the quantity of quicksilver used has greatly increased, on some dredges being about ten times the amount formerly used.

CONSUMPTION OF QUICKSILVER.

The consumption of quicksilver on several dredges under Oroville conditions is shown by the accompanying table furnished by Mr. C. G. Leeson, of Oroville, and is of interest in being, so far as the author knows, the first table giving such results that has been published.

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[a] Griffin, F. W., personal communication.

Quicksilver losses on Oroville dredges.

Period.	Name of dredge.														
	L. B. No. 2.			L. B. No. 3.			Empire.			Hunter.			Victor.		
	Number of clean ups.	Amount of loss per clean up.	Per cent lost.	Number of clean ups.	Amount of loss per clean up.	Per cent lost.	Number of clean ups.	Amount of loss per clean up.	Per cent lost.	Number of clean ups.	Amount of loss per clean up.	Per cent lost.	Number of clean ups.	Amount of loss per clean up.	Per cent lost.
		Ounces.			*Ounces.*			*Ounces.*			*Ounces.*			*Ounces.*	
First six months of 1908	No record.														
Second six months of 1908	9	65.6	1.9	11	26.7	1.07	12	47.1	1.3	12	221	3.6	9	67	2.4
First six months of 1909	12	114	3.5	12	69.7	2.5	12	47.2	1.4	12	52	1.1	12	83	2.8
Second six months of 1909	12	92	3.1	12	144	5.4	7	108	3.6	12	61	1.5	11	49	3.03
First six months of 1910	No record.														
Second six months of 1910	13	98.5	2.6	9	192	2.3	9	67	1.2	7	74	1.05	8	106	2.04
First six months of 1911	No record.														
Second six months of 1911	12	93	3.35				11	85	2.66	12	211	6.64	12	158	5.48
First six months of 1912	12	102.8	3.75				12	85.9	2.6	12	86	3.07	11	117	4.5
Second six months of 1912	13	97.8	4.1				13	83.5	2.97	11	190	8.0	12	141.5	4.7
First six months of 1913	12	88.25	4.14				11	99	3.06	10	126.5	3.15	12	69.58	2.26
Second six months of 1913	12	116	5.06				13	65	2.04	12	54	1.56	12	81	2.60
First six months of 1914	11	96	4.12				11	115	4.07	12	82	2.35	12	91	2.91
Second six months of 1914										13	120	3.2	12	55	1.55
First six months of 1915										12	144	3.5	11	49.5	1.5
Second six months of 1915															

NOTE.—The average per cent lost on the L. B. No. 2 per 6-month period was 3.5 per cent; the average per cent lost on the L. B. No. 3 per 6-month period was 2.72 per cent; the average per cent lost on the Empire per 6-month period was 2.46 per cent.

The quantity of quicksilver placed on the dredge tables varies with the size of the dredge and with the ideas of the operator—from 150 pounds or less on the smaller dredges up to 3,000 pounds on the larger dredges. The Yuba Consolidated Co. has gradually increased the quantity of quicksilver placed on the tables and found a corresponding improvement in the percentage of recovery. A few years ago 150 pounds was considered sufficient for the 7-foot dredges, but now these dredges with a table area of 2,800 square feet use about 1,000 pounds for the sluices and traps.

On some small dredges the loss of quicksilver has amounted to as much as 7 pounds a week when only 2 flasks at a time were being used on the tables. This would be about 5 per cent of the total amount used. On the Jenny Lind dredge the loss of quicksilver was greatly reduced when the clay sluice previously described (p. 120) was put in, showing how the clay had robbed the sluices.

FACTORS AFFECTING EFFICIENCY OF TABLES.

The efficiency of the gold-saving tables (Pl. LXI, *B*, p. 136) is affected by the grade. It is generally found that a grade of 1½ inches to the foot is the most efficient slope for the tables and sluices, though the nugget sluice of the Tonapah No. 3 dredge described elsewhere is run on a steeper grade, and at Ruby, Mont., a grade of 1 inch to the foot is used.

Another factor affecting the efficiency of gold-saving tables is the quantity of material sent to the tables. If the grade of the sluice is too flat, or the volume of water insufficient, there will be an accumulation of sand, and amalgamation will be hindered. If the tables are crowded beyond their capacity there will be a loss of gold-saving efficiency.

A third factor affecting gold-saving tables is the amount of water. Too much water may carry off some of the fine gold, but more gold is lost from insufficient washing than as gold carried in suspension. The proper amount of water to use in dredging varies with the material dug, but the principle to follow is to regulate the water so as to bring as much of the gold as possible into contact with the quicksilver.

At Ruby, Mont., in order to test the efficiency of the tables, additional gold-saving devices consisting of undercurrents were placed on Dredges Nos. 3 and 4. The undercurrents on the No. 3 dredge, a tail-sluice dredge, are placed on the tail-sluice pontoon, and consist of 10 tables connected by launders to the lower end of the tail sluice, where the fines pass through a grizzly. On the Conrey No. 4 dredge, the undercurrents are placed near the stern immediately under the main gold-saving tables. There are 12 tail sluices to the tables on this dredge, and at the lower end of each tail sluice there is a plate perforated with holes three-eighths of an inch in diameter, through which the fines pass onto the undercurrent tables. The gold saving tables and sluices of this dredge are shown in Plate LIII, *B* (p. 98).

During 36½ months' operating the return from the undercurrent arrangement on the No. 4 dredge amounted to about 0.5 per cent of the total recovery, indicating the efficiency of the tables and sluices of the stacker dredge when very fine or rusty gold is not present. On the No. 3 sluice dredge the undercurrent tables and undercurrents have shown recoveries of something like 8,000 ounces of amalgam since December, 1908, indicating that there is greater danger of loss of fine gold on dredges of the sluice type than on dredges of the stacker type.

The recoveries made by the save-alls on the different dredges are indicative of possible losses from spilling from the buckets caused by stickiness of material and overfilling and noncleaning of buckets. The different gold-saving devices in the well save-alls are somewhat similar; but on some dredges it is easier to guard against a spill than on others. On the Conrey No. 4 dredge it is possible that there is less spill than on any of the other Conrey dredges and consequently less percentage of gold caught. The recovery from the well save-all varies from 2.76 per cent of the total clean-up on the No. 4 dredge to 5.01 per cent on the No. 3. The undercurrent system was used on the No. 4 dredge mainly with the idea of determining the loss of gold by the tables, and it has proven a comprehensive check on this work besides resulting in a saving of over 1,000 ounces of amalgam to July 1, 1915.

The nature of the gold recovered from the Conrey ground is distinctly coarse, but not many nuggets are found. Even in the No. 3 sluices the largest nugget so far recovered was worth $17 and there are few nuggets found that are worth more than $5. The fineness of the gold varies from 0.836 to 0.873, the lower-grade and coarser gold being found in the upper part of the gulch.

At Breckenridge, Colo., the screen of the No. 3 dredge of the Tonapah Placer Co. is filled with plates next to the lower section, having 2½-inch holes. The fines from the tail of the screen go to the "nugget" sluices, built somewhat steeper than the main tables. This arrangement results in the recovery of considerable nugget gold that would otherwise be lost.

Undercurrents were put on the No. 3 sluice dredge of the Tonopah Placer Co. at Breckinridge, and resulted in large savings of gold formerly lost. A sketch of the undercurrent system, furnished by Mr. E. J. Hopkins, is shown in figure 22.

Undercurrents have not been extensively tried in California except on dredges of the sluice type; on one California table dredge a blanket was tried on an undercurrent in the main sluice, but the recovery made was not of much importance. The placing of undercurrents on the main sluices of some dredges would unquestionably effect a saving even greater than that made by the undercurrent on the No. 4 dredge at Ruby, and the practice should be more widely adopted.

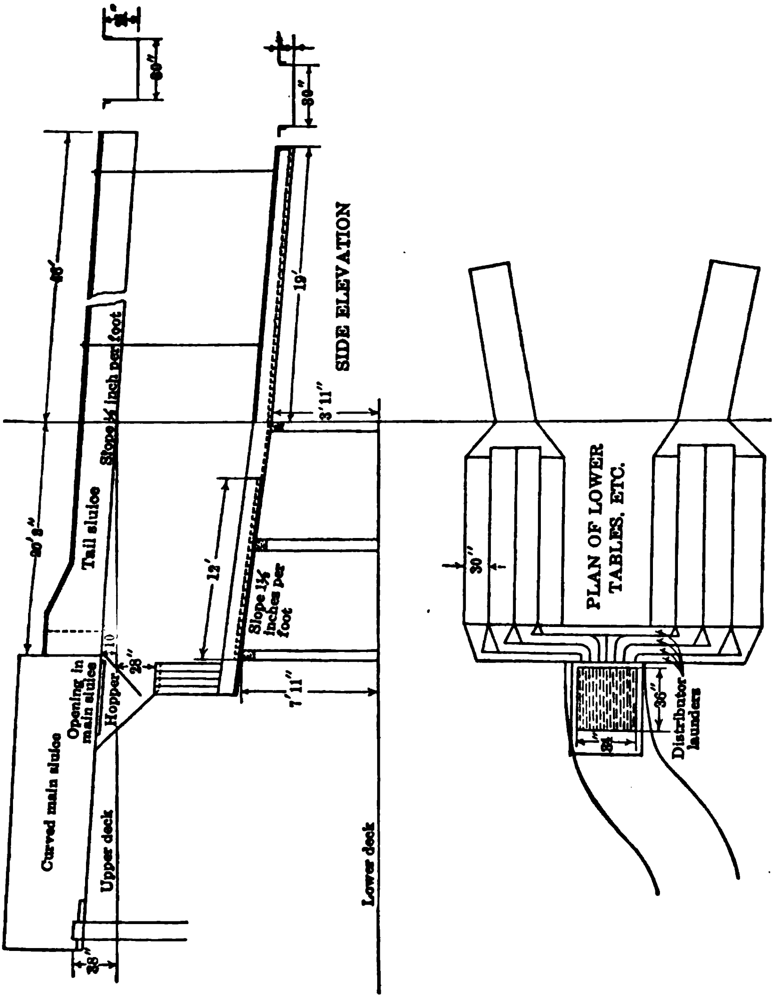

FIGURE 22.—Plan and elevation of undercurrent system used on No. 3 dredge at Breckenridge, Colo.

TABLE AREA.

The table area varies on different makes of dredges and with the ideas of different operators, although it is governed to some degree by the character of the gold in the ground handled. A good rule for the table area seems to be to make it as large as possible, as a safeguard against losses.

California practice on the large dredges handling 350 to 500 yards an hour is to have a table area of 6,000 to 8,000 square feet. Dredges handling 140 to 300 yards an hour have 2,000 to 2,800 square feet of table surface, and smaller dredges 1,000 to 1,500 square feet. These figures may be compared with those resulting from the practice at Ruby where, as saving of the gold is easy, less than half of this space is used. The Ruby figures are as follows: The No. 4 dredge, handling an average of 420 yards an hour since it started operation, has 2,955 square feet. Dredges Nos. 1 and 2, handling averages of 174 and 132 yards per hour, have 1,200 square feet of table area, and the sluice dredge, which has handled an average of 161 yards an hour since commencing operation, is equipped with approximately 1,000 square feet. On the 18-foot dredge of the Boston & Idaho Co., which handled more than 4,600,000 cubic yards during 1914 and more than 600 yards an hour at times, the table area is approximately 4,000 square feet and a part of this table area was cut out as being unnecessary, most of the gold being coarse and easily saved, though there may have been some loss of fine gold. The dredge of the future will no doubt be equipped with jigs to take the place of a considerable part of the table area, and experiments with jigs will be watched with interest.

Water is supplied to the revolving screen by a high-pressure pump through nozzles placed at either end or both ends of the screen (Pl. LX, *B*, p. 124), or by a pipe passing through the center of the screen which is equipped with adjustable nozzles (Pl. LIX, *B*, p. 124). Additional water is fed the tables by the low-pressure pumps. The grade of the screen is about the same as that of the table. It is vital that plenty of water under sufficient pressure be fed to the screen in order to insure satisfactory disintegration and washing of the material, especially if clay is present. A high loss of gold is due to inefficient washing in the screen.

LOSS OF GOLD IN SUSPENSION.

Although the greatest percentage of the gold recovered in dredging is caught on the upper parts of the first three or four tables, doubtless there is on some dredges a considerable loss of gold caused either from insufficient washing of the material on the screen or too small a quantity of water supplied the tables, and on most dredges some quicksilver will be found at the lower end of the tail sluices. In addition a considerable amount of gold is carried in suspension in dredge water. Tests to determine whether there was any loss have been made on several fields with the same result. Samples were taken of the water while the dredge was in operation and of the mud that settled in the dredge pond. The latter test showed that the gold was lighter than the mud and settled more slowly, and the farther the sample was taken from the stern of the dredge the richer it proved to be after the water had been evaporated.

No one has yet worked out a scheme entirely to overcome loss; some very fine gold is caught in the Neill jig and the Hardinge mill arrangement put on the Natoma No. 7 dredge; the concentrate from the jig proves on examination to contain not only rusty flake gold, but also very fine and floury bright gold, but it is doubtful whether this fine gold was previously being carried in suspension. Much of the gold that is carried in suspension would amalgamate if brought into sufficient contact with the quicksilver, but on account of its finely divided condition it is carried over the tables in suspension with the water and is one of the gold losses in the present method of handling the dredge material. This loss seems largely unavoidable. The concentration necessary for the saving of the main gold content causes a loss of some of this light material and in addition to the loss there is some gold that is put in suspension by the breaking up of the ground by the bucket line and does not go over the tables at all.

CLEANING OF TABLES.

The gold tables are cleaned at regular intervals, varying somewhat with the character and the gold content of the material. On the large dredges the main tables are cleaned every week and the side sluices every other week. For doing this work the large dredging companies employ clean-up crews, the dredge crew assisting as far as necessary, or attending to overhauling and repairing the dredge.

The time required to clean the table and sluices varies from 4 to 8 hours on the different dredges. The method of cleaning varies, but the principle is the same. Commencing at the upper end, the riffle frames are taken up, washed, and laid to one side. Water from a small hose under pressure is used to assist in gathering the amalgam and the concentrates, and these are scraped into buckets and washed in a "Long Tom" or small sluice. The amalgam and quicksilver is collected and strained and the amalgam retorted and melted.

In some districts shot, lead, nails, coins, etc., are picked up by the dredge and lodge on the tables. This material, all of which collects some amalgam, forms a second product which is treated whenever a sufficient amount accumulates. The following method of cleaning was developed at Ruby, Mont.:

Above the last stop in the lower end of the table is fitted a converging iron trough, emptying through a short length of pipe into the clean-up box. The riffles are taken off the table, water is turned on, and all the gravel, gold, mercury, etc., is washed into the clean-up box, which is 2 feet 11 inches long, 18 inches wide, and 15 inches high. The box is divided into four compartments by an iron plate extending from the top of the box to within about 3 inches of the bottom. In each of the two central compartments is an iron plate 3½ inches high, extending from the bottom. The material entering the box must pass alternately under and over the partitions to get to the overflow of the box. The box is placed on a slope of 1½ inches in 12 inches, and mercury is placed in front of each of the bottom plates and at the lower side of box, thus forming three mercury traps through which the gold and sand must pass. The material overflows from the box into a small tail sluice containing matting and expanded metal and thence down the tail sluice of the dredge. This method has been demonstrated to be most efficient. Any gold not fully amalgamated can be worked into amalgam by slightly stirring the box as the material passes through. Two men clean up the Conrey No. 4 dredge, one of the largest in the world, in four to five hours, saving wages and time as well as making a more perfect clean-up than by the former method employed.

GOLD SAVING IN YUBA FIELD.

The following notes on gold saving in the Yuba gold field represent the practice followed by one of the world's leading companies. Over $20,000,000 has been taken out to date. Owing to excellent natural conditions gold saving in this field is rather simple. The ground is clay, gravel, and sand, with practically no loam and little clay except on bedrock. There is plenty of clean water, so that screening and washing are easily handled.

The dirt goes from the buckets to the hopper, the buckets being washed with a heavy stream of water to insure clean dumping. What dirt does hang to the buckets falls on the grizzly above the bucket idler. The fines go through and are washed over riffles in the save-all, the construction of which is similar to that of the main gold tables.

Water is supplied to the material in the screen through nozzles in the upper end and in the lower end of the screen. The washing is effective and the gravel that goes to the conveyer belt is clean. The undersize from the screen pass through the distributor to the gold tables, where more water is fed from other pumps. On account of the high proportion of sand in this ground, more water is necessary than under usual conditions, a most important consideration.

On the large dredges the gold tables are in two banks, 10 sluices in each, on both sides of the screen. The sluices are 30 inches wide and are set with a fall of 1½ feet to 12 feet. The total table area is 7,800 square feet. This may be compared with the table area of the No. 4 dredge at Ruby, 2,900 square feet, as showing the different requirements in handling fine and coarse gold.

The ordinary Hungarian riffle used here is made of wood with an iron bar on top; and some experiments are being made with manganese bars for this purpose.

A number of trap or quicksilver riffles are set in each sluice. These are made much like the ordinary riffle, except that the bottom is solid and the spaces between are divided into compartments which are filled with quicksilver. These riffles have been found to be efficient The quicksilver is apt to sink down in the soft sand in the other riffles and not come in contact with the dirt. About 3,000 pounds of quicksilver is used on the tables every week on the No. 14 dredge. The 7-foot boats with a table area of 2,800 square feet use about 1,000 pounds a week. This increased use of quicksilver has been one of the important changes. A few years ago 150 pounds was considered ample for a 7-foot boat.

The clean-up is made every week. The riffles are taken up and washed. The dirt is concentrated at the stops to about 20 per cent. of its original volume, much light sand being left with the amalgam, quicksilver, and black sand. This light sand often carries some rusty gold. The concentrate is taken up and washed in a "long Tom," and the amalgam, quicksilver, and most of the platinum are taken out. The sand is screened at the lower end through an 8-mesh screen and the fines are sacked and taken in to the mill.

The mill is an amalgam barrel with a long flat sluice beneath. The sand is added in the following proportion: One thousand pounds to 20 pounds of mercury, ¼ pound of cyanide, and 80 gallons of water. This mixture is ground two hours and washed out in the sluice. The cyanide is an important factor. It keeps the quicksilver in good shape and aids in amalgamation. There is a gain of about 10 pounds of quicksilver to 100 tons of sand if cyanide is used, and a loss of about 20 pounds if it is not used. It almost entirely prevents flouring of the quicksilver and brings together the flour gold that is in the sand.

Tests made of the sand from the mill after washing have showed practically no gold or platinum.

The retorting and melting is by the ordinary process.

On most dredges the main gold tables are inclosed with heavy wire netting to prevent pilfering of the amalgam.

WORKING COSTS.

The working costs of gold dredges are always of interest, although it should be recognized that comparisons that are at all representative are difficult. In this section are given a number of working costs for dredges operating in different fields, with notes of some points that should be taken into consideration when making comparisons between different dredges. The experienced engineer, of course, will not fall into the error often made by the inexperienced or casual investigator of trying to compare and reconcile the working results of dredges operating under widely different physical and economic conditions.

REQUIREMENTS IN COST ACCOUNTING.

In making comparisons of operating costs of dredges working in different fields, it is well to keep in mind several important facts. The dredges compared should be operating under somewhat similar conditions, and if transportation and other economic facilities are not closely similar, allowances should be made for any vital difference. Operating costs should be similarly tabulated, for seeming low operating costs often result from a particular method of bookkeeping.

Depreciation charges are generally arbitrary assessments based on previous experience, and occasionally these are left out of stated operating costs. In the matter of repairs, bucket-line renewals are not always charged to working costs though monthly repairs are included. This item alone might amount to ½ cent to 1½ cents a cubic yard.

Figures should be obtained over as long a period as possible, for dredges operating under favorable circumstances and for short periods have shown bare working costs of less than 1½ cents a cubic yard, or approximately 1 cent a ton of material handled. Dredges operating even in the same locality may over different periods encounter entirely different physical conditions in the ground to be dug, as shown by examples given (p. 107). The age of the dredge may also have a considerable bearing on costs, the older dredges generally having higher repair costs.

Similar methods of calculating yardage should be employed, and in this connection the following comparison of yardage dug by the different dredges at Ruby during 1914, as determined by bank measurement and by the theoretical capacity by arranging an accurate automatic tally of the buckets dumping during the same time, is of interest; it illustrates some of the reasons for the seemingly low operating cost per cubic yard occasionally attributed to some of the dredges built and operated outside this country.

Comparison of yardage dug by different dredges at Ruby in 1914.

Dredge No.	Size of buckets.	Average depth dug.	Actual yardage.	Theoretical capacity.	Buckets per minute.	Kilowatts per yard.	Yards per hour.
	Cubic feet.	*Feet.*	*Cubic yards.*	*Cubic yards.*			
1	7	26.6	1,100,906	6,293,400	15.71	1.78	165
2	7	26.2	1,140,195	5,925,300	14.79	1.42	171
3	9	34.6	1,068,164	4,542,590	13.85	1.79	171
4	16	49.7	3,093,513	7,525,224	20.24	1.23	499

In foreign fields the yardage handled is often based on a theoretical capacity of the dredge for the total running time, after some allowance has been made for buckets partly filled. How little this percentage would represent actual conditions is shown by the foregoing table, for with the exception of the No. 4 dredge, which handled 41 per cent of the theoretical capacity, the other dredges averaged only 20 per cent of the buckets actually dumping. In estimating yardage from bucket capacity, it must also be remembered that calculations are generally made for loose gravel instead of from bank measurement, a difference of about 25 per cent.

Similar figures are shown by the following tabulated statement of the dredges of the Natomas Consolidated for the year 1913, the actual capacity being 44.5 per cent of the theoretical for the time in commission, and 55 per cent for the total dredging time.

Record of dredges of the Natomas Consolidated of California for the year 1913.

[Comparison of actual with theoretical capacity.]

Dredge No.	Bucket size.	Depth worked.	Actual total capacity.	Working efficiency.	Actual as compared to theoretical capacity.		Running expense per cubic yard.	Total expense per cubic yard.
					Total time.	Dredging time.		
	Cubic feet.	*Feet.*	*Cubic yards.*	*Per cent.*	*Per cent.*	*Per cent.*	*Cents.*	*Cents.*
Natomas 1	13½	22	2,575,160	78.6	44.8	57.0	1.65	4.09
Natomas 2	8½	27	2,141,869	86.2	59.0	68.4	1.80	4.19
Natomas 3	8½	28	1,726,206	84.9	47.7	56.1	2.17	4.74
Natomas 4	13½	22	2,279,016	79.8	39.6	49.7	1.46	3.56
Natomas 5	9	53	1,048,519	80.3	50.3	62.7	2.03	11.05
Natomas 6	9	48	1,339,077	67.1	38.4	57.3	2.34	8.70
Natomas 7	9	55	1,156,930	80.5	47.6	59.1	2.39	4.65
Natomas 8	15	62	1,807,235	80.6	33.0	41.0	2.87	6.54
Natomas 9	15	64	2,363,888	84.0	40.7	48.4	2.44	6.69
Natomas 10	15	45	2,470,085	78.8	42.5	54.0	2.21	5.15
Feather 1	7½	26	929,308	83.4	32.0	38.4	2.59	6.47
Feather 2	7½	33	1,483,404	84.9	51.1	60.2	1.48	4.15
Feather 3	13½	27	3,090,081	82.2	59.1	72.0	1.82	4.46
Total or average	....	....	24,410,778	80.9	44.5	55.1	2.04	5.35

The following statement from the report of the Tongkah Harbour Tin Dredging Co., Siam, for the year ended September, 1914, clearly illustrates the contentions of the writer and is expressive of the results obtained when different methods of computing yardage are employed:

> The working costs were 5.740d. (11.48 cents) per yard, as against 3.186d. (7.632 cents) last year. The smaller yardage output of this year as compared to that of

previous years results from a change in the method of calculation. Formerly the yardage was estimated as the product of the number of hours worked multiplied by a yardage factor. Apparently the factor remained the same irrespective of the nature of the ground dredged. Now the yardage is computed from measured width of cut by distance pulled ahead and depth of face worked.

Yardage based on:

Theoretical capacity is based on buckets dumping per minute as follows: Nos. 1, 2, 3, 4, 7—22; all others, 20 per minute.

Efficiency equals total dredging time divided by time actually in commission during the year (less holidays).

Running expenses consist of labor, material, power, and water, and total expense includes repairs, which averaged $2.49, and taxes, insurance, general and other charges, which averaged $0.82 a cubic yard for all dredges.

When yardage costs are based on calculations of theoretical capacity, the advantage would lie with the operator having the most vivid imagination regarding dredge capacity; his operating costs per cubic yard would seemingly be reduced to a most desirable figure, though his total costs would not be affected.

The capacity of a dredge and the working costs per cubic yard do not as a general thing concern the individual operator or the shareholder, and it makes little difference to either how the figures are obtained. What does concern both is that the recovery exceeds the operating costs and leaves a profit. Only when it is desired to make some comparison of the operating cost and the capacity of different dredges do the cost per cubic yard and the method of arriving at the yardage handled become of importance.

Hence working costs can not be fairly used in such comparisons unless uniform and proper methods of determining them are employed and also unless operating conditions are somewhat similar. It is not generally recognized that ground in the same locality often varies so much that dredges similar in construction, design, and bucket capacity show different operating costs per cubic yard; neither is it generally recognized that figures for the same dredge may vary greatly in different years. A case in point is that of the Hunter dredge, a 5-foot close-connected boat of the Oro Light & Power Co., at Oroville. This dredge during operations extending over three years handled an average of about 200 cubic yards an hour. Afterwards the boat was moved to another part of the same property, where for a number of months, on account of difficult digging, it handled only 80 cubic yards an hour. Another example which may be quoted to illustrate different results of similar dredges is the following extract from an old report of Oroville Dredging (Ltd.):

> The 5-foot dredge of the Oroville Dredging & Exploration Co., handling a loose gravel which requires little power to displace, and with all of the conditions as nearly ideal from a dredging standpoint as could be wished, averaged for seven months 2,737 yards a day. Comparing this with the daily average of 1,723 yards obtained by

Dredge No. 2 of the Boston & California, a dredge having the same digging power and (theoretical) capacity, we have a striking illustration of the difference to be obtained in different formations in the same district.

The amount of ground handled by another Oroville dredge with 6-foot buckets also varied in different parts of the property. In some places a cemented overburden had to be broken with explosives; in others much heavy clay, difficult to dig, was encountered. The operating costs and yardage handled may vary considerably over different years, as shown in the following table:

Operating cost of Oroville dredge with 6-foot buckets.

Year.	Cubic yards.	Total operating costs.	Cost per cubic yard, cents.
1910 [a]	649,639	$36,764	5.66
1911	874,845	53,306	6.09
1912	710,257	50,150	7.06
1913	832,970	48,322	5.80
1914	819,721	50,587	7.03
	3,887,432	239,129	6.15

[a] 10 months.

In this connection the following statement is of interest regarding the different costs of Conrey No. 3 dredge: From April to July, 1912, and from August to November, 1913, inclusive, the dredge worked downstream and averaged 163.4 cubic yards a working hour, with an average operating cost of 5.40 cents a cubic yard. During the last four months of 1912 and several months of 1913 the dredge dug upstream and averaged 203.5 cubic yards an hour, at a cost of 4.87 cents, there being an increase of 25 per cent in yardage and a decrease of 10 per cent in cost in working upstream as compared with working downstream.

COSTS FOR SAME DREDGE MAY VARY BY YEARS.

Cost for different years at the same dredge may also vary greatly on account of the method frequently employed of charging all renewals and repairs to the year in which they are made even though the benefit from such work extends over several years. For instance, Exploration No. 3 dredge, of Oroville Dredging (Ltd.), handled 1,228,110 yards during 1912, at a cost of 4.18 cents a yard. According to the annual report of the company the cost of operation was reduced 1.54 cents a cubic yard below that of the previous year; this difference was due to the fact that the bucket line had been overhauled and charged for in 1911, and required little attention during 1912.

Another example of this practice is shown in the annual report of Natomas Consolidated for 1913. The cost of operation of Natoma

Nos. 5 and 6 for 1913 is given at 11.05 cents and 8.70 cents a cubic yard. This is due to the fact that extraordinary repair and reconstruction cost of 8.18 cents per yard for No. 5 and 5.34 cents per yard for No. 6 are charged against the operating costs of the dredges for the year; No. 5 being out of commission 4 months and No. 6 a little over 2 months while undergoing general overhauling and repair. A preceding table (p. 151) shows the wide difference between "running" expense and total cost of operation for all dredges of this company for 1913.

HOW OPERATING COSTS MAY INDICATE INEFFICIENCY.

A fact that must be kept in mind is that low operating costs per cubic yard may possibly mean low gold-saving efficiency. Although high yardage records and low operating cost per cubic yard are desirable the operator should strive for the best economic results. In ordinary gravel the winch man can easily crowd the tables, and the desire of the dredge master to get yardage records, as was often the case in early California dredging, may result in considerable unwashed gravel going to the tailing piles. On some dredges a careful study showed that it was advisable to reduce the speed of the buckets in order not to crowd the gold-saving tables, and the net result proved the economic success, at these dredges at least, of more thoroughly washing a smaller amount of gravel instead of crowding the screen and tables with all that the dredge could dig. In the Oroville field a large amount of the ground dug in early days was not thoroughly washed under these conditions and there was consequently a considerable loss of the gold content.

The buckets in use in California range in capacity from 3 to 15 cubic feet and some of the dredges being built in 1916 will have 17-foot buckets. Dredge operators, following California practice, prefer the close-connected bucket line to the open-connected type and also prefer digging spuds to headlines, having demonstrated to their entire satisfaction over a long period of experimenting, the superiority of these methods. In practice, the close-connected bucket line dumps at the rate of 18 to 22 buckets a minute, whereas the open-connected averages 12 to 14 buckets a minute. On some of the older boats when the change was made from open to close connected buckets it was claimed that the yardage was nearly doubled and the expense per cubic yard greatly decreased.

COST FIGURES FOR CERTAIN DREDGES.

To illustrate the advantage spuds have over headlines the following example may be given: Two 13-foot Natomas dredges, in ground of apparently the same character within 200 yards of each other, one working on spud, the other on headlines, showed a difference in capacity of 100 yards per hour in favor of the spud method. An instance where headlines proved satisfactory was that of the 18-foot dredge of the Boston & Idaho Co. On account of extremely easy digging no spuds have been used, though the dredge was designed with spud casings should operation have shown the advisability of adding spuds.

As has been stated, the working costs of a dredge may vary greatly for different years and the results of any one year's operation may not be either higher or lower than the general average. In this connection it is interesting to have the figures of operation for over the entire life of a dredge on a property.

Pacific No. 1, a 7½-foot close-connected dredge, went into commission in May, 1906, and worked continuously until the hull sprang a leak and sank December 3, 1911. The hull was repaired and the dredge continued at work until July 14, 1913, when it was dismantled. The machinery was then moved to Mammoth Bar and placed on a new hull, where it is now working. During the seven years of operation at Oroville the dredge handled a total of 9,500,000 cubic yards, from an average depth of 34 feet. Figures of cost for the entire period are furnished as follows:

Average working costs of Pacific No. 1 dredge.

Labor	$0.011
Power	.007
Repairs and supplies	.015
Taxes and insurance	.002
Miscellaneous	.002
Depreciation [a]	.007
Total	.044

Even more interesting than the figures above are the figures for a 3½-foot dredge working under average Oroville conditions for 7½ years.

Average working costs of a 3½-foot dredge.

Dredging time, days	2,248
Cubic yards dredged	3,246,836
Cubic yards per calendar day	1,132
Cubic yards, per operating hour	60
Labor cost, cents per cubic yard	3.0
Power cost, cents per cubic yard	1.8
Water cost, cents per cubic yard	.1
Repairs and supplies, cents per cubic yard	2.3
General expenses, cents per cubic yard	1
Total cost per yard	8.2

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[a] Depreciation was figured at a rate to return the cost of the dredge with an estimated life of 10 years.

The original cost of the boat was $52,014.75, and after operations ended the machinery was sold for $6,000, leaving a net cost for the dredge of $46,014.75, or 1.11 cents a cubic yard, making the complete and total cost of operations exclusive of the original cost of property and for prospecting approximately 9.3 cents a cubic yard of material dug.

Complete figures of similar scope for some of the larger dredges are not yet obtainable, as the dredges have not yet finished work on the ground for which they were built. The following table, showing total costs for the four Conrey dredges, however, are of interest as showing original cost of dredges and all charges of working:

Total costs of dredges of Conrey Placer Mining Co.

Item.	Dredge No.			
	1.	2.	3.	4.
Ledger balance, July 31, 1915	$35,129.37	$34,677.77	$66,600.00	$146,000.26
Depreciation to date	69,164.47	68,837.96	118,964.68	150,205.78
Cost of dredges	104,293.84	103,515.73	185,564.68	296,206.04
Charged to profit and loss:				
Raising No. 1	3,418.85			
Raising No. 2		3,893.98		
Reconstruction No. 2		20,778.59		
Reconstruction No. 3			12,880.48	
Total cost of dredges, including all changes and repairs not charged to operating costs	107,712.69	128,188.30	198,445.16	296,206.04
Total yardage from starting to July 31, 1915	7,817,520	5,515,756	8,166,281	9,880,333
Total operating costs from starting to July 31, 1915 [a]	$442,530.53	$382,739.41	$550,974.35	$479,115.94
Total in cents per yard	5.66	6.94	6.75	4.85
Upkeep of chain from starting to July 31, 1915	$80,500.00	$82,100.00	$123,000.00	$107,000.00
Total in cents per yard	1.03	1.49	1.51	1.06

Total yardage from starting to July 31, 1915, all dredges	31,379,890
Total operating costs from starting to July 31, 1915, all dredges	$1,855,360.23
Total in cents, per yard	5.91
Total cost of upkeep of chain from starting to July 31, 1915	$392,600.00
Total, in cents, per yard	1.25

[a] Including upkeep chain.

NOTE.—If the dredges were written off with above yardage, the total cost of operating would be increased by 2.32 cents a yard, but as the dredges have considerable life remaining, the cost of plant per yard will be considerably less than this figure.

The chief disadvantage of dredging in Montana as compared with California is the severity of the winter, the thermometer falling as low as 30° to 40° F. below zero. The company continues work throughout the year, except at No. 3 dredge, where the long sluice would make heating very expensive. Each dredge has a steam plant which is used to heat the boat and to furnish steam for attacking the ice that accumulates on ladders, rollers, sheaves, etc., and the frozen material that gathers in the hopper; during the coldest weather the suction of the dredge pumps will freeze if the dredge is shut down for short periods. Plate LXII shows the Conrey No. 3 dredge in winter.

The large boat of the Boston & Idaho Dredging Co. began work in May, 1911, and to December 31, 1915, had handled 17,417,894 cubic yards at a cost of 2.772 a cubic yard. The buckets were changed from 14 to 18 cubic feet in the latter part of 1913, greatly increasing the yardage; the amount of ground handled being 3,517,000 cubic yards in 1913, and 4,670,124 cubic yards in 1914.

Working costs for the entire period of operation are segregated as shown below. Power is generated at the company's hydroelectric plant and is charged at $4 a horsepower a month. Maintenance and repairs is an arbitrary figure for covering suspense account and charging off all supplies on hand.

Operating costs, Boston & Idaho Dredging Co., 1914.

	Total.	Per cubic yard. *Cents.*
Labor	$69,731.86	0.400
Power	113,317.15	.651
Maintenance and repairs	211,429.48	1.214
Supplies	12,322.51	.071
Superintendence and office, stable	47,946.08	.275
Sundries	6,184.33	.036
Taxes and insurance [a]	15,276.95	.089
Drilling, total charge	6,206.89	.036
Total cost	482,415.25	2.772

The depth of ground ranges from 14 to 36 feet. Digging is generally easy, but in places large bowlders and buried trees are encountered; the latter are either cut through by the buckets or so loosened as to float to surface. The original cost of the dredge was approximately $225,000, so although the dredge is still at work and the total yardage to be handled on the present property is not yet determined, it will be shown that the cost of the dredge per cubic yard dug will probably not exceed 1 cent.

The cost of electric power varies greatly. Some companies can obtain favorable rates because of the favorable situation of their properties or the feasibility of otherwise arranging their own proper plant. Some companies, such as the Boston & Idaho Co., operate their own power plant. The Boston & Idaho Co. makes a bookkeeping charge of $4 a horsepower-month for the power consumed on its dredge.

For the large companies power rates range from $\frac{3}{4}$ cent to $\frac{1}{8}$ cent a kilowatt-hour; for the smaller companies in different fields the power rate may be 1½ cents or more.

The power consumption and cost of power per cubic yard varies because of difference in first cost of electricity, and, in greater proportion, because of the conditions encountered in digging. The following table of power cost per cubic yard illustrates these variations:

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[a] Taxes do not include United States income tax or State "net profit" tax.

Power cost for dredges, 1914.

Dredge	Bucket capacity.	Average depth dug.	Yardage handled.	Power cost per cubic yard.
	Cu. ft.	*Feet.*	*Cu. yds.*	*Cents.*
Boston & Idaho	18	30	4,670,124	0.58
American Division No. 1, Pacific	7½	....	165,871	.75
Oroville Division No. 4	7½	....	1,353,594	.71
Natomas No. 4	13½	22	2,049,853	.50
Natomas No. 9	15	54	2,346,824	1.41
Feather No. 1	7½	33	1,815,079	.54
Yuba No. 6	7½	48	1,927,804	.66
Yuba No. 14	16	64	3,242,359	1.01
Conrey No. 4	16	50	3,108,933	.98
Conrey No. 3[a]	9½	34	1,068,144	1.44
Marysville No. 4	8	63	2,112,654	.78
Tonopah No. 1	9½	(?) 38	919,397	1.2
Tonopah No. 3[a]	5	(?) 45	743,120	2.4

[a] Flume dredge with a high power consumption.

As an example of the working costs of a small dredge in an isolated district, the following figures obtained from the manager's report are given. The figures should be of especial interest to the small operator or to persons who contemplate financing properties in the far north.

Costs at Berry dredge, Seward Peninsula, Alaska.

Operating time, June 3 to October 17; 3½ cubic feet buckets; operating days, 131; operating, 74.3 per cent; total ground handled, 213,355 cubic yards; average depth, 11.9 feet; some frozen ground; wood used as fuel cost $10 a cord on the bank.

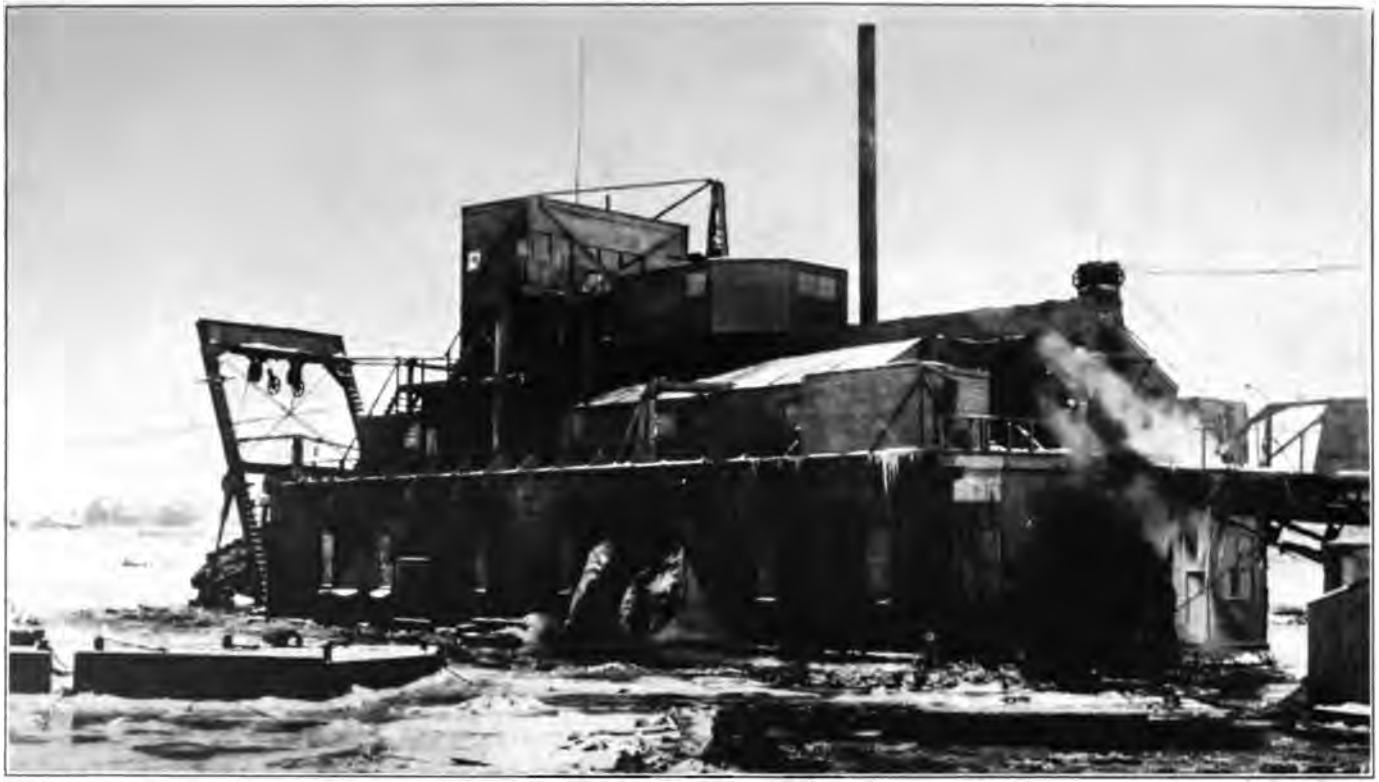
AMORTIZATION CHARGES INCLUDING GENERAL EXPENSE (estimated).

Items.	Total.	Cost per cubic yard.
		Cents.
Dredge labor, including traveling expenses	$12,565.46	5.89
Repairs and supplies used, 1915	3,682.09	1.72
Wood used, 1915	5,499.76	2.57
Organization expense apportioned to 1915	87.69	.04
Insurance and taxes	623.25	.28
Bullion express	822.37	.38
Office expense	721.56	.34
Total	24,002.18	11.24

ACTUAL EXPENSE FOR DREDGE OPERATION.

	Cents.
Depreciation of equipment, taken at $14,000 a year	6.56
Interest on investment ($161,000), $9,660 a year	4.52
Preparation of ground and general expense, divided among 600,000 cubic yards, $24,091.99	4.01
Total	15.09
Estimated gross expense per cubic yard	26.33

Mr. Hamil reports that the cost of boarding the whole force of men was 4,039 man-days at the rate of $2.02 a man-day	$8,158.78



CONREY NO. 3 DREDGE IN WINTER.

In writing this report and preparing the tables that show the segregation of working costs it was deemed advisable to use the individual tables of the different companies, because the different methods of apportioning costs render difficult the making of uniform tables for dredges under different managements. In addition, the author presents a table prepared for a paper read before the American Institute of Mining Engineers in 1912.[a] The costs given therein are practically the same as would prevail under similar conditions to-day.

A paper on dredging would be incomplete without inclusion of some reference to the work at the Panama Canal. According to W. G. Combey, in a paper on dredging in the Panama Canal presented at a meeting of the International Engineers' Congress in San Francisco, September, 1915, the grand total of material removed from the Panama Canal and auxiliary works by dredging from 1905 to December, 1914, was 115,700,000 cubic yards. The total cost, which includes work of ladder, dipper, and suction dredges and the total cost of operation, maintenance, material, and plant and all administrative charges, was 26.77 cents a cubic yard.

Operators of gold dredges have been interested in the performance of a large bucket-ladder dredge that had two lines of buckets. In one line the buckets had a capacity of 1 cubic yard, in the other 2 cubic yards. These bucket lines were interchangeable, one being used for heavy digging and the other where the work was lighter. The total material handled by this dredge from April, 1912, to January, 1915, is given at 335,000 cubic yards, and the average cost as 36.59 cents a yard.

The material from bucket, dipper and sea-going suction dredges was delivered to barges and hauled 4 to 10 miles to permanent dump. The intermittent nature of the work and the long haul accounts in a measure for the seemingly high operating cost. The table following prepared from Mr. Combey's report is given for such interest as it may have. Comparisons between the work of these dredges and those operating in different placer fields is of course futile.

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[a] Janin, Charles, Present-day problems in California gold dredging: Trans. Am. Inst. Min. Eng., vol., 42, 1912, p. 873.

Operating costs of Panama dredges.

Kind of dredge.	Capacity of buckets.	Depth dug.	Digging time.	Cubic yards handled.	Cost per cubic yard.[a]	Remarks.
	Cu. yds.	*Feet.*	*Months.*		*Cents.*	
Sea-going ladder	2	45	34	3,351,633	36.61	Material: Hard clay and rock; haul 3 to 4 miles.
Dipper	5	5	12	743,839	29.4	Mud, stiff clay, and rock; haul 3 to 4 miles.
Do	5	45	12	420,152	34.3	Mud, clay, coral rock, and rock; haul 5 to 6 miles.
Do	15	35.47	10	1,228,717	[b] 42.4	Hard and soft rock; haul 9 to 10 miles.
Do	15	35.47	8	939,508	[c] 42.9	Do.
	In. suc.					
Pipe-line suction	20	42	10	562,015	23.9	
Do	20	42	12	1,204,520	21.2	Mud, clay, gravel, and broken rocks, 3,000 feet; clay and broken rock; pipe line 7,000 feet long, using one relay.
Suction dredge delivering to barges.	....	45	12	3,099,022	10.9	Mud, sand, and clay; haul 4 to 5 miles.
Total quantity all material removed by dredging, 1905 to January, 1915.	....	....	....	115,722,733	26.7	

[a] Costs are for material placed on dump and include all operations, maintenance, material, plant, and administrative charges. Cost of plant on arbitrary figure assessed per cubic yard. Cost of different dredges approximated is as follows: 2-yard ladder, $400,000; 5-yard dipper, $95,000; 20-inch sea-going suction, $330,000; 20-inch pipe suction, $100,000 to $160,000.

[b] A credit of $23,680 for unused material has been allowed.

[c] A credit of $10,948 for unused material has been allowed.

Data on operating costs of dredges of the Yuba Consolidated Goldfields are shown in the table following.

Operating reports of Yuba Consolidated Goldfields for year ended February 28, 1915.

Dredge No.	Total dredging time.		Average daily time.		Per cent of possible running time.	Ground dredged.	Daily average.	Average depth.	Operation.										Insurance.		Taxes.		General expense.		Total expense.	
									Labor, cost per cubic yard.	Labor, per cent of total cost.	Material, cost per cubic yard.	Material, per cent of total cost.	Power, cost per cubic yard.	Power, per cent of total cost.	Repair labor, cost per cubic yard.	Repair labor, per cent of total cost.	Repair material, cost per cubic yard.	Repair material, per cent of total cost.	Cost per cubic yard.	Per cent of total cost.	Cost per cubic yard.	Per cent of total cost.	Cost per cubic yard.	Per cent of total cost.	Cost per cubic yard.	Per cent of total cost.
	Hrs.	*min.*	*H.*	*m.*		*Cu. yds.*	*Cu. yds.*	*Feet.*	*Cts.*		*Cts.*		*Cts.*		*Cts.*		*Cts.*		*Cts.*		*Cts.*		*Cts.*		*Cts.*	
3 [a]	3,095		20	22	85.1	434,765	2,860	64.3	1.54	30.1	0.07	1.3	1.04	20.4	0.40	7.8	1.26	24.7	0.14	2.7	0.13	2.6	0.53	10.4	5.11	.....
3 [a]	2,407	20	20	45	86.8	395,265	3,407	67.8	1.29	27.9	.03	0.6	1.21	26.2	.27	5.9	.91	19.6	.00	0.0	.21	4.5	.71	15.3	4.63	.....
4 [b]	1,211	15	20	53	87.1	205,908	3,550	63.1	1.19	17.4	.12	1.8	1.04	15.1	.91	13.2	2.83	41.2	.20	3.0	.11	1.6	.46	6.7	6.86	.....
4 [b]	3,453		21	11	88.5	632,667	3,881	65.1	1.13	31.8	.03	.09	1.00	28.3	.19	5.4	.39	10.9	.00	0.0	.18	5.2	.62	17.5	3.54	.....
6	7,504	55	20	40	86.3	1,927,805	5,310	48.1	.84	25.5	.03	.09	.66	20.1	.38	11.4	.98	29.3	.06	1.8	.07	2.2	.29	8.8	3.29	.....
7	7,742	25	21	20	89.0	1,485,390	4,091	67.4	1.08	19.2	.05	.09	.88	15.7	.53	9.5	2.52	44.9	.07	1.3	.10	1.7	.38	6.8	5.61	.....
8	7,859	20	21	39	90.4	1,705,750	4,699	77.6	.93	18.4	.05	1.0	.83	16.5	.53	10.4	2.23	44.2	.07	1.3	.08	1.6	.33	6.6	5.05	.....
9	7,426	50	20	28	85.4	1,215,740	3,349	67.0	1.32	22.9	.06	1.0	1.16	20.0	.64	11.0	1.93	33.3	.11	1.9	.11	1.9	.46	8.0	5.79	.....
11	7,276	50	20	3	83.7	1,315,306	3,623	63.0	1.17	24.2	.05	1.1	1.06	21.8	.65	13.5	1.27	26.2	.10	2.1	.11	2.2	.43	8.9	4.84	.....
12	7,213	5	19	52	83.0	1,541,379	4,246	52.1	.99	18.7	.04	.08	.91	17.2	.78	14.7	2.03	38.3	.09	1.7	.09	1.8	.36	6.8	5.29	.....
13	7,125	5	19	37	82.0	2,946,018	8,116	55.4	.66	17.0	.09	2.3	1.01	26.0	.36	9.2	1.30	33.6	.14	3.5	.13	3.3	.20	5.1	3.89	.....
14	7,740	15	21	19	89.1	3,242,359	8,932	70.0	.64	18.7	.08	2.4	1.04	30.2	.35	10.1	.92	26.8	.00	0.0	.23	6.7	.17	5.1	3.43	.....
Total	70,055	20	.....	.....	.....	17,048,352	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Average	.....		20	39	86.2	.....	5,024	61.4	.92	20.8	.06	1.4	.95	21.7	.47	10.7	1.47	33.4	.08	1.7	.13	3.0	.32	7.3	4.40	.....

[a] Dredge No. 3 dismantled July 31, 1914; reconstructed in steel, started work Nov. 5, 1914.
[b] Dredge No. 4 dismantled Apr. 27, 1914; reconstructed in steel, started work Sept. 18, 1914.

LABOR COSTS.

The greater efficiency and larger capacity of the modern, as compared with the earlier, dredges have steadily decreased labor costs per cubic yard in spite of shorter working hours and higher wages for the men. On some of the early dredges the labor alone cost as much per cubic yard as the total operating charges at present.

Wage scales vary somewhat in different localities, but are approximately the same in different areas in the same State. The following table gives the approximate scale of wages in California, Montana, and Alaska. In the latter country board is generally included, or an additional allowance is made for it.

Approximate scale of wages at dredges.[a]

	California.	Montana.	Alaska.
Dredge masters	$150 a month	$6 a day	$200 a month.
Winchmen	$4 a day	$5 a day	50 to 60 cents an hour.
Oilers	$3 to $3.50 a day	$3.50 a day	
Shoremen	$2.50 a day	$3.50 a day	
Helpers	$2.50 a day	$2.50 a day	

[a] Since this was compiled wages have advanced in some localities.

An indication of the number of men employed in dredging is the total force required by the Conrey Co. in different departments for the normal working of its dredges and for company business. The force is distributed as follows:

Men employed in dredging by Conrey Co.

Dredge crew No. 1 dredge	8
Dredge crew No. 2 dredge	8
Dredge crew No. 3 dredge	10
Dredge crew No. 4 dredge	10
Office and dwellings	4
Transformer station and pole line	2
Machine shop	8
Clean-up crew	3
Laborers	22
Miscellaneous	10
Total	85

In California the Natomas Consolidated employs a total of about 1,500 men in all branches of its operations.

ACCOUNTING METHODS OF NATOMAS CONSOLIDATED.

To illustrate the bookkeeping methods employed, the following list of account numbers used by the Natomas Consolidated and the distribution of repair charges for two of the dredges for the month of August, 1914, are presented·

JANUARY 1, 1915.

Gold-dredging department.

NATOMA DIVISION.

LIST OF ACCOUNT NUMBERS.

OPERATING "NATOMA NO. 1" DREDGE:

Running expense—

11	Labor.
12	Material.
13	Power.
14	Water.

Repairs—

15	Labor.
16	Material.

General expense—

17a	Restraining débris.
b	Levee.
c	Fire protection.
d	Accident insurance.
e	Division expense (including pro rata of company, Sacramento, auditing and purchasing expenses.

General items—

18a	Taxes.
b	Insurance.
c	Depreciation.

Natomas Consolidated of California.

DREDGE OPERATING COSTS.

DISTRIBUTION OF LABOR AND MATERIAL FOR REPAIRS, AUGUST, 1914.

NATOMA DIVISION.

Dredge No. 1:			
Labor			$574
Supplies—			
Miscellaneous lumber	$28.90		
3 return idlers	30.72		
2 wrecking sheaves and shackles	51.12		
1 G. E. 50 h. p. motor No. 88209	60.00		
12 forged U. T. heel plates	48.00		
13 perforated shaker plates	278.82		
4 motor bearings	41.50		
1 10-inch C. T. pump head, suction side	49.40		
750 feet 1-inch Leschen wire rope	236.63		
Sundries	144.73		
Hauling	93.60		
		$1,063.42	
Natoma shop—			
Repairs to main hopper	33.80		
Repairs, general	21.62		
		55.52	
			1,118.
Total			1,692.
Dredge No. 2:			
Labor			$239.
Supplies—			
100 bucket bushings	247.86		
24 bucket pins	524.16		
2 ladder rollers	112.04		
750 feet of ⅞-inch 6/19 Roebling wire rope	94.58		
90, 30, 40¼, 32, and 43½ riffles	100.30		
24 rib bars, job 569	95.76		
Sundries	136.27		
Hauling	145.71		
		1,456.68	
Natoma shop, repairs, general		11.21	
			1,467.
Total			1,706.

MISCELLANEOUS COST TABLES AND FORMS.

The tables and forms following need no special comment, as they are self-explanatory. The figures presented should prove of considerable value. Moreover the forms used for recording them are of interest.

Operating costs of dredge

OROVILLE DIVISION, DREDGE NO. 1.

Item No.	Period covered.	Hours run.	Cubic yards worked.	Yards per hour.	Total costs.					
					Labor.	Power.	Repairs and supplies.	Taxes and insurance.	Miscellaneous expenses.	Dep- ciatio
1	Total or average, 7 months, 1906.	3,773	602,803	160	$7,742.75	$6,227.64	$7,566.91	$1,489.67	$1,220.10	$5,
2	Total or average, 1907.	6,889	1,109,226	161	13,408.15	11,148.10	18,245.08	2,553.72	4,974.31	9,
3	Total or average, 1908.	6,774	1,365,892	202	14,040.98	8,980.70	21,949.06	3,712.79	2,691.99	9,
4	Total or average, 1909.	7,048	1,215,214	172	14,590.22	8,548.69	18,857.95	2,113.63	2,878.94	9,
5	Total or average, 1910.	7,054	1,488,503	211	14,403.01	9,124.97	23,691.37	1,460.40	2,928.68	9,
6	Total or average, 1911.	6,616	1,495,331	226	15,571.36	8,419.87	19,227.34	2,920.80	2,938.43	9,
7	Total or average, 1912.	6,687	1,498,504	224	14,914.08	9,649.80	27,690.23	2,920.80	4,905.26	9,0
8	Total or average, 6 months, 1913.	3,722	730,746	196	6,636.40	5,092.20	3,202.18	1,460.40	1,896.90	4,5

OROVILLE DIVISION, DREDGE NO. 2.

Item No.	Period covered.	Hours run.	Cubic yards worked.	Yards per hour.	Labor.	Power.	Repairs and supplies.	Taxes and insurance.	Miscellaneous expenses.	Dep- ciatio
9	Total or average, 7 months, 1906.[1]	3,699	217,274	59	$6,543.19	$3,406.12	$7,186.94	$350.00	$1,477.95	$2,3
10	Total or average, 1907.[2]	6,148	469,253	76	10,226.90	4,827.27	13,636.32	600.00	4,942.29	3,9
11	Total or average, 1908.	6,808	709,508	104	10,839.62	4,511.10	15,834.90	1,550.07	2,626.46	3,9
12	Total or average, 1909.	5,487	609,442	111	11,731.75	3,859.80	5,434.14	1,162.72	2,931.62	3,9
13	Total or average, 1910.	7,116	683,198	96	11,237.88	5,417.45	7,890.57	835.20	2,964.44	3,9
14	Total or average, 1911.	6,792	757,091	111	10,791.12	4,626.80	13,961.53	1,531.20	2,868.05	3,6

OROVILLE DIVISION, DREDGE NO. 3.

Item No.	Period covered.	Hours run.	Cubic yards worked.	Yards per hour.	Labor.	Power.	Repairs and supplies.	Taxes and insurance.	Miscellaneous expenses.	Dep- ciatio
15	Total or average, 7 months, 1906.	4,265	506,284	181	$6,594.51	$4,902.48	$6,188.54	$551.25	$1,374.72	$3,5
16	Total or average, 1907.	7,002	884,203	126	11,363.63	7,627.20	13,965.27	945.00	5,358.78	6,0
17	Total or average, 1908.	7,453	989,966	133	11,018.23	6,914.80	15,242.28	2,578.72	2,722.08	6,0
18	Total or average, 1909.	6,964	803,429	115	12,032.39	5,709.75	13,481.67	1,510.02	2,884.49	6,0
19	Total or average, 1910.	7,162	916,194	128	11,992.46	6,434.45	12,309.08	1,044.00	2,908.30	6,0
20	Total or average, 1911.	7,265	712,142	98	12,135.82	7,333.80	10,879.58	2,088.00	2,948.83	6,0
21	Total or average, 1912.	5,727	688,733	120	8,853.23	5,865.61	8,354.80	1,566.00	2,576.91	4,5

[1] Operating on tract M, 30 per cent royalty.

[2] Operating 4 months on tract M, 30 per cent royalty.

Pacific Gold Dredging Co.

OROVILLE DIVISION, DREDGE NO. 1.

Total costs—Contd.		Costs per yard, cents.								Acres worked.	Item No.
Express and treatment.	Total.	Labor.	Power.	Repairs and supplies.	Taxes and insurance.	Miscellaneous expenses.	Depreciation.	Express and treatment.	Total.		
$206.62	$29,703.69	1.284	1.033	1.255	0.347	0.202	0.870	0.034	4.927	11.26	1
319.41	59,648.67	1.209	1.006	1.645	.230	.449	.811	.029	5.378	19.96	2
438.06	60,813.58	1.028	.657	1.607	.272	.197	.659	.032	4.452	23.47	3
352.31	56,341.74	1.201	.703	1.552	.174	.237	.740	.029	4.636	21.82	4
376.68	60,985.11	.967	.613	1.592	.098	.197	.605	.025	4.097	28.09	5
314.57	58,392.37	1.041	.563	1.286	.195	.197	.602	.021	3.905	32.66	6
280.48	69,360.65	.995	.644	1.848	.195	.328	.600	.019	4.629	27.10	7
135.18	22,923.26	.908	.697	.438	.200	.259	.616	.019	3.137	11.61	8
OROVILLE DIVISION, DREDGE NO. 2.											
$101.22	$21,396.42	3.011	1.567	3.308	0.161	0.680	1.073	0.046	9.846	....	9
184.96	38,413.74	2.179	1.029	2.906	.128	1.053	.852	.039	8.186	8.32	10
305.68	39,633.83	1.528	.636	2.232	.218	.370	.563	.043	5.590	15.16	11
149.38	29,265.41	1.925	.633	.892	.191	.481	.656	.024	4.802	13.76	12
182.98	32,524.52	1.645	.793	1.155	.122	.434	.585	.027	4.761	14.26	13
164.78	37,606.48	1.425	.611	1.844	.202	.379	.484	.022	4.967	13.83	14
OROVILLE DIVISION, DREDGE NO. 3.											
$166.90	$23,228.40	1.303	0.968	1.213	0.109	0.272	0.691	0.033	4.589	9.81	15
233.12	45,493.00	1.285	.863	1.579	.107	.606	.679	.026	5.145	18.52	16
269.44	44,740.55	1.112	.698	1.540	.260	.275	.606	.027	4.518	18.13	17
235.19	41,853.51	1.498	.711	1.678	.188	.358	.747	.029	5.209	15.36	18
285.47	40,973.71	1.309	.702	1.343	.114	.318	.655	.031	4.472	16.73	19
186.94	41,571.47	1.704	1.030	1.528	.293	.414	.843	.026	5.838	12.36	20
141.31	31,857.87	1.285	.852	1.213	.227	.374	.653	.021	4.625	11.73	21

Operating costs of dredges of Paci

OROVILLE DIVISION, DREDGE NO. 4.

Item No.	Period covered.	Hours run.	Cubic yards worked.	Yards per hour.	Total costs.					
					Labor.	Power.	Repairs and supplies.	Taxes and insurance.	Miscellaneous expenses.	Depr ciatio
22	Total or average, 7 months, 1906.	......	......	......	......	......	......	......	......	......
23	Total or average, 1907.	......	......	......	......	......	......	......	......	......
24	Total or average, 1908.	6,321	1,491,910	236	$13,040.95	$9,704.07	$14,447.31	$3,499.88	$3,582.60	$9, 1
25	Total or average, 1909.	7,091	1,371,263	193	14,410.53	9,120.75	18,918.69	2,113.63	2,884.07	9, 9
26	Total or average, 1910.	7,410	1,383,332	187	14,140.09	9,723.38	21,350.21	1,460.40	2,928.81	9, 9
27	Total or average, 1911.	7,372	1,383,795	188	14,627.24	10,225.88	12,613.20	2,920.80	2,920.38	9, 9
28	Total or average, 1912.	6,999	1,556,579	222	14,051.40	10,098.04	19,900.62	2,920.80	4,125.40	9, 9
29	Total or average, 1913.	7,501	1,444,533	192	13,519.77	10,866.20	19,501.08	4,140.88	5,487.95	9, 9
	1914.									
30	January	611	99,104	162	1,480.55	870.67	1,669.28	307.45	407.91	8
31	February	518	104,866	202	1,100.00	682.40	1,229.00	326.16	360.58	8
32	March	666	128,574	193	1,304.55	882.20	1,952.65	326.16	351.79	8
33	April	647	116,883	181	1,118.55	864.00	1,327.99	326.16	366.19	8
34	May	549	120,218	219	1,249.07	720.00	883.51	334.12	352.95	8
35	June	461	95,990	208	1,433.32	587.20	1,436.91	332.25	374.98	8
36	July	635	113,952	180	1,111.09	802.20	222.88	254.63	386.48	8
37	August	652	111,176	170	966.30	858.00	1,148.94	192.13	275.01	8
38	September	635	117,755	185	1,219.20	834.40	174.24	323.88	275.37	8
39	October	655	125,902	192	1,111.58	894.40	840.00	257.74	286.84	8
40	November	500	99,582	199	1,277.40	682.80	1,362.40	257.74	421.37	8
41	December	660	119,592	181	1,157.35	898.20	1,815.49	200.84	287.70	8
42	Total or average	7,189	1,353,594	188	14,528.96	9,576.47	14,063.24	3,438.76	4,147.12	9, 9

AMERICAN DIVISION, DREDGE NO. 1.

Item No.	Period covered.	Hours run.	Cubic yards worked.	Yards per hour.	Labor.	Power.	Repairs and supplies.	Taxes and insurance.	Miscellaneous expenses.	Depr ciatio
43	Total or average, 3 months, 1913.	1,801	374,992	208	$4,282.24	$2,892.00	$4,530.53	$180.81	$2,047.21	$3, 0
	1914.									
44	January	518	119,368	230	1,364.07	884.00	1,444.09	157.49	663.91	1, 0
45	February	463	87,783	190	1,082.59	924.00	2,256.55	129.17	639.74	1, 0
46	March	652	122,970	190	1,206.93	944.00	3,910.88	129.17	627.39	1, 0
47	April	667	150,521	226	1,169.04	1,204.00	1,056.86	129.17	652.39	1, 0
48	May	668	156,663	234	1,196.72	1,104.00	1,009.42	133.32	594.22	1, 0
49	June	661	171,510	259	1,178.70	1,092.00	1,916.28	133.32	617.67	1, 0
50	July	678	133,614	197	1,336.95	1,068.00	3,713.07	218.74	605.17	1, 0
51	August	660	123,457	187	1,179.83	1,194.80	1,481.91	156.23	496.47	1, 0
52	September	600	121,316	202	1,098.55	957.40	365.51	289.57	527.47	1, 0
53	October	620	152,103	245	1,202.03	1,165.60	3,254.09	221.84	526.73	1, 0
54	November	633	153,400	242	1,136.28	1,046.40	1,449.12	221.84	736.84	1, 0
55	December	644	166,012	258	1,210.05	952.00	473.05	151.05	520.65	1, 0
56	Total or average	7,464	1,658,717	222	14,361.74	12,536.20	22,330.83	2,070.91	7,208.65	1, 2

BUTTE CREEK DIVISION, DREDGE NO. 3.

Item No.	Period covered.	Hours run.	Cubic yards worked.	Yards per hour.	Labor.	Power.	Repairs and supplies.	Taxes and insurance.	Miscellaneous expenses.	Depr ciatio
	1914.									
57	January	......	......	......	......	......	......	......	......	......
58	February	......	......	......	......	......	......	......	......	......
59	March	......	......	......	......	......	......	......	......	......
60	April	......	......	......	......	......	......	......	......	......
61	May	......	......	......	......	......	......	......	......	......
62	June	......	......	......	......	......	......	......	......	......
63	July	......	......	......	......	......	......	......	......	......
64	August	......	......	......	......	......	......	......	......	......
65	September	......	......	......	......	......	......	......	......	......
66	October	525	45,156	86	$923.70	$423.36	$218.20	$110.00	$350.89	$50
67	November	545	69,114	127	935.85	580.23	843.02	110.00	468.01	70
68	December	570	71,611	125	1,060.61	538.65	827.73	110.00	399.68	60
69	Total or average	1,640	185,881	113	2,920.16	1,542.24	1,388.95	330.00	1,218.58	1,80

Gold Dredging Co.—Continued.

OROVILLE DIVISION, DREDGE NO. 4.

Total costs—Contd.		Costs per yard, cents.									
Express and treatment.	Total.	Labor.	Power.	Repairs and supplies.	Taxes and insurance.	Miscellaneous expenses.	Depreciation.	Express and treatment.	Total.	Acres worked.	Item No.
....	....	....	....	....	....	....	....	....	....	....	22
....	....	....	....	....	....	....	....	....	....	....	23
$332.63	$53,770.54	0.874	0.650	0.968	0.235	0.240	0.614	0.022	3.603	31.86	24
297.20	57,740.87	1.051	.665	1.379	.154	.211	.729	.022	4.211	27.67	25
361.52	59,960.41	1.022	.703	1.543	.106	.212	.723	.026	4.334	25.00	26
322.63	53,626.13	1.057	.739	.912	.211	.211	.722	.023	3.875	27.06	27
300.17	61,392.43	.908	.649	1.278	.188	.265	.642	.019	3.944	30.73	28
347.71	63,859.59	.936	.752	1.350	.287	.380	.692	.024	4.421	23.20	29
25.59	5,594.45	1.494	.879	1.684	.310	.412	.840	.026	5.645	1.45	30
25.81	4,556.95	1.049	.651	1.172	.311	.344	.794	.024	4.345	1.75	31
60.62	5,710.97	1.015	.686	1.518	.253	.274	.648	.049	4.442	2.19	32
35.67	4,871.56	.957	.739	1.136	.279	.313	.713	.031	4.168	1.61	33
16.49	4,389.14	1.039	.599	.735	.278	.293	.693	.014	3.651	1.56	34
31.00	5,028.66	1.493	.612	1.497	.346	.391	.868	.032	5.239	1.20	35
24.57	3,634.75	.975	.704	.195	.223	.340	.731	.022	3.190	1.66	36
33.42	4,306.80	.869	.772	1.034	.173	.247	.749	.030	3.874	1.73	37
19.58	3,679.17	1.035	.709	.148	.275	.234	.707	.016	3.124	1.67	38
22.51	4,246.07	.883	.710	.667	.205	.228	.662	.018	3.373	1.70	39
17.39	4,852.10	1.283	.686	1.368	.258	.423	.837	.017	4.872	1.19	40
69.07	5,261.65	.968	.751	1.518	.168	.240	.696	.058	4.399	1.74	41
381.72	56,132.27	1.073	.708	1.039	.254	.306	.738	.029	4.147	19.45	42

AMERICAN DIVISION, DREDGE NO. 1.

Express and treatment.	Total.	Labor.	Power.	Repairs and supplies.	Taxes and insurance.	Miscellaneous expenses.	Depreciation.	Express and treatment.	Total.	Acres worked.	Item No.
$133.51	$17,066.30	1.142	0.771	1.208	0.048	0.546	0.800	0.036	4.551	7.66	43
48.33	5,561.94	1.143	.741	1.209	.132	.556	.838	.040	4.659	2.52	44
19.03	6,051.08	1.233	1.053	2.571	.147	.729	1.139	.021	6.893	1.71	45
20.53	7,838.90	.962	.768	3.180	.105	.510	.813	.017	6.375	2.20	46
26.96	5,238.37	.777	.800	.702	.086	.433	.664	.018	3.480	2.59	47
20.76	5,058.44	.764	.705	.644	.085	.379	.639	.013	3.229	3.13	48
39.84	5,977.81	.687	.637	1.118	.077	.360	.583	.023	3.485	3.12	49
70.32	8,012.25	1.001	.799	2.779	.163	.453	.749	.053	5.997	2.57	50
58.57	5,567.81	.956	.968	1.200	.126	.402	.810	.047	4.509	2.43	51
63.06	4,301.56	.906	.789	.301	.239	.435	.824	.052	3.546	2.57	52
36.21	7,406.50	.790	.767	2.139	.146	.346	.657	.024	4.869	3.64	53
102.89	5,693.37	.741	.682	.945	.144	.480	.652	.067	3.711	3.45	54
165.06	4,471.86	.729	.573	.285	.091	.314	.602	.099	2.693	2.67	55
671.56	71,179.89	.866	.756	1.346	.125	.434	.723	.041	4.291	32.60	56

BUTTE CREEK DVISION, DREDGE NO. 3.

Express and treatment.	Total.	Labor.	Power.	Repairs and supplies.	Taxes and insurance.	Miscellaneous expenses.	Depreciation.	Express and treatment.	Total.	Acres worked.	Item No.
....	....	....	....	....	....	....	....	....	....	....	57
....	....	....	....	....	....	....	....	....	....	....	58
....	....	....	....	....	....	....	....	....	....	....	59
....	....	....	....	....	....	....	....	....	....	....	60
....	....	....	....	....	....	....	....	....	....	....	61
....	....	....	....	....	....	....	....	....	....	....	62
....	....	....	....	....	....	....	....	....	....	....	63
....	....	....	....	....	....	....	....	....	....	....	64
....	....	....	....	....	....	....	....	....	....	....	65
$9.82	$2,535.97	2.045	0.938	0.483	0.244	0.777	1.107	0.022	5.616	1.85	66
17.39	3,654.50	1.354	.839	1.220	.119	.677	1.018	.025	5.287	2.16	67
36.86	3,073.53	1.481	.752	.458	.184	.888	.838	.051	4.292	2.38	68
64.07	9,264.00	1.571	.830	.747	.188	.855	.969	.034	4.983	6.39	69

Balance sheet used by Conrey Placer Mining Co.[a]

Assets.			Increase.	Decrease.
Cash mines bank				
Cash mines office				
Cash with treasurer				
Bullion in hand				
Accounts receivable (list)				
Stock—warehouse (list)				
Stock—fuel				
Prepaid accounts				
Dredge No. 1				
Dredge No. 2				
Dredge No. 3				
Dredge No. 4				
Machine shop and tools				
Minor personal property				
Lands, buildings, etc				
Substation and pole line				
Deferred charges (list)				

Liabilities.			Increase.	Decrease.
Accounts payable (list)				
Notes payable				
Accounts accruing				
Capital stock				
Reserve for upkeep of chain				
Surplus account				

.... *Bookkeeper.* .... *Manager.*

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[a] Jennings, Hennen, History and development of gold dredging in Montana: Bull. 121, Bureau of Mines, 1916, p. 25.

Profit-and-loss report used by Conrey Placer Mining Co.[a]

Profit-and-Loss Report for Month of ....

Cubic yards dredged.	Month of..............		Year to date...months.	
Dredge No. 1				
Dredge No. 2				
Dredge No. 3				
Dredge No. 4				
Total				
Operating costs.		Average per yard.	Months.	Average per yard.
Dredge No. 1				
Dredge No. 2				
Dredge No. 3				
Dredge No. 4				
Total				
Indirect expenses.				
Salaries, mines				
Salaries, Boston				
Expenses, Boston				
Interest				
Insurance				
Taxes				
Telephones				
Office expense				
Building repairs				
Living quarters				
Miscellaneous expenses				
Stable expense				
Maintenance substation and pole line				
Roads, ditches, and fences				
Prospect drilling				
Ranch expense				
Total indirect expenses				
Total operating costs				
Total expenses				
Balance net earnings to surplus account				
Net bullion and earnings.				
Dredge No. 1				
Dredge No. 2				
Dredge No. 3				
Dredge No. 4				
Net bullion, total				
Ranch returns				
Rents				
Lights				
Scrap				
Interest receipts				
Shop receipts				
Miscellaneous receipts				

.... *Bookkeeper.* .... *Manager.*

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[a] Jennings, Hennen, work quoted, p. 26.

Form of operating report used by Conrey Placer Mining Co.[a]

Operating Report for Month of .... Dredge No. ....

Time.	Month of	Year to date	months.
Running hours	per cent.	Hours	per cent.
Lost hours	per cent.	Hours	per cent.
Total hours	100 per cent.	Hours	100 per cent.

Excavation.		Months.
Cubic yards dredged .... virgin..		
Cubic yards dredged .... tailings..		
Cubic yards dredged .... gross..		
Average per running hour		
Average per bucket		
Buckets per minute		
Average kilowatt-hours per yard		
Acres dredged		
Average depth to bedrock		

Production.		Average per yard.	Months.	Average per yard.
Gross bullion				
Less express				
Net bullion to profit-and-loss report				
Operating cost.				
Labor				
Fuel				
Power				
Supplies				
Upkeep of chain				
Repairs:				
Digging appliances, labor				
Digging appliances, material				
Pumping appliances, labor				
Pumping appliances, material				
Gold saving and washing appliances, labor				
Gold saving and washing appliances, material				
Electrical appliances, labor				
Electrical appliances, material				
Heating appliances, labor				
Heating appliances, material				
Hull and housing, labor				
Hull and housing, material				
Miscellaneous				
Totals to profit-and-loss report				

.... *Bookkeeper.* .... *Manager.*

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[a] Jennings, Hennen, work quoted, p. 27.

Summary of yearly yardage and expenses, Conrey Placer Mining Co., Sept. 1, 1906, to July 31, 1915.[a]

Year.	Cubic yards excavated.	Average operating costs per cubic yard, in cents.							Indirect expenses.[b]	Total.
		Labor.	Fuel.	Power.	Supplies.	Upkeep of chain.	Repairs.	Total.		
1906	322,174	1.65	0.08	1.59	0.20	1.40	1.07	5.99	3.80	9.79
1907	1,152,478	1.34	.06	1.34	.04	.87	1.12	4.77	2.63	7.40
1908	2,248,576	1.45	.08	1.69	.09	1.50	1.45	6.26	1.12	7.38
1909	2,429,524	1.60	.11	1.30	.09	1.69	1.89	7.61	1.22	8.83
1910	2,670,354	1.66	.13	1.29	.10	1.53	1.97	6.88	1.11	7.99
1911	3,566,691	1.29	.12	1.22	.09	1.31	1.80	6.11	1.04	7.15
1912	4,463,354	1.41	.17	1.34	.13	1.37	2.34	6.82	1.05	7.87
1913	4,625,436	1.37	.12	1.36	.09	1.09	1.93	5.96	.92	6.88
1914	6,268,626	1.13	.07	1.15	.09	.94	1.25	4.63	.73	5.36
1915[c]	3,632,677	1.13	.06	1.15	.08	.98	1.65	5.05	.94	5.99
Total	31,379,890	1.37	.11	1.32	.10	1.00	1.76	5.91	1.05	6.96

[a] Jennings, Hennen, work quoted, p. 28.

[b] Includes salaries, Ruby and Boston, insurance, taxes, maintenance of pole line and substation, roads, ditches, bonus to employees, legal expenses, accidents, all office expenses, etc., but interest on loans and other indebtedness is excluded.

[c] To July 31.

Operating report of Conrey Placer Mining Co., covering electric dredges Nos. 1, 2, 3, and 4, from starting to July 31, 1915.[a]

TIME.

Item.	Dredge No. 1.	Dredge No. 2.	Dredge No. 3.	Dredge No. 4.	Total.
Running hours, per cent	71	67	65	65	67
Lost hours, per cent	29	33	35	35	33
Total	100	100	100	100	100
EXCAVATION DATA.					
Cubic yards dredged, virgin ground	7,817,520	5,515,756	8,166,281	9,880,333	31,379,890
Cubic yards dredged, tailings[b]	72,678	752,240	479,872	145,192	1,449,982
Cubic yards dredged, gross	7,890,198	6,267,995	8,646,153	10,025,525	32,829,872
Average per running hour	174	132	161	420	195
Average per bucket	.18	.18	.19	.38	....
Buckets per minute	15.75	12.98	14.15	18.62	....
Average kilowatt-hours per yard	1.55	1.89	1.73	1.36	....
Acres dredged	223.16	143.62	144.43	134.92	646.13
Average depth to bedrock, feet	22.1	28.1	40.8	45.3	....
OPERATING COSTS, AVERAGE COST PER YARD, IN CENTS.					
Labor	1.50	1.68	1.75	0.79	1.37
Fuel	.11	.13	.13	.07	.11
Power	1.27	1.55	1.48	1.09	1.32
Supplies	.10	.11	.09	.09	.10
Upkeep of chain	1.03	1.49	1.51	1.08	1.25
COST OF REPAIRS, IN CENTS.					
Digging appliances, labor	0.19	0.32	0.28	0.23	0.25
Digging appliances, material	.32	.46	.62	.61	.51
Pumping appliances, labor	.04	.02	.03	.01	.02
Pumping appliances, material	.07	.03	.04	.01	.04
Gold saving and washing appliances, labor	.28	.28	.22	.17	.23
Gold saving and washing appliances, material	.60	.66	.36	.57	.54
Electrical appliances, labor	.04	.05	.05	05	.05
Electrical appliances, material	.03	.04	.06	.04	.04
Heating appliances, labor	.01	....	.01	....	....
Heating appliances, material	.02	.02	.02	....	.01
Hull and housing, labor	.01	.05	.04	.01	.03
Hull and housing, material	.01	.02	.01	.01	.01
Miscellaneous	.03	.03	.05	.02	.03
Total operating costs	5.66	6.94	6.75	4.85	5.91
Indirect expenses, excluding interest[c]	....	....	....	....	1.05
Grand total	....	....	....	....	6.96

[a] Jennings, Hennen, work quoted, p. 29.

[b] The tailings dredged are not used in calculating costs.

[c] In indirect expenses are included salaries, Ruby and Boston, insurance, taxes, maintenance of pole line and substation, roads, ditches, bonus to employees, legal expenses, accidents, all office expenses, etc., but interest on loans and other indebtedness is excluded.

Operating costs of dredges of Natoma

Dredge No.	Size of buckets.	Time.								
		Total days.	Total hours.	Lost.		Dredging.		Efficiency.	Average dredging per day.	
Natoma Division.				*Hrs.*	*min.*	*Hrs.*	*min.*	*P. ct.*	*Hrs.*	*min.*
1	13½	363	8,712	1,606	55	7,105	5	81.6	19	34
2	8½	363	8,712	1,109	25	7,602	35	87.3	20	57
3	8½	364	8,736	1,564	10	7,171	50	82.1	19	42
4	13½	303	7,272	1,204	40	6,067	20	83.4	20	1
5	9	363	8,712	1,542	10	7,169	50	82.3	19	45
6	9	363	8,712	1,452		7,260		83.3	20	
7	9	363	8,712	2,322	30	6,389	30	73.3	17	36
8	15	363	8,712	2,341	30	6,370	30	73.1	17	33
9	15	363	8,712	1,481	55	7,230	5	83.0	19	55
10	15	363	8,712	2,183	55	6,528	5	74.9	17	59
F. R. Division.										
1	7½	363	8,712	902	25	7,809	35	89.6	21	31
2	7½	363	8,712	1,212	35	7,499	25	86.1	20	40
3	13½	363	8,712	1,251	45	7,460	15	85.6	20	33
Natoma Division		3,571	85,704	16,809	10	68,894	50	80.4	19	18
F. R. Division		1,089	26,136	3,366	45	22,769	15	87.1	20	55
Both divisions		4,660	111,840	20,175	55	91,664	5	82.0	19	40

Consolidated of California.

Average depth.	Total ground worked.	Ground worked per day.	Area dredged.	Cost per cubic yard, cents.											
				Total expense.	Running expense.					Repairs.		General expenses.	Smelting and express.	Taxes.	Insurance.
					Labor.	Material.	Power.	Water.	Total.	Labor.	Material.				
Feet.	*Cu. yds.*	*Cu. yds.*	*Acres.*												
24	2,884,250	7,946	75.9584	3.93	0.59	0.06	0.70	0.08	1.43	0.19	1.75	0.41	0.03	0.08	0.04
21	1,816,849	5,005	54.0199	3.89	1.03	.18	.71	.19	2.11	.19	.95	.44	.03	.08	.09
37	1,617,392	4,443	27.0221	5.33	1.13	.15	.75	.14	2.17	.21	2.29	.48	.03	.09	.06
22	2,049,853	6,765	56.8314	2.67	.65	.05	.50	.07	1.27	.17	.59	.49	.03	.05	.07
54	1,802,644	4,966	20.5995	4.27	.77	.08	.84	.07	1.76	.33	1.56	.43	.03	.08	.08
40	1,396,212	3,846	21.6344	6.49	1.29	.28	1.06	.13	2.76	.28	2.64	.56	.03	.12	.10
60	1,469,431	4,048	14.8511	5.82	1.01	.14	1.25	.11	2.51	.52	1.99	.53	.07	.17	.03
67	2,053,568	5,657	19.0005	7.14	.84	.15	1.32	.10	2.41	.43	3.58	.57	.03	.10	.02
54	2,346,824	6,465	27.1771	5.25	.83	.18	1.41	.11	2.53	.26	1.74	.50	.03	.10	.09
37	2,273,942	6,264	38.0307	5.26	.88	.17	1.09	.12	2.26	.30	1.96	.51	.07	.14	.02
33	1,815,079	5,000	33.8020	2.50	.80	.09	.54	.....	1.43	.08	.47	.34	.02	.07	.09
36	1,509,316	4,158	26.7344	3.01	.87	.06	.60	.....	1.53	.13	.64	.41	.04	.15	.11
41	2,818,239	7,764	42.6772	3.31	.80	.15	.94	.....	1.89	.09	.77	.33	.03	.12	.08
34	19,710,965	5,520	355.0851	4.91	.87	.13	.96	.11	2.07	.28	1.88	.49	.03	.10	.06
38	6,142,634	5,641	103.2136	2.99	.82	.11	.73	.....	1.66	.10	.65	.35	.03	.11	.09
....	25,853,619	5,548	458.2987	4.45	.85	.13	.91	.08	1.97	.24	1.59	.45	.03	.10	.07

Working costs of gold dredging in California.[a]

Capacity of buckets.	Time in commission.		Working period for figures given.		Actual working time; hours during working period.[b]	Quantity handled.	Average depth of gravel.	Operating expenses, cents per cubic yard.							Remarks.
								Labor and material.	Electric power.	Water.	Repairs.	General.	Taxes and insurance.	Total expense.	
Cu. ft.	*Yrs.*	*Mo.*	*Yrs.*	*Mo.*		*Cu. yds.*	*Ft.*								
3	....		1		2,809	173,655	27.0	2.77	0.90	0.14	4.15	0.78	0.49	9.23	Difficult digging.[c]
3	5	9	1		7,216	458,882	26.9	2.03	.69	....	3.28	.63	.37	7.00	Working under favorable conditions.
3.5	7		1		....	395,316	35.0	2.83	1.53	.228	1.74	1.32	....	7.67	
3.5	6	6	1		7,344	461,882	35.0	2.85	1.48	.195	1.71	1.07	....	7.32	Compact gravel land subject to overflow.
4	9		1		7,057	484,387	20.6	1.83	.89	.31	2.58	.65	.26	6.52	Remodeled dredge, uneven bedrock, in places shallow.
5	6		1		....	481,184	25.0	3.28	1.46	.39	2.97	1.52	....	9.55	Difficult ground, in places cemented gravel.
5	2	5	1		....	635,146	27.0	3.14	1.45	....	2.40	1.28	.41	8.70	Difficult ground.
5	2		1		....	582,891	30.0	3.28	2.02	.32	2.59	1.37	1.00	9.60	Difficult digging.
5	5	6	1		7,344	615,009	25.0	3.06	1.42	.29	3.06	1.14	....	8.98	Do.
5	4	6	1		....	812,355	36.0	[d] 2.30	1.08	....	2.95	....	.35	6.65	Medium gravel with considerable clay, much brush on top soil.
5	3	5	1		6,798	1,148,480	25.5	.88	.52	.05	1.77	.25	.35	3.80	Loose gravel, heavy overburden of sandy loam.
5	2	5	1		6,790	1,148,802	29.9	.82	.49	.03	1.89	.25	.16	3.64	Do.
5	4	7	1		6,644	599,614	38.5	1.77	.92	.25	[e] 4.03	.47	.23	7.67	Difficult digging, working against 20-foot bank.
7		9½	1		5,088	838,885	35.0	1.19	.69	....	1.22	.26	.17	3.53	Difficult digging, gravel coarse, partly cemented.[f]
7	1		1		6,313	1,114,605	27.6	1.21	.62	.03	1.81	.29	.11	4.07	Compact gravel.
7	3	9	1		6,390	1,033,694	26.5	1.08	.64	.14	[g] 2.69	.34	.20	5.09	Compact gravel, heavy digging.
7	2	9	1		6,917	1,017,167	28.1	1.10	.65	.15	[h] 2.19	.28	.14	4.51	Do.
7	3		1		6,352	935,322	33.4	1.26	.85	.06	3.06	.31	.34	5.88	Compact gravel.
7	3		1		6,700	1,194,146	27.5	1.05	.58	....	2.78	.32	.37	5.10	Do.
7.5	2	11	2	11	13,464	3,458,229	27.9	.27	.41	....	1.50	.24	.24	[i] 4.42	Medium compact bench gravel.
7.5		9½		9½	5,582	944,879	28.9	.95	.58	....	1.30	.27	.39	3.55	Medium compact gravel with heavy overburden.
7.5	2	6	1		6,402	1,369,844	70.2	.99	.77	....	1.95	.45	....	4.16	Medium gravel overlain with hydraulic tailings.
7.5	2	6	1		6,900	1,281,351	67.8	1.09	.98	....	2.01	.45	....	4.53	Do.
8		6		6	3,162	583,927	42.5	1.69	.59	....	1.14	.28	.22	3.92	Light gravel, dredge working against 10-foot bank.
8		4½		4½	2,369	626,624	24.0	(j)	(j)	(j)	(j)	(j)	(j)	2.47	
9		5		5	....	580,310	51.0	(j)	(j)	(j)	(j)	(j)	(j)	4.98	Cemented gravel, difficult digging; 20-foot bank above water level.
13.5		8		8	4,478	1,803,201	19.0	1.02	.47	....	.60	.12	.09	2.30	Fine gravel, easy digging.

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[a] Janin, Charles, Present-day problems in California gold dredging: Trans. Am. Inst. Min. Eng., vol. 142, 1911, p. 873.

[b] Total possible time in year's work, 8,784 hours.

[c] A 7-foot dredge is now working this ground at a profit.

[d] Including general expense, management, etc.

[e] Heavy repair cost due to new tumbler, conveyor belt, repairs to digging ladder, screens, etc.

[f] This dredge successfully replaced an open-connected bucket dredge which could not handle ground at a profit.

[g] Replacing tumbler shafts, conveyor belt, and new screen included in repairs.

[h] New steel spud and screen in repairs.

[i] Depreciation charges included in total expense.

[j] Segregated costs not given.

Operating costs of dredges of Tonopah Placers Co.

Item.	Total cost.	Cost per cubic yard.
DREDGE No. 1.		
Labor	$18,296.69	$0.019
Supplies	8,445.09	.009
Power	14,262.36	.016
Total direct charges	41,004.14	.044
Superintendence	3,002.42	.003
Office	1,198.12	.001
Insurance	842.92	.001
Taxes	3,696.62	.004
General	670.02	.001
Company cottages	1,021.87	.001
Stable	115.39	......
Auto	557.21	.001
Total indirect charges	11,104.57	.012
Total	52,108.71	.056
Yardage, cubic yards	919,397	......
Cost per cubic yard	.056	......
DREDGE No. 2.		
Labor	17,218.08	.013
Supplies	9,782.57	.007
Power	17,501.72	.013
Total direct charges	44,502.37	.033
Superintendence	3,244.48	.002
Office	1,352.21	.001
Insurance	842.42	.001
Taxes	8,781.70	.006
General	615.99	.001
Company cottages	394.47	......
Stable	115.39	......
Auto	557.21	......
Total indirect charges	15,903.87	.012
Total	60,406.24	.045
Yardage, cubic yards	1,332,739	......
Cost per cubic yard	.045	......
DREDGE No. 3.		
Labor	20,838.80	.028
Supplies	16,630.97	.022
Power	17,925.62	.024
Total direct charges	55,395.39	.074
Superintendence	3,799.98	.005
Office	1,570.22	.002
Insurance	965.57	.001
Taxes	3,491.68	.005
General	619.64	.001

Legal	$138.85	......
Drilling	294.49	$0.001
Auto	557.27	.001
Stable	92.50	......
Total indirect charges	11,530.20	.016
Total	66,925.59	.090
Yardage, cubic yards	743,120	......
Cost per cubic yard	.090	......
Total cost all dredges	179,440.54	......
Total yardage	2,995,256	......
Average cost per cubic yard	0.059	......

Operating costs and miscellaneous data, dredges of Marysville Gold Dredging Co.

DREDGE NO. 3.

MISCELLANEOUS DATA.

Item.	1910	1911	1912	1913	1914
Working days	362	364	364	363	3
Working hours	8,688	8,736	8,736	8,712	8,7
Hours digging	6,106.39	6,050.59	6,493.00	7,322.34	7,373.
Time digging, per cent	70	69	74	84	
Area in acres	18.114	16.124	15.1373	15.7166	17.27
Average depth, feet	60	62	66	65	
Cubic yards handled	1,728,302	1,532,784	1,515,443	1,624,055	1,831,8
Cubic yards per day	4,774	4,211	4,163	4,474	5,0

COSTS PER CUBIC YARD.

Labor	$0.0182	$0.0161	$0.0166	$0.0160	$0.01
Power	.0070	.0074	.0088	.0085	.00
Supplies	.0235	.0382	.0330	.0282	.02
Taxes and insurance	.0021	.0047	.0030	.0031	.00
Miscellaneous	.0026	.0025	.0057	.0051	.00
Total	.0534	.0689	.0671	.0609	.05

DREDGE NO. 4.

MISCELLANEOUS DATA.

Item.	1911	1912	1913	1914
Working days	239	364	363	3
Working hours	5,736	8,736	8,712	8,7
Hours digging	4,428.06	7,075.10	6,901.08	7,415.
Time digging, per cent	77	81	79	
Area in acres	11.6645	15.5522	19.7450	22.55
Average depth, feet	59	61	58	
Cubic yards handled	1,058,797	1,453,428	1,808,001	2,112,6
Cubic yards per day	4,430	4,000	4,980	5,8

COSTS PER CUBIC YARD.

Labor	$0.0156	$0.0176	$0.0142	$0.01
Power	.0081	.0102	.0083	.00
Supplies	.0191	.0318	.0191	.01
Taxes and insurance	.0041	.0025	.0030	.00
Miscellaneous	.0023	.0069	.0045	.00
Total	.0492	.0690	.0491	.04

Operating costs of dredges of Oro Electric Corporation.

OROVILLE DREDGES.

Year.	Dredges operating.	Cubic yards dredged.	Days elapsed.	Average hours running time per day.	Average cubic yards per month per dredge.	Average cubic yards per day per dredge.	Average cubic yards per hour per dredge.	Average cost per month per dredge.	Cost per hour of running time.	Cost per cubic yard, in cents.				
										Maintenance.	Operating.	General.	Insurance and taxes.	Total.
1903.	0.25	196,479	214	5.39	19,560	652	120.9	$836	$5.17	......	......	......	....	4.27
1904.	1.83	1,008,090	824	10.20	32,850	1,095	107.4	2,365	7.73	......	......	......	....	7.20
1905.	3.00	2,210,609	1,095	16.74	60,700	2,020	120.7	3,942	7.85	......	......	......	....	6.50
1906.	3.67	3,131,020	1,309	19.10	69,060	2,302	120.5	3,360	5.86	......	......	......	....	4.78
1907.	4.75	3,755,115	1,582	17.02	69,300	2,310	135.7	3,670	7.19	1.72	2.95	0.54	0.086	5.31
1908.	4.58	4,572,573	1,830	16.04	74,880	2,496	155.6	4,192	8.71	2.24	2.46	0.43	0.476	5.61
1909.	5.00	4,004,352	1,825	16.53	66,060	2,202	133.2	4,043	8.15	2.69	2.64	0.44	0.361	6.13
1910.	5.00	4,308,476	1,825	18.08	70,890	2,363	130.6	3,539	6.52	1.65	2.65	0.44	0.263	5.00
1911.	4.58	3,646,695	1,641	18.74	65,520	2,184	116.6	3,644	6.48	1.79	2.95	0.44	0.374	5.56
1912.	4.00	2,225,948	1,464	17.63	46,374	1,520	86.2	3,687	6.85	2.66	4.25	0.43	0.606	7.95
1913[a]	4.00	3,465,711	1,460	18.71	72,202	2,374	126.9	3,745	6.57	1.58	2.92	0.34	0.347	5.19
1914.	3.76	3,303,647	1,374	19.08	73,218	2,404	126.0	3,057	5.26	0.80	2.73	0.30	0.352	4.18

CAMANCHE DREDGE (9 cubic feet).

1913.	0.88	1,812,533	322	20.02	170,351	5,629	281.1	5,550	9.16	1.02	1.79	0.31	0.140	3.26
1914.	1.00	1,704,044	365	19.38	142,004	4,669	239.4	5,668	9.56	1.23	2.16	0.31	0.289	3.99

[a] March, 1913, one dredge was converted from 5 to 6 cubic feet capacity.

RECOVERY OF GOLD COMPARED TO CONTENT INDICATED PROSPECTING.

What percentage of the gold is recovered by a dredge, whether t actual content of the material dredged or the yield indicated by pr pecting is considered, has always been an interesting question. It generally acknowledged that the modern dredge with its improv washing plant and large area of gold-saving tables recovers, exce where considerable sticky clay or where "rusty" gold is present high proportion of the gold in the material washed. The proporti of gold recovered from the amount estimated to be in the ground i different matter, and requires the consideration of several factors n directly connected with the metallurgy of gold dredging. Obvious to determine the difference between the actual gold content of t ground and the recovery on the dredge, it is necessary to kno (1) the gold content of the ground previous to dredging, or (2) t loss in the tailing.

Too much dependence on a vendor's statement of past producti or his estimate for the future accounts for many a wide discrepan between expectation and fulfillment, and the same may be said careless or inexperienced sampling or, sometimes, no sampling wh ever. In the early days of dredging, inexperience in the use of dr for sampling placer ground often led to over estimation of the g content; this error accounts in part for the first dredges recoverin low proportion of the gold indicated by drill sampling. Experien has minimized this danger, and the results obtained over a period years have shown that there is little reason to-day for a wide discr ancy between the actual recovery and the content estimated by pr pecting, provided the prospecting was done by an experienced engin with sufficient funds and time at his disposal.

The thorough sampling of a placer, as described in the chapter prospecting, is by no means the simple matter the uninitiated oft believe it to be. Much depends on the care taken and the experien of the engineer in charge.

If great care has been exercised in sampling, every detail being do properly, each sample will represent as nearly as possible the g content per cubic yard at the place sampled. Although each h may be taken as indicative of the content of the ground where it sunk, the proper placing of the holes is of importance. The prop location of holes and the interval at which samples should be tak can be determined only after a study of the ground. There is hard and fast rule to follow. Some areas have been considered sufficiently prospected with one drill hole to 10 acres; on other areas the engineer has not been fully satisfied with one hole to one acre.

METHODS OF PROSPECTING.

Preliminary holes are generally sunk and carefully sampled to determine whether the gold occurs in wide or in narrow channels, and whether its distribution is fairly even or in irregular pay streaks. Holes badly spaced with too many in the richer part of the ground would yield too high an average content. Ground that shows a spotted or irregular gold content will obviously need more tests than ground yielding fairly uniform samples. Where sampling discloses a channel or narrow belt having a higher mineral content than that of the whole area, the outlines of the channel or belt should be determined as closely as the purpose of the examination warrants. Such sampling is needed at French Gulch near Breckenridge, Colo. where the principal gold content is in a narrow channel, and ground outside the limits of the channel is too poor for profitable dredging.

Sometimes it is feasible to segregate certain parts of a property and eliminate low-grade areas. When a tract is large and the time for examination is limited, tests to confirm the samples indicated by preliminary prospecting may be concentrated on a piece of ground large enough to determine the advisability of purchasing the property and installing at least one dredge. From these statements it is clear that the accuracy with which the average mineral content of placer ground is indicated by an examination depends on the care taken in sampling, the number of samples taken, and the situation of the holes with reference to the character of the deposit. Everything hinges on the experience and judgment of the man in charge.

FACTORS USED IN CALCULATIONS.

In estimating the probable recovery from dredging ground most engineers, after determining the average content indicated by prospecting, take a certain proportion, generally 75 or 80 per cent, of this indicated content as recoverable by the dredge. These figures are based on actual tests made to determine the dredge recovery, and also on the percentages of the prospecting content recovered in dredging certain carefully prospected areas.

Some engineers, when sampling with drills, use a different factor in their calculations than that generally employed; that is, they reduce the metal content per cubic yard of the place sampled by an amount that, in accord with their experience, represents recovery. Use of this factor, practically a combination of drill factor and recovery estimates, discounts the content that otherwise would have been assumed from the regular drill factor. For instance, in place of the usual factor 27, a new factor, 33⅓ per cent, is used. Engineers following such a practice should mention it clearly in their reports. Failure to do so might cause the future of a property to be discounted further by persons to whom the report is presented, if these persons are not conversant with the methods employed in computation.

Attempts to confirm the gold indications of individual drill holes by sinking shafts around the drill holes, or sinking other drill holes near the first, generally yield widely different results. The gold is not uniformly distributed throughout the gravel, so that the results from single drill holes do not indicate of average content. There are rare instances on record of the dredge recovery exceeding estimates, not only for the drill showing on certain small areas between clean-up periods, but also for the total area dredged. Thus in a small area dredged by Natoma No. 4, 120 per cent of the yardage, and 115 per cent of the estimated recovery were obtained. In most discrepancies of the kind it will be found that ordinary methods of valuation were discounted in preparing the estimates.

The following reference to the Pato property, taken from the report of the Oroville Dredging Co., dated September 30, 1911, shows the prevailing practice in discounting the prospecting indications. "The original estimates by the engineers give us an area containing 13,637,347 cubic yards of material, yielding gross $3,202,986, there having been deducted from the values recovered by protecting tests 25 per cent for variation, possible losses, etc."

RECOVERY FROM DIFFERENT PARTS OF TABLE.

Figure 22, representing the gold-saving table of a 6-foot close-connected bucket dredge, furnished by W. H. James of Oroville, shows the percentage of the total gold recovered from different parts of the table. It is based on careful data kept at each clean-up for a period of three years.

The total table area is 1,457 square feet; 89 per cent of the gold recovered is arrested on 292 square feet, as shown on those portions of the table marked 1. Including the recovery from the save-all, 92.2 per cent of the gold was obtained from 332 square feet; 4.68 per cent more was recovered from 360 square feet additional. Thus 96.9 per cent of the total gold recovered on this dredge came from 692 square feet of table area. The remaining 3.1 per cent was obtained on the additional 765 feet. Although the total table area on this dredge was 1,457 square feet, that on a dredge having the same bucket capacity, but designed for ground containing much clay and difficult to wash, was increased to 3,800 square feet.

Table area varies with the ideas of designers and with the conditions to be encountered. From the first dredges with only a few square feet of table area to the modern 15-foot dredge with 8,000 square feet is a long step, as is also the increase of capacity from 10 cubic yards or less per hour of actual mining time for early dredges to 500 cubic yards per hour for months at a stretch for some modern boats.

Figure 22 is interesting as showing how large a proportion of the gold is recovered on the upper part of the table, the bulk of all the gold recovered being actually taken from the first 3 feet at the top end. It is advisable, however, to have as large a table area as space will permit, even though it is not often necessary to clean up the lower end. Tests with the Neill jig, described in the chapter on metallurgy, indicate the advisability, on some boats, of supplementing

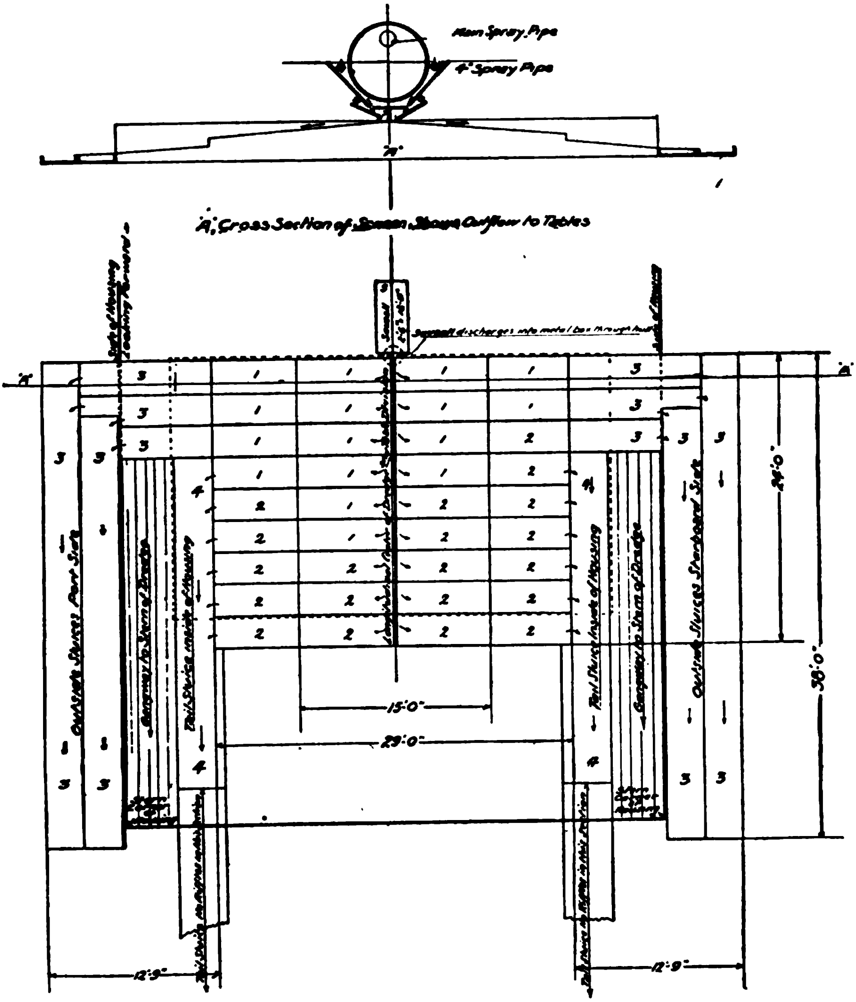
Main Spray Pipe
4" Spray Pipe
"A"
"A" Cross Section of Screen Shows Outflow to Tables
Screen discharges into metal box through hull
"A"
"A"
Outside Sluices Port Side
Gangway to Stern of Dredge
Tail Sluice inside of Housing
Tail Sluice inside of Housing
Gangway to Stern of Dredge
Outside Sluices Starboard Side
1
2
3
4
15'0"
29'0"
24'0"
38'0"
12'9"
12'9"

FIGURE 22.—Table plan of gold-saving area on dredge equipped with revolving screen, showing distribution of flow and recovery from segregated areas.

the gold-saving tables with other devices. Few operators have ev complained of having too large a table area on their dredge, an many would add to the existing tables if space permitted. Invest gation of the tables of a dredge will generally show some loss quicksilver, and consequently of gold, over the tail sluice in additio to the fine gold carried off in suspension.

From the amalgam returns, the percentages recovered from th different gold-saving appliances on the Conrey dredges at Rub Mont., from the start of their work to December 31, 1913, are under:

Proportion of total recovery from gold-saving devices on Conrey dredges.

	Per cen
Dredge No. 1.	
All 20 tables	92. 5
Tail sluices	3. 3
Well save-all	4. 1
	100. 0
Dredge No. 2.	
All 20 tables	93. 4
Tail sluices	3. 4
Well save-all	3. 0
	100. 0
Dredge No. 3.	
Upper flume	55. 6
Lower flume, first section	27. 1
Lower flume, second section	6. 7
Lower flume, below undercurrent	. 4
Undercurrent	1. 3
Well save-all	5. 0
Tables	3. 0
Tail sluices	. 3
Miscellaneous	. 3
	100. 0
Dredge No. 4.	
First 16 tables	87. 9
Last 8 tables	1. 9
Tail sluices	6. 8
Undercurrent	. 5
Well save-all	2. 7
	100. 0

GOLD IN TAILINGS.

To sample the dredge tailing is difficult. To be trustworthy, an test must extend over a considerable period. Practically all suc tests have indicated an appreciable loss, but most of the tests di not, so far as the author knows, determine the gold recovery wit any accuracy. Even the extensive tests on Natoma No. 7 dredg prior to installing the Neill jig, although they demonstrated a hig loss of gold, did not establish this with any exactness. Most statements of exact percentages of loss of gold have proved, upon investigation, to be merely guesses made by the dredge master. In connection with one of the most thorough tests made for this purpose, at Oroville, a small bucket-elevator was used. The tailing examined was taken by means of perforations in the bottom of the tailing sluices; it passed to a small wooden sump, in which the lower sprocket wheel of the elevator was mounted. The material was somewhat concentrated, because of the method of collection, but this could not be avoided. The speed of the elevator was arranged so that approximately 2 cubic yards of material would be collected during the shift. This collected tailing afterward was carefully washed through a rocker, and panned. All the tests showed an appreciable loss of quicksilver and amalgam, but no definite figure of loss, compared with the total gold content of the material handled, was obtained.

In general, it may be said that the loss of gold in the tailing is less than the amount lost in front of the boat, both from bucket spill not caught in the hopper and save-all, and from failure properly to clean bedrock. It is estimated by those competent to judge that the average recovery of the actual gold content, while varying with different operating conditions and efficiency of labor, is not less than 90 per cent under normal conditions, and sometimes it may be considerably more. Compare this with the following extract from the Canadian Mining Review of January 30, 1903, in reference to the old Cobeldick dredge at Lytton:

> The director who was sent out from England to investigate for the company the working of the dredge made the discovery that of the gold dredged up from the bottom less than 10 per cent was recovered on the tables, the remaining 90 per cent going off again with the tailing, although the saving appliances on this machine were about the most complete of any in British Columbia.

It would be interesting to learn what tests the director made to arrive at this conclusion. Probably the recovery was only 10 per cent of the estimated amount, and the blame was put on the gold-saving appliances instead of where it rightly belonged. The question was, perhaps, not so much one of poor recovery as of an overestimate of gold content.

EXAMPLES OF RECOVERIES BY DIFFERENT DREDGES.

In a careful test to determine dredge recovery, 22 holes were sunk on a measured area. Results were from 3 to 52 cents a cubic yard, and averaged 19 cents; the regular drilling factor was used in the calculations without any discount for recovery. In working, the recovery was, as nearly as the author can remember, 17 cents a cubic yard, or about 90 per cent of the content shown by averaging the samples obtained in drilling. In another test 17 holes were put down in front of a dredge, the content of the area closely calculated according to drill methods then employed, and the total content discounted 20 per cent. The dredge recovered 99 per cent of the total amount estimated to be in the ground. Those companies in California that get the highest recovery of the estimated content are those that discounted the prospecting indications by 20 or 25 per cent. To have much value, the recovery percentage based on prospecting must be founded on bank measurement of the yardage handled—not on the size of the dredge bucket and the running time of the dredge.

The following results, obtained by a California company working one dredge only, are of considerable interest, because the company kept careful records from the beginning.

Results obtained by a California company.

Acres dredged		144
Cubic yards dredged		6,980,800
Gold content per cubic yard as shown by drill holes, before discounting	cents	21.05
Actual recovery	do.	12.92
Recovery of amount indicated by drill holes	per cent	61.38
Drill holes to area (approximately 1 hole to 3 acres)		47

RESULTS FROM A SMALL AREA.

Acres dredged	40
Cubic yards dredged	1,232,868
Prospecting content (cents per cubic yard)	17.18
Recovery (cents per cubic yard)	15.35
Percentage of recovery	89.34

Six holes to 40 acres.

The results obtained by the dredges of the Natomas Consolidated and the Yuba Consolidated are of especial interest on account of the magnitude of the operations. The dredges of the Natomas Consolidated from the beginning of operations to December 31, 1915, handled approximately 150,000,000 cubic yards. The following excerpt, in regard to the recovery, is taken from C. M. Rolker's report on the Natomas Consolidated in 1909:

> The writer wishes to emphasize that according to the statement of the general manager, Newton Cleaveland, the values of the different bore tests, which are marked on the company's maps, are the original values calculated, reduced by 15 per cent to make them representative of recovery.

In 1913, Frank Griffin, the president of the Natomas Company, stated that the recovery made by the dredges was 15 per cent less than the engineers estimated. This would be about 72 per cent of the yield indicated by the drill holes previous to the 15 per cent reduction, mentioned in Mr. Rolker's report. It should be noted, however, that all of the ground has not been dredged, and it is only when the property is worked out that the actual recovery of the gold assumed to be in the ground will be ascertained.

The dredges of the Yuba Consolidated had handled over 135,000,000 cubic yards to February, 1915. The actual recovery as compared with that estimated is not yet ascertainable, but, according to a large shareholder, it is about 75 per cent of that indicated by prospecting. The percentage of recovery is difficult to determine as a dredge will at times work for months without encountering a drill hole and until the property is finally worked out, no figures for recovery as compared to first estimates, can logically be made. Final figures will be interesting, as the company has never been advertised, the stock is privately held, and the profits are said to exceed those of every other gold-dredging company.

The California dredges of the Oroville Dredging (Ltd.) have handled over 50,000,000 yards to January, 1916. At the time when the property was floated in London there were estimated to be 38,500,000 yards with an average content of about 17.2 cents, and the Bear River property was stated to contain an average of 12 cents. The prospecting results as indicated by drill holes on some of this ground had been discounted previous to the consolidation, but seeming not on the larger part of the property. The average recovery from dredging 44,800,000 cubic yards, exclusive of Bear River gravels, was 11.17 cents per yard, indicating a recovery of approximately 65 per cent of what was expected.

The Bear River area was even more disappointing. The average from 1,832,585 cubic yards during three years was 5.26 cents, or about 44 per cent of the estimated content. The recovery made by the Pato dredge, operating on the holdings of this company in Colombia, have been the subject of considerable comment, as the yield has been at times far in excess of the average gold content as estimated for the entire area. It is clearly understood by those with experience in such matters that the recovery made by a dredge over short periods often varies greatly. Discrepancies of recovery over such periods compared with the average gold content assumed for the whole area are no more significant than high gold content in a few drill holes as compared with the average of the property, other than perhaps to indicate a richer spot or "channel" in the area. In other words, because the recovery from a property for a short period in the beginning of operations is higher than the average for such property, there is no reason to expect the whole area to exceed the assumed average.

The results from Pato show that during two seasons 2,500,000 cubic yards, yielding an average of 43.8 cents, has been dug in an area of 92 acres, which were estimated to contain a total of 3,780,000 cubic yards having an average content of 40.4 cents. Although the recovery of the dredge per cubic yard for the first 750,000 yards from the "tested" area was 54 per cent more than the estimated content as indicated by the drill for the whole 92 acres, and occa sioned considerable comment at the time, it meant only that th dredge was operating in the richer parts of the "tested area." N comparison of recovery as compared with average prospectin content could rightly be made on a small section of a large tract.

It has been stated [a] by the Colorado Gold Dredging Co., operatin in Colorado that the recovery from its dredge under the best con ditions averaged about 80 per cent of the gold previously estimate to be in the gravel. Other Colorado companies, however, did no succeed in obtaining as high a recovery. In fact, a number of th earlier dredging ventures in Colorado proved unprofitable, thoug doubtless the gold content assumed to be in the ground was suffic ent to justify the venture.

Although most of the figures mentioned have been based o prospecting done under the direction of careful engineers, there a numerous instances, particularly of early drillings, where the recover made by the dredge has been 40 per cent or less of the assume average gold content of the ground. The dredges of one Californ company handled 8,156,571 cubic yards and obtained a recover of 40.6 per cent of the gold indicated by prospecting.

FAULTY PROSPECTING A CAUSE OF SEEMINGLY POOR RECOVER

At times a poor recovery is made by a dredge on account of carele or inexperienced work during the examination. Although bo holes indicated 5 to 13 grains per cubic yard, one of the dredgin companies in West Africa never recovered more than 2 grains p yard. Ransome [b] mentions that methods employed in drilling an calculating gold content are not always uniform and cites an instanc showing the comparative results of two methods used by the Colo rado Gold Dredging Co. to estimate the value of ground tested b drill holes. By one method of computing, using the diameter of th drilling shoe, a content of 4.6 cents a yard was indicated, and by th other, figuring the volume of the material pumped from the sam hole, a gold content of 25.4 cents a yard was indicated.

Mention may also be made of a property near Nome in Alask where the value of the gold recovered was less than 20 cents p yard from gravel that was claimed would average more than $1. may be said, however, that this instance represented not so much poor recovery as fraudulent representation. What was neede here, and would be valuable elsewhere under like conditions, wa not so much a factor representing dredge recovery as one to discoun vendor's statements.

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[a] Bradford, A. H., and Curtis, R. P., Dredging at Breckenridge, Colo.: Min. and Sci. Press, vol. Sept. 11, 1909, pp. 361-366.

[b] Ransome, F. L., Geology and ore deposits of the Breckenridge district, Colorado: Prof. Paper U. S. Geol. Survey, 1911, p. 181.

RECOVERIES FROM REDREDGED GROUND.

Some ground that was redredged in California in moving boats to new positions returned 1 to 3 cents a yard—not sufficient to pay operating expenses, but from one area a dredge recovered gold to the value of 10 cents a yard when working through tailing. An investigation resulted, and a considerable part of the tailing was redredged at a profit. Imperfect washing of the material caused by the poor arrangement of the washing plant and the gold-saving tables, which were replaced with more modern equipment, was the cause of the poor recovery, and perhaps the desire to make yardage records with the earlier dredge was also an essential factor. Elsewhere, from gravel containing much clay or cemented gravel, the recovery of gold may be low and the tailing show sufficient gold to make redredging practical. In some parts of the Oroville field clay dredged six years ago and since exposed to the weather has not yet disintegrated.

In northern California the recovery recently made by a dredge was 15 cents a yard when working through the tailing from a former dredge. The reason was that the earlier dredge had not reached bedrock. The first dredge did not prove a financial success but the second has been a profitable venture from the beginning.

At Jenny Lind, Calaveras County, as high as 7 cents a yard has been recovered in redredging ground. On the other hand, Dredge No. 3, Feather River, of the Natomas Consolidated Co., dug through 500,000 cubic yards of tailing from the old Cherokee track, also digging about 100,000 yards of virgin ground during the same run. The recovery from the 600,000 yards handled averaged only 1½ cents a yard, and the greater part of this recovery must have been from the new ground, indicating that the tailing was practically barren.

The Pennsylvania dredge at Oroville in moving to a new location cut through over half a mile of tailing and recovered about $450, no attempt being made to dig deeper than necessary for flotation. Other dredges in the Oroville district in working through tailings from former operations have recovered from 1 cent to as high as 10 cents a cubic yard over short periods, the high recovery representing that of a dredge digging near bedrock ridges that had been left between cuts in former dredging.

A washing plant was built by the Natomas Consolidated to handle the fine material screened from the rock sent to the rock-crushing plant. All material smaller than ⅛ inch was sent to this plant, which had riffled sluices and tables. The recovery when the plant was first tried indicated about 3⅓ cents a yard for the material sent to the tables and about ⅓ cent for the total material handled. The upper 2 feet of fine material left after the coarser rock is taken to the rock-crushing plant has given rather high results at times, and it may be possible to redredge some material at a profit after the rock has been removed and by reclaimimg the land behind the dredge add to the profit of the operator.

At Ruby, Mont., although the entire property has never been systematically prospected as a whole according to modern methods, small areas have been more or less closely prospected by drill holes and afterwards worked. The percentage of recovery by the dredges has generally averaged about 80 per cent of the probable recovery indicated by prospecting, the ordinary drill factor being used for valuing the gold content of each hole. The percentage of the actual gold content recovered by the dredge is of course a different matter. Drill holes in the tailing of the present dredge have shown little gold and when it was necessary to dig through any of this tailing in moving the dredge to a new position the recovery has been about 0.54 cent a cubic yard in a total of approximately 125,000 yards thus reworked, indicating a close recovery of the gold content on the first dredging where the ground is easily washed. In May, 1915, the No. 3 dredge dug through a quantity of tailings from the old steam dredges and made a recovery of 2.05 to 3.73 cents a cubic yard, demonstrating that the earlier dredges were not as efficient gold savers as the modern ones. The figures indicate a loss with the formerly used flume dredge varying from 10 to perhaps 20 per cent of the actual content.

RECOVERY OF GOLD AT OROVILLE, CAL.

The following table prepared by one of the most careful California operators is of considerable interest, and is based upon the following data: In dredging the property, a clean-up was made at the end of 15 days' dredging. It was found frequently that there had been a prospect drill hole on the area dredged during a 15-day period. If the hole was near the center of the area the recovery was compared with the content indicated by the drill hole. If the hole was reached at or near the end of the 15-day period, an average of two such periods would be used as the recovery figure for comparison with the content indicated by prospecting.

Comparative drill-sheet values and actual dredging results at Oroville, Cal.[a]

[Arranged in order of highest drill-sheet results and reduced to percentages of recovery.]

FIGURES OF ENGINEER "A".

Designation of drill hole.	Drill-sheet value.	Actual dredge value.	Per cent recovered.
	Cents per cubic yard.	*Cents per cubic yard.*	
F	59.80	20.72	34.60
G	48.51	26.50	54.60
C	42.30	19.56	46.30
E	33.53	13.36	39.80
U	32.79	15.56	47.50
I	26.20	4.60	17.50
L	23.81	14.83	62.30
A	21.35	9.25	43.30
N	21.23	10.37	48.80
V	20.80	14.00	67.30
M	20.25	11.17	55.16
K	18.52	14.80	80.00
J	15.47	7.97	51.50
A_2	14.30	13.28	92.90
D	11.14	17.00	152.60
B_2	10.76	5.77	53.60
T	8.31	13.82	166.30
B	7.53	10.40	138.00
Y	6.02	19.35	321.00
Z	2.58	15.50	600.00
X	1.85	7.25	390.00

FIGURES OF ENGINEER "B."

Designation of drill hole.	Drill-sheet value.	Actual dredge value.	Per cent recovered.
6	77.18	23.58	30.00
2	41.28	14.00	35.00
14	29.75	10.05	33.78
5	26.14	12.00	46.00
4	25.76	10.25	40.00
3	22.12	12.93	59.00
7	20.61	11.50	51.00
8	6.24	5.24	84.00
16	4.47	4.60	103.00

FIGURES OF ENGINEER "C."

Designation of drill hole.	Drill-sheet value.	Actual dredge value.	Per cent recovered.
F [b]	60.40	20.72	34.30
6 [b]	54.54	23.58	43.20
W	36.78	28.00	76.10
15	34.22	16.86	49.00
h	33.53	30.00	90.00
a	27.72	15.24	54.10
C [b]	27.10	19.56	72.00
j	22.24	17.80	80.00
b	21.15	8.30	39.20
i	18.88	15.44	82.39
d	18.37	20.60	112.20
5 [b]	17.83	12.00	67.30
e	17.49	19.16	109.50

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[a] Janin, Charles, Recovery of gold in dredging: Min. and Sci. Press, vol. 109, 1914, p. 717.

[b] Check hole to verify results of hole previously drilled having similar designation. (See above.)

The data show that in the field worked by this dredge at least the record of a single drill hole can not be regarded as representative of the actual gold content of the surrounding area. This has been the experience of other operators. In one field, where the drill hole indicated 2.58 cents a yard, as shown on the table, the dredge recovered 15.5 cents, or 600 per cent, of the content given by the drill hole; in another field, from ground where the drill showed a content of 59.8

cents, the dredge recovered 20.7 cents, or 34.6 per cent; and in a third, where 26.2 cents was indicated, only 4.6 cents, or 17.5 per cent, was recovered.

Any estimate to be at all trustworthy can be made only from the average of the content indicated from a number of drill holes that have been intelligently placed and carefully sampled. If the work of prospecting has been done efficiently, an estimate of the gold content of a property can be made with some degree of accuracy, but if the work has been carelessly performed by unreliable or inexperienced engineers, the results obtained are so worthless as to render any refinement or calculation merely ridiculous.

RECOVERY AT SNELLING, CAL.

The following example of recovery as compared to prospecting is extracted from a letter of J. W. Neill: [a]

> The tract in question, belonging to the Yosemite and Mining Co. at Snelling and comprising 30 acres, lay on the opposite side of the river from the general mass of the ground and was left until the last for this reason. It was of importance to know whether this area was profitable, as, if so, we would cross the river, and dismantle the boat close to the county road, thus saving much distance in hauling. Holes were drilled as per the first map, 400 feet apart up and down the stream, by 200 feet across the flow, making 20 holes in all. The map shows the depth and the value as shown by the samples. The area was in part badly overgrown with brush, and in part in a slough, so we had to drill a few holes rather out of alignment. The top soil, consisting of sandy loam, averaged about 6 feet deep; below this was gravel consisting of small cobbles and sand. The bedrock was soft lava ash.
>
> The average of these holes, figured in the regular way, is 5.29 cents per yard. We use the "0.01 formula" in our estimates, and for the probable dredge yield deduct from this 20 per cent for losses. This would make the probable value of the area only 4.32 cents. Moreover, in this area there are three high holes—25 cents, 14 cents, and 20 cents. If we cut these in half, as being freaks, we get a value for the area of 3.63 cents and 2.90 cents dredge yield.
>
> To see if there was not a possible area of sufficient value to warrant dredging a second series of holes was put down, as in map No. 2, and for this prospecting an average of 4 cents per yard, drill yield, or 3.3 cents dredge yield, was obtained, including two freaks of 35 cents and 13 cents. By cutting these in two a drill value of only 3.27 cents and a dredge value of 2.61 cents is indicated. These two samplings, therefore, checked within a small fraction of a cent against each other, and the total average of both, which is for 45 holes in 30 acres, 4.57 cents, including all holes, or 3.42 cents if half the freak holes be deducted. The comparison would therefore be: First 20 holes, 3.63 cents; second 25, 3.27 cents; of all 45, 3.42 cents. After much discussion we selected a piece of the ground, crossed the river, and dredged, as shown in map No. 3, seven acres. This dredged area covered 14 drilling holes whose average value was 7.75 cents, or if we deduct as before the half of three high ones—20 cents, 33 cents, and 13 cents—we get 5.55 cents as the drill value, or an expected dredge yield of 4.44 cents. The records of our bullion product show for these three months an average yield of 6.74 cents per yard. In other words, we saved by the dredge a little more than 80 per cent of the drill returns of all holes, including the freaks.

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[a] Neill, J. W., Letter relative to methods of testing placer gravels: Min. and Sci. Press, vol. 109, Aug. 8, 1914, pp. 221-222

POWER PLANTS FOR DREDGE INSTALLATIONS IN ISOLATED DISTRICTS.

In the construction and operation of dredges in out-of-the-way places the power factor is one of the highly important problems. Where electricity is available it is the most nearly ideal power for dredge operation, but often electricity is either not available or can be developed only at an excessive cost. For the operation of only one dredge, obviously great expense can not be undertaken in power lines and electric-power installation, but if a company is building a number of dredges or if several companies contemplate dredge operation in a new field a central power plant to generate electricity will as a rule prove economical. Should electric power be neither available nor feasible then some other form of power must be considered.

The first steam dredges were excessive consumers of fuel, and operating costs were high in comparison to those when electricity became available for use on the same dredge. An example of the reduction in the power cost in dredges is that shown on the operation at Ruby, Mont. With the cost of coal at $4.90 a ton the fuel costs of the two Conrey, 7½-foot and 10-foot, steam dredges during 1904 to 1906 was 3.51 and 3.07 cents a cubic yard. Although these were old-type and inefficient machines, the figures cited illustrate the high power cost under steam operation even with a low fuel cost. The cost of power and fuel for the present Ruby dredges averages less than 1½ cents a cubic yard.

Practically all dredges now operating in the United States use electricity. In Alaska where the single-dredge installation is the rule and on the Seward peninsula where upward of 100 dredges have been built, 40 of which were either operating or under construction in 1913, only two used electricity. In this field and in the Philippines the American dredge constructor has had the opportunity to exercise his ingenuity and here most experiments have been made and the greatest development has taken place in the power arrangement of the self-contained dredge. In 1912, the writer[a] published an article on dredging in the Seward Peninsula which included a table giving the fuel used and the quantity consumed on most of the dredges in that field. Following is an abstract:

Fuel consumption on dredges in the Seward Peninsula, 1912.

Name of dredge.	Size of buckets.	Average daily capacity.	Horsepower of dredge.	Fuel used each day.	Quantity.	Cost of fuel per unit.	Approximate cost of fuel per cubic yard.	Operating conditions.
	Cubic feet.	*Cubic yards.*						
Solomon	3	1,800	160	Coal	7 tons	$27.00	10.5	Average.
Three Friends	5	2,000	230	Coal	12 to 17 tons	22.00	15.4	Difficult.
Wild Goose	3½	1,800	140	Distillate	300 gallons	0.25	4.2	Average.
Plein	3½	800	130	Crude oil	14 barrels	3.00	5.2	Some frozen ground.
Blue Goose	5	800	160	Wood	7 to 12 cords	12.75	12.7	Average.

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[a] Janin, Charles, Gold dredging on the Seward Peninsula: Min. and Sci. Press, vol. 105, Sept. 28, 1912, p. 396.

Since the table above was made, in 1912, there have been a number o changes made on these dredges, and two have shut down. The result indicated, although representative of work at the time specified, canno be considered as representative of the best present-day practice. I must also be noted that the fuel costs do not include labor, etc.

The use of gas engines demonstrated that a great saving could b made in the cost of fuel on dredges using coal, wood or crude o when transportation of such fuel from the seaport was necessary, an continued experimentation demonstrated that still further savin could be made by improved methods.

EXAMPLE OF FUEL SAVING BY REDESIGNING POWER PLANT.

An interesting example of reduction of fuel costs is furnished b the operation of the Three Friends or Seward dredge. This dredg was built in 1908, and for reasons satisfactory to those interested i the company, was equipped with slide-valve engines and locomotiv boilers. The coal consumption varied from 12 to 17 tons a day an the cost of coal delivered to the dredge was $22 a ton; with 14 ton at $22 as the average consumption, the fuel cost was $308 a da The average daily capacity of the dredge was 2,000 cubic yards, s that the fuel cost was 15.4 cents a cubic yard. In the latter part o 1912 oil burners were installed on the dredge and fuel oil costing abou $3 a barrel was used as fuel. This change, although reducing the fu cost somewhat, did not give the results the management anticipate some days as high as 60 barrels of oil being consumed. The write pointed out [a] the folly of trying to do anything with the antiquate equipment, but it was not until 1914 that any radical change wa attempted. The entire plant was redesigned. Semi-Diesel engine were used to generate electric power at a shore plant and electri motors were used on the dredge. The old machinery was consigne to the rock pile.

Figures of operating cost of the new plant, furnished by the management, are as follows:

Operating cost of Three Friends Dredge for 24 hours, 1914.

200 gallons of distillate, at $0.18 at power plant	$36.00
3 winchmen, at $7	21.00
3 oilers, at $5	15.00
3 engineers, at $8	24.00
2 roustabouts, at $5	10.00
1 dredgemaster, at $15	15.00
1 bookkeeper, at $7.50	7.50
1 cook	6.00
1 mess	22.00
Oil, lubricating	7.50
Repairs	15.00
Total	179.00

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[a] Janin, Charles, Gold dredging on the Seward Peninsula: Min. and Sci. Press, vol. 105, Sept. 28, 191 pp. 394-399; Gold dredging in California: Min. and Sci. Press, vol. 106, Jan. 4, 1913, pp. 65-69.

A point that was not at first realized in changing the power equipment on this dredge was the decrease in weight resulting from replacing the cumbersome inefficient steam plant with electric motors. The change permitted the dredge to work without digging so deeply into bedrock, thus permitting a greatly increased yardage. The yardage handled by the dredge has been variously estimated at 2,000 to 3,000 cubic yards a day, making an operating cost for fuel used of approximately 1½ to 2 cents a yard.

POWER COST ON TWO DREDGES ON SEWARD PENINSULA.

Two dredges recently constructed on the Seward Peninsula and equipped with entirely different power arrangements are of considerable interest in showing a great reduction in power as compared to dredges having self-contained power plants using similar fuel in the northern field, and the operation of these dredges should be carefully studied by anyone contemplating dredge operation. The Berry dredge was equipped with two 75-horsepower Wolf locomobile engines using wood for fuel. The writer has obtained the following figures from the operating report of Gerald Hutton, dredge superintendent for the company; they may therefore be considered authentic.

Data covering operation of Berry dredges on the Seward Peninsula, Alaska.

[Operating days, June 3 to October 17; closed down for winter and for thawing ground.]

Net operating time (131 days—100 per cent), hours	3,144	
Total lost time (25.7 per cent), hours	805	
Total digging time (74.3 per cent), hours	2,339	
Total depth dug, feet	11.9	
Total yardage handled, cubic yards	213,355	
Total yardage handled per day, cubic yards	1,628	
Total wood used, cords	461.35	
Total cost of wood used, 464.35 cords, at $10	$4,643.50	
Cost of getting to dredge	594.37	
Wastage, estimated	261.89	
		$5,499.75
Total cost of wood per cubic yard, cents		2.57
Total operating cost per cubic yard, cents		11.24

The lubricants used throughout the season were as follows:

Lubricants used by Berry dredge during operating season.

Lubricant.	Total quantity used.	Per day.	Cost for season.[a]
	Gallons.	*Gallons.*	
Hecla B. cylinder oil	84	0.64	$92.40
Atlantic red oil	235	1.8	141.00
	Pounds.	*Pounds.*	
Arctic cup grease	400	3.8	44.00
Dixon special grease	100		16.55
Mica gear grease	556	2¾	30.26
Total expense for the season			324.21
Total lubricating cost per day			2.47
Total cost of lubricating engines per day			1.78

[a] Including freight and hauling. The consumption of lubricating oil on the engines will be reduced the coming season, as the oil filter was not received until fairly late in the season under consideration.

The engines gave minimum trouble, but experience gained in their use showed the advisability of having on hand certain duplicate parts in order to obviate long delay from any breakage that might occur. The time lost by the digging engine was 4 hours 20 minutes, being lost principally through inspection of the different parts. The time lost by the pump engine was 18 hours 55 minutes.

The key holding the hoisting-pump crosshead to the piston rod sheared and was replaced by a tool-steel key, which gave no further trouble. The babbitt bearing of the hoisting-pump became hot and melted, probably on account of failure of the oil feed. The brasses were replaced by spare brasses at a small expense.

Although the power cost on this dredge is high as compared to the cost of power on a dredge working in the United States and using electrical current costing from ¾ to 1½ cents for each cubic yard produced, as shown in the chapter on operating costs, the power cost per cubic yard is a remarkable showing for a self-contained dredge in the northern fields, and may reasonably be compared with the stated cost of the Three Friends dredge, as the figures given for that dredge are to be considered estimates only.

The Bangor Creek dredge operating near Nome is also equipped with 3½ cubic feet buckets. Power is supplied by two semi-Diesel Bolinder engines with a total of 140 horsepower. Oil fuel is used, a mixture of 4 parts 16° crude and 1 part 48° distillate. It is stated that the average consumption of this oil is 180 gallons a day, and the average daily yardage handled over a depth of 30 feet is 1,500 cubic yards.

The successful work of recent American dredges in the Philippine Islands has greatly stimulated the dredging industry in that field. The Guamos dredge uses wood for fuel. During 1913, according to the annual report of the operating company, the dredge was in actual operation for 5,731 hours and handled 814,500 cubic yards of material. During this period 4,500 cords of wood was consumed, costing $5 per cord delivered on the dredge. The fuel cost per cubic yard thus indicated is 1.38 cents, an encouraging figure considering the conditions under which the dredge works.

NOVEL POWER PLANT OF PHILIPPINE DREDGE.

A radical innovation was the motive power designed for the Mambuloa dredge in Ambos Camarines in the Philippines.[a] A steam turbo-generator was placed on the dredge to provide the electric current, being the first instance of such a plant being used for gold dredging. Although no reports of operation are yet available, this type of power plant has been successfully used on the United States collier *Jupiter*. The dredge plant consists of a 300-horsepower wood-burning boiler with condenser and a 625-kilowatt-ampere (500 kilowatts at 80 per cent power factor) turbo-generator running at 3,600 revolutions per minute, 440-volt, alternating-current three-phase, 60-cycle, with direct connected exciter. The main drive motor is 150 horsepower with reversible speed controller, which will reduce full speed one-third. The other motors are as follows:

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[a] Janin, Charles, New Philippine gold dredges: Mining Mag., London, vol. 13, August, 1915, p. 88.

Data on Mambuloa dredge.

	Horsepower.
Main winch	20
High-pressure pump	120
Low-pressure pump	75
Screen drive	40
Stacker	40

In addition to the main power plant there is an auxiliary steam engine of 20 horsepower which can operate the winch and move the dredge in case of accident to the turbo generator, during a storm or other emergency. This engine is used for lighting the dredge when the main plant is shut down, and can also be used to take the place of the exciter in the generator should that get out of order.

Of interest in connection with this dredge is the fact that the water for the boilers comes through a special pipe line from a fresh-water spring in the hills. This plan was adopted to obviate the possibility of damage from salt water in the river, which caused serious boiler trouble on one of the dredges in the Paracale district.

A dredge of this type is expensive in first cost, as the plant includes not only an electrically-driven dredge, but also a separate power-plant built on the dredge instead of the usual plant on shore. It is asserted by the makers that the operating cost is less under these conditions than with a separate plant on shore, as all operations are under the direct supervision of the dredge master, and only the one working crew is necessary. Should a second dredge be built, as is contemplated if results are in keeping with expectations, the power plant may be moved from the dredge and suitably rearranged on shore for two dredges. The dredge has been designed with this possibility in view, and for this reason a 625-kilovoltampere generator, which is in excess of present requirements, is placed on the dredge. Seemingly another boiler is to be added to the shore plant if that is later constructed. Although no figures of operation are yet available, details will be awaited with interest on account of the new features involved.

Suction gas engines have been tried in some dredging operations in the Federated Malay States, but such figures of actual results as are obtainable would indicate that this type of power as applied has not proved a decided success.

RECLAMATION OF LAND AFTER DREDGING IN CALIFORNIA.

HYDRAULIC MINING ENDED BY LEGISLATION.

The regulation of hydraulic mining in California in the early eighties practically abolished that form of mining in the State. California had previously been famous for its hydraulic mining, and there still remain enormous gravel deposits that could be profitably worked by hydraulic methods if such operations were permissible and if early conditions regarding water supply still remained.

In the early eighties the production of gold from hydraulic mining was about $10,000,000 a year. After injunctions and regulations had been imposed the production dropped to less than $1,000,000 a year, and in 1915 only $420,770 was obtained from this source. It can be seen that at present hydraulic mining is of relatively small importance as compared to dredge mining, which has to its credit over $7,000,000 annually since 1909 and about $8,000,000 for 1915, a total closely approximating $90,000,000 having been won by this form of mining in California from 1898 to January, 1917. According to a review of hydraulic mining by the Miners' Association in 1881, "California has produced (1848–1880) between $1,100,000,000 and $1,200,000,000 of gold, of which very much the larger portion has come either directly or indirectly from the ancient or Pliocene river channels."

At different times legislation for regulating gold dredging has been proposed, not only to prevent the accumulation of débris from the dredges, but to prohibit the dredging of land that can be used for agriculture. An act of Congress dated March 1, 1893, provides for the establishment of the California Débris Commission, to consist of three officers of the Army Engineer Corps. The main work of the commission in later years has comprised efforts to improve navigation and to control flooding of the rivers, the work relating to mining being regarded of minor importance, although the commission practically has full control of the débris situation. An excellent summary of legislation relating to mining débris is given by Charles Yale.[a]

DREDGING IN CALIFORNIA NOT INIMICAL TO AGRICULTURE.

The welfare of the agricultural interests with regard to any damages or inconveniences resulting from dredge mining has been carfully looked after by the California Anti-Dèbris Association, a protective organization first formed in the early seventies. Committees from the association have on different occasions investigated conditions in some of the dredging fields of the State, and through their efforts a number of abuses have been corrected. Stream pollution by the dredges has been considered at different times, and if the spill from a dredge was found flowing into a river remedial measures were taken. The dredge men in certain districts have for years been impounding the overflow from the dredge ponds, and have spent considerable money in preventing this water from flowing into rivers. Most of the dredges now operating in California, with the exception of those in the Yuba River, are working inland and not in the river beds proper. They are not therefore obstructing the rivers with tailing or causing overflow during flood times with resultant damage to adjoining land. A few years ago, a committee appointed by an antidredge convention reported that the dredges along the Yuba and American Rivers, instead of sending down débris, were impounding considerable sand and keeping it from going into the streams.

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[a] Yale, Charles G., Mining debris legislation: California Mines and Minerals, souvenir ed., California Miners' Association, 1899, pp. 255–262.

A bill under consideration by the California legislature in 1913 proposed a basis of limitation of the industry more severe than that of damage by pollution of streams. Its purpose was to prohibit the dredging of any land that could be used for agriculture. This agitation provoked considerable discussion, and a paper published by the writer[a] was given wide circulation at the time. A number of excerpts are taken from the paper for this chapter. Fortunately for the good of the State, as well as for the interests actively affected, the bill was defeated. A similar bill was introduced, early in 1915 but was withdrawn. It is probable that other similar bills will be introduced in the future.

SIZE OF CALIFORNIA DREDGING AREAS.

The amount of land attacked by the bill mentioned was not great. Compared to the total area of arable lands in the State, it seems almost infinitesimal. At present there is probably 12,000 acres of land suitable for dredging in the State, and only about 2,500 acres of this could be classified as agricultural. According to figures of the California Development Board, there is 28,000,000 acres of arable land in the State, and if dredging of 2,500 acres of such land were projected, it will be seen that less than 0.01 per cent of the total area of arable land would be involved. The distribution of this agricultural land is approximately as follows: Butte County, 400 acres; Yuba County, 650 acres; Sacramento County, 1,200 acres; Placer County, 100 acres; Trinity County, 50 acres; Stanislaus County, 100 acres; total, 2,500 acres.

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[a] Janin, Charles, Proposed regulation of gold dredging: Min. and Sci. Press, vol. 106, March 8, 1913, p. 381.

Nearly one-half of the total agricultural land to be dredged owned by one company, the Natomas Consolidated of Californi The loss of any small part of this land useful for agriculture is full compensated for by the reclamation department of the compan The project will bring into production 60,000 acres of overflow lan along the Sacramento River that, when fully reclaimed, will ad greatly to the wealth of the State. It must also be considered th most of this overflow land was idle and practically worthless befo the work of reclamation was begun, and that it would probably hav been idle for a number of years to come except for the work of t Natomas Consolidated, made possible in part from the profits resul ing from gold dredging. At present much of this land has bee reclaimed and is being cultivated for grain, beans, rice, and alfalfa.

COMPARATIVE INCOME FROM DREDGING.

For comparative purposes the following figures may be of interes Assuming that there is 12,000 acres of land yet to be dredged in t State, and that the average depth of this land is 11 yards, the would be an average of 53,240 cubic yards, or in round figures 50,00 cubic yards, of material to the acre. The average operating cost dredging in the State may be taken as 5 cents a cubic yard, or $2,50 an acre. In 12,000 acres there would be distributed $30,000,000 labor and other operating costs. The average gross value of th material may be taken as 10 cents a yard. Accordingly, $60,000,00 will be added to the production of the State from land that otherwi would have little or no productive value.

Land that was formerly planted in vineyards and handled such at a loss has been dredged at a profit to the owner and to t State. In one area of several hundred acres the gross amou realized from grapes averaged, over a period of several years careful cultivation and attention, about $25.40 an acre per yea The net profit was less than $10 an acre for any one year and som years the property was cultivated at a loss. The gross recovery o such of this "vineyard" land as has been dredged has average $2,930 an acre. Although these figures can not be applied to a vineyards, the gold produced from dredging such land may be regarde as added to the permanent wealth of the State, whereas the crop from the vineyard can hardly be regarded in any such light.

Most of the land that has been dredged in the State was originall unsuitable for either horticulture or agriculture. A considerab part of the dredging land was previously mined by methods in us prior to the development of gold dredges, and the condition of mos of this land was such as to make it useless for any other purpos Open or partly caved shafts and pits, old dumps and heaps of ston were everywhere, and practically the only vegetation was a fe scrubby trees. The area of orchards and vineyards already dredged is probably not more than 2,500 acres; most of this is around Oroville. The loss resulting from destroying these orchards and vineyards, some of which had already been attacked by the phylloxera, can not be considered great. Though there are exceptional instances, it may be said that few of these orchards and vineyards returned much of a profit to their owners prior to dredging.

The removal of the tailing for use in the rock-crushing plants is a step toward the ultimate reclamation of that land. After the larger stones have been removed from the ground at Natoma, experiments have been made in planting olive and eucalyptus trees without any further treatment than the addition of a shovelful of loam around the roots of the trees as they are put in place. Although only a few acres have been planted in this way, enough has been done to demonstrate the feasibility of the plan and to give ample evidence of the fertility of this part of dredged ground. Successful experiments in planting trees and grape vines in the dredge tailing have been made at Oroville. Much of the success of the experiments was due to the abundance of clay present, which added greatly to the fertility of the tailing. The results show that the ground after dredging is not so worthless as was at first supposed. The results of these experiments have already been published elsewhere.[a]

RESOILING METHODS TRIED.

Numerous methods for resoiling behind the dredge have been suggested and some have been given thorough trial. One of these, obviously impracticable, entailed the use of a telescopic dredge ladder. Another suggestion was to use a dredge with two bucket lines, one to work on the gravel and the other to work simultaneously on the overburden. A somewhat more practical suggestion is to use an independent stripping plant to work in advance of the dredge. However, this method involves great expense in the handling of the material in front of the dredge and delivering it on the worked ground behind. The cost of reclaiming land by such a method would probably greatly exceed the value of the land.

With the Australian pump dredge[b] the material could be worked in sections and delivered advantageously to different places, but the working cost would closely approximate 25 cents a yard, an excessive cost on any ground being dredged in the United States.

Attempts to reclaim or resoil land during dredging operations have not so far been carried on to any great extent in California. Hurst[c] describes a resoiling device on a small dredge near Loomis.

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[a] Editorial. Min. and Sci. Press, vol. 105, Oct. 26, 1912, p. 541.

[b] Janin, Charles, An Australian pump dredge: Min. and Sci. Press, vol. 103, Nov. 11, 1911, p. 614.

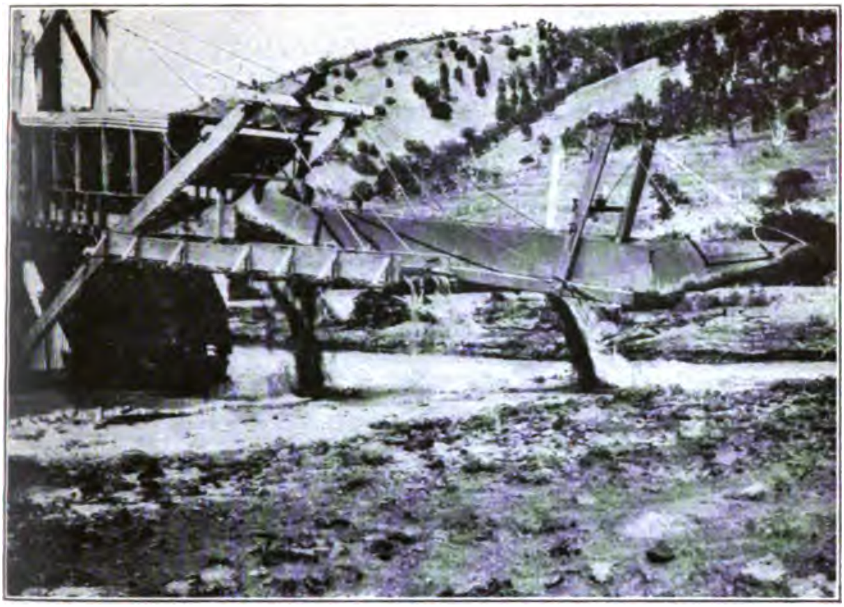
[c] Hurst, G. L., Resoiling after dredging in California: Min. and Sci. Press, vol. 107, Nov. 8, 1913, pp. 719-720.

The ground worked was only 12 to 15 feet deep and averaged about 75 per cent of fine material, about 2 feet being top soil. The principle employed was to increase the height of the upper tumbler above the deck to give greater clearance to the tail sluices. The material from the tables passed to three tail sluices, one on each side and one in the center, the center sluice passing 20 feet beyond the side sluices. The conveyor stacker was eliminated and the oversize from the screen fell on a small grizzly, the smaller material going to the central sluices and the coarser to the side sluices. The reclamation of this ground presented no unusual difficulties and a total of about 50 acres was resoiled and planted, but the device used would not prove satisfactory on much deeper and coarser gravel.

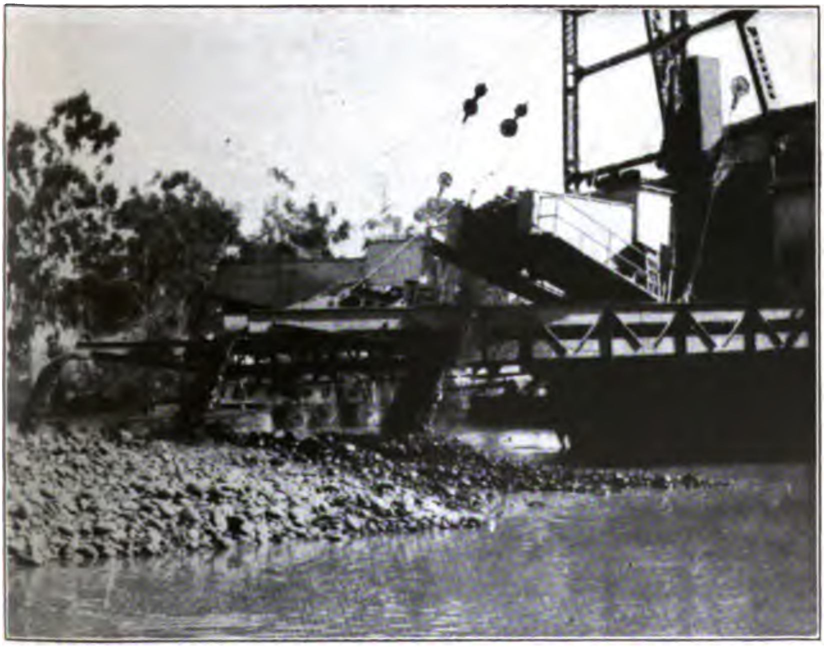
With the exceptions mentioned and a few attempts on Butte Creek in Butte County no determined effort has been made in the past to reclaim dredging land, although considerable thought has been given the matter at times by different dredging operators, and various plans for the reclamation of land already dredged has been proposed.

In addition to the gold recovered from the gravel, the reclamation of the land for agriculture might be a considerable factor in estimating the total profit to be won from the use of a mammoth dredge for this class of work. The first dredges, in turning over the ground, necessarily deposited the top soil on the bottom under the gravel and bowlders from the tailing stacker. After the greater part of the coarser gravel has been removed for use as crushed rock, with some such arrangement as that being tried in New Zealand (Pl. LXIII *A*) in resoiling experiments, this soil now below the gravel could be largely redeposited on the top of the coarser material in resoiling. Any estimate of the operating cost of a dredge of this character is, of course, pure speculation, but there seems every reason to expect that, under favorable conditions, this cost would be low.

In the Oroville field work, some encouraging recoveries have occasionally been made, when a dredge moving to a new location has redredged tailing. These in general are due to inefficient washing of the ground in the first instance, or to the dredge digging in a part of the bedrock section that had been passed over in the first dredging. However few dredge operators think that this land can be redredged at a profit, although a much lower operating cost has been shown by the dredges when working through tailing. Unquestionably this material can be handled much easier than ground that has never been dredged, especially in certain sections of Oroville where the gravel is cemented.



A. SLUICE BOX AND SOIL CHUTE OF HINNOMUNJE DREDGE, OMECO, NEW ZEALAND.



B. TAILING DISTRIBUTOR ON NATOMA NO. 4 DREDGE.

LAND RECLAMATION DEMANDS DREDGE CHANGES.

To reclaim land while the dredge is at work it is necessary to make some radical changes in the distributing end of the dredge, which as has been stated, is not difficult when shallow land is to be dredged. In land with a deposit of 20 feet or more, the problem is not so easy. Considerable thought has been given to this matter by the engineers of the Natomas company and a dredge has recently been built embodying their ideas to solve the problem of replacing the soil behind the dredge in suitable condition for farming. In order to get sufficient clearance for the tail sluices all the machinery is raised about 6 feet higher than usual. To obtain a more even distribution of the coarse material two tailing stackers are fed from a double hopper arranged at the end of the revolving screen to divide equally the material supplied each. The dredge is equipped with double-deck tables and tail sluices which are set much closer than usual, a number of sluices on the lower tables being omitted in order to allow room for cleaning up, etc. The double tailing-stackers which are shorter than usual, being 56 feet long with an elevation of 1 inch in 12 inches as compared to the usual stacker inclination of 18° (or 3.7 inches in 12 inches) are mounted on swivels, as are also the side sluices. It is not expected to swing these tailing distributors from side operation, as the swing of the dredge will provide for lateral distribution. The swivel arrangement is used only to insure horizontal adjustment for these distributors under different conditions of operation. Plate LXIII, *B*, shows the tailing distribution of this dredge at work.

The dredge started work in January, 1916, and operations are being watched with interest. A number of changes have been made since the dredge first started work when it was found the arrangements originally planned were not successful, and after the problem of recoiling is effectively solved similar arrangements will be adopted on No. 1 dredge when that boat is rebuilt. The land that is to be dredged is 20 to 40 feet deep and a great part of it is planted to vineyard. If an extra cost of one-half cent a yard is caused by experimentation, special equipment, and operation, and assuming that the ground to be dredged will average 40,000 yards to the acre, an extra cost of $200 an acre will be incurred. After the problem of resoiling coincidental with dredging has been solved, the extra cost involved will be small but at present the Natomas company, working for the best interests of the State, is going to considerable more expense than could be borne by a smaller company.

FUTURE OF DREDGING.

The highest point of production for the gold-dredging industry
the United States has been reached, and during the next few year
marked falling off in gold production from dredging operations m
be expected. The success attending the flotation of the larger dre
ing companies proved an incentive for explorations and exami
tions in various States, and all land thought attractive for dredgi
was soon taken. The problem of finding suitable ground for dredgi
is difficult, although with the great improvements made in dredgi
machinery and the resultant reduction in operating costs, it is obvio
that gravel beyond the possibility of economic dredging under ea
conditions could now be profitably handled. This is shown by t
operations of nearly all the large dredging companies whose availa
dredging ground has been greatly increased as working costs ha
been reduced.

A fact that also draws attention to areas once considered unpro
able is that the life of some of the successful dredging companie
drawing to a close; that is, the land for the working of which t
companies were originally formed is being rapidly worked out. S
eral of these companies are realizing that their ground is nea
exhausted and that they have some dredges in good condition,
for some years of operation. What is probably more importa
they also realize that they have also well-organized working forc
valuable experience in dredging matters, money in the treasury, a
the confidence of shareholders.

Aside from such influences, there is no question that under fav
able conditions areas that were once considered unsuitable for e
nomic dredging can now, with improved methods, be profital
handled. The possibilities of reclamation of dredged land m
prove important in estimating the profits to be won from dredgi
areas in California. The success attending the efforts of some expe
ments in this direction in a small way has greatly broadened t
expectations for the future of what was, until recently, conside
worthless land. A more liberal attitude on the part of the agric
turists and arrangements permitting the modification of some
the rulings of the Debris Commission in California would open to t
dredging industry areas that are at present closed.

FEASIBILITY OF REDREDGING CERTAIN AREAS.

The possibility of redredging some of the tailings from the ea
dredges at Oroville, partly with the idea of reclaiming lands th
would command a fair market, is occasionally discussed in the no

technical press. This proposal has not met with any enthusiasm among the conservative operators, but it should be realized that the greater improvements in the dredging industry are not due to the most conservative men; one must be optimistic to undertake new methods. The reclamation of ground after dredging is discussed in a previous chapter. Altogether it seems probable that dredging enterprises will continue in California for some years to come, although the amount of gold won from dredging operations in the State was at its peak in 1913. During 1915–16 there have been a number of new dredges planned and at the time this is written there are nine new dredges either under construction or projected for the California fields alone. Four of these are 16-foot boats, three 9-foot, and two are old dredges being moved to new locations. The writer pointed out in a paper read before the San Francisco meeting of the American Institute of Mining Engineers in California, 1911,[a] that dredges of even larger capacity than those now in use for gold dredges had been considered in the reclamation of dredged areas. The following extract from the paper is pertinent:

> With the development of the gold dredge to its present efficiency, the question is often raised as to when the limit in size for economic dredge installation will be reached. Much depends upon the conditions met in operation. There is no question as to the mechanical possibility of larger buckets. In Boston harbor a bucket-elevator dredge equipped with buckets of 2 cubic-yard capacity has been successfully operating for some years on harbor work, and on the Danube River in Germany a bucket-elevator dredge having 2.5 cubic-yard buckets is now in operation. While the mechanical possibilities have thus been proved, to apply such radical changes in size to the gold dredge of to-day would necessitate an entirely different arrangement of the gold-saving tables and would probably result in a general modification of the whole gravel-washing apparatus now in use. Even the most optimistic advocates for increasing the size of the dredge buckets would hesitate at recommending a 2 cubic yard bucket, which is nearly four times the present size of the buckets on the largest gold dredges in operation, but there are a number of engineers who believe that the bucket-elevator dredge with buckets having a capacity of 1 cubic yard will be constructed before long. While a dredge of this character would necessarily be equipped with heavier machinery and a larger hull than those on the present 15 cubic-foot boats, it is, as before stated, quite possible that, with modifications of the washing apparatus, the hull of the 1 cubic-foot dredge may not be proportionately larger. The present 15 cubic-foot boats have a hull 60 by 150 by 12.5 feet, with a deck overhang of 5 feet on either side, making a total width of 70 feet. The gold-saving tables are of the double-bank type and have an approximate area of 7,000 to 8,000 square feet. Without some change in the washing apparatus, it can readily be seen that 14,000 square feet of table area would necessitate a hull of greatly increased size, or additional tiers of tables, for which an increased length of bucket ladder would be required to elevate the gravel to the additional height, or a general change in the design of the boat. Practice has demonstrated that when digging free-washing gravel the table area of the 15-foot boats is considerably in excess of all requirements, and some operators contend that it would not be necessary to proportionally increase the table area when buckets of 1 cubic-yard capacity are constructed.

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[a] Janin, Charles, Present-day problems in California gold-dredging: Trans. Am. Inst. Min. Engrs., vol. 42, 1912, pp. 855–873.

There may be a field for dredges of this size, for instance, in the Oroville and Folsom fields, to redredge the tailing piles left from the first dredging operations. After many of the cobbles have been removed for the rock-crushing plants, the ground, if dredged, will, in many cases, yield a fair return. Especially would this be the case in the areas where the early dredges worked, as the gold-saving apparatus of the first successful dredges was not as efficient as that in present use.

ALASKA FIELDS PROMISING.

In Alaska the success attending the operations of some of the dredges recently built there has greatly widened the field for the dredge-construction companies. Previously the demand has been for small dredges, especially on the Seward peninsula, but the successful operation of the large dredges of the Canadian Klondike Co. on the Yukon, and the Bonanza Creek operations of the Yukon Gold Co. have encouraged investigation of areas in other parts of the northern field. Unfortunately, many of the dredges built in Alaska have been constructed prior to adequate prospecting of the ground they were to work. Failure to sample a dredging property until the dredge has been built, when "large practical samples" can be taken, is almost to invite loss. Although the dredge may prove a success mechanically, the company is generally doomed to economic disaster unless more favored with good luck than good judgment.

Besides the possibility of new dredging enterprises in California and Alaska, the attention of dredge operators is being attracted to areas in other western States where the gold content has previously been considered too low. The difficulties surmounted in dredging operations in California, Montana, Colorado, and other fields, and the low working costs obtained in these fields and in Idaho, open possibilities unthought of a few years ago.

DREDGING OUTLOOK IN OTHER COUNTRIES.

In addition to the possibilities in the United States and Alaska, considerable attention has been given to foreign fields. Chief among the countries being considered are Russia, including Siberia; South America, principally meaning Colombia; and the federated Malay states. The success attending the work of some Russian dredges and the purchase early in 1916 of three American dredges for the Russian field has stimulated inquiry into the possible openings for American capital in dredging projects in that field. It is rumored that the Russian Government has considered the advisability not only of taking off the high duty heretofore imposed on machinery for gold mining, but also of subsidizing to some extent gold-mining companies; such procedure would undoubtedly stimulate investigations of dredging areas in that country. In new enterprises in Russia or Siberia, it will be found advisable to have the assistance of influential Russians. It is held by engineers qualified to pass an opinion that in Russia and Siberia, so far as operating conditions are concerned, there are a number of areas of gravel that can be profitably dredged, and the examination of some of these fields may result in more dredges being erected in the near future. A pertinent point in this connection is that the Russian peasant in some localities makes a really first-class mechanic, so that it is not necessary for a foreign dredging company to import expensive laborers for all positions on a dredge. Conditions of operation in Russia and Siberia, aside from taxes and government regulations, are very similar to those in the Yukon and Alaska, and operating costs should, under ordinary conditions, be closely similar. The success made by the gas, oil, and wood engines in Alaska, described elsewhere, should encourage dredge installations in localities where electric power is not obtainable.

There are undoubtedly in Colombia and other South American countries areas that offer promising returns from dredging operations. A number of investigations have been made in some of the countries in the interest of American capital, but the difficulties of transportation, of climate, and of procuring sufficient cheap power have proved somewhat discouraging. Here also some of the improvements in power arrangements for the self-contained dredge will be worth consideration.

In the Philippines the success of the Guamos dredge has encouraged other operations and an American dredge built in 1914 for the Mambulao Dredging Co. embodies some striking innovations for generating power on board the dredge. This dredge is described elsewhere (see p. 196) the results of its operation are awaited with interest.

An American dredge was constructed for the Chiksan Mining Co. in Korea during the latter part of 1916. Its success will do much to encourage further dredge installation on the Orient. In Portugal a dredge built under plans of an American dredge-construction company for American capitalists is being operated in tin deposits, and its success has awakened interest in tin dredging. In 1915–16, several orders for new dredges for the Malay field were placed with American dredge-construction companies, and this field offers considerable promise to the wide-awake constructor.

Although there is room for considerable improvement in the gold dredge, the wasteful and inefficient methods usually employed on tin dredges should give our dredge-construction engineers a splendid opportunity for increasing the recovery. The profits made by some of the tin dredges have been large, and doubtless no necessity has yet been felt for inquiring closely into methods for decreasing the loss in the dredge tailing. It is when low-grade material is handled that metallurgical methods are improved. It is probable that American operators will be offered more or less attractive opportunities to participate in some of these tin-dredging projects in the near future. Altogether the possibilities of successful dredging operations in some of the foreign countries mentioned seem to be attractive.

SUCCESSFUL DREDGING OPERATIONS DEMAND COMPETENT EXPERTS.

In making investments in dredging enterprises in foreign countries, as elsewhere, much money may be saved if prospective investors will insist on having reports by experienced engineers. With the drawbacks of operating in foreign countries, together with unusual government regulations, and other disadvantages incident to dredge work away from home, possible dredging areas in the United States, though of lower grade than some that are being at present worked, and of much lower gold content than is necessary for profitable operation in foreign countries, will probably be more attractive to conservative investors than the foreign fields. Many investors, however, will be attracted by the higher gold content claimed for the gravels in foreign fields, and some operators, acting on the reports of competent engineers, will risk enterprises in foreign countries, and, profiting by their knowledge of the industry, will build suitable dredges and will realize large returns on the initial investments.

Close estimates of the possible profits from gold-dredging projects can be made, but competent experts must have charge of the details of examination and prospecting and the development operations must be under the direction of efficient management, which entails competent engineering advice.

LIST OF DREDGING OPERATIONS.

A fairly complete list of dredging operations throughout the world follows. The list does not include Australian, New Zealand, nor most Siberian dredges. In a table of this kind full credit cannot always be given the makers, as, for instance, reconstructed dredges and dredges that were constructed by one company and part of the machinery is furnished by another.

Incomplete list of all dredging operations.[a]

[Not inclusive of Australian, New Zealand, nor most Siberian dredges.]

Name of company.	Headquarters.	Situation of dredge.	Make of dredge.	Bucket capacity.
				Cu. feet.
Alaska Gold Dredging Co.	Council, Alaska	Council, Alaska	Special	2½
Alaska-Kougarok Dredging Co.	Nome, Alaska	Kougarok River	do	2¾
Alta Bert Gold Dredging Co.	Trinity Center, Cal.	Trinity Center, Cal.	Union Con	7½
American Gold Dredging Co.	Nome, Alaska	York, Alaska	American	1½
Do	do	do	do	1½
American Placer Mining Co.	Spokane, Wash	Pierce, Idaho	Risdon	3½
American Tin Co.	San Francisco, Cal.	York, Alaska		
Amur Gold Mining Co.	Blagovestensk, Siberia.	Kharga River	Conrad	5
Anderson, Andy	Nome, Alaska	Center Creek	Union	3½
Andrada Mines (Ltd.)	London	Macequece, Portuguese East Africa.	Bucyrus	7½
Anglo Alaskan Gold Dredging Co.	Nome, Alaska	Nome, Alaska	Special	2
Arctic Gold Dredging Co.	do	Hobson Creek	Union Con	2¾
Bangor Dredging Co	do	Bangor Creek	do	3½
Behring Dredging Co.	do	Teller Creek	do	2½
Berry Dredging Co., C. J	Circle, Alaska	Mammoth Creek	do	3½
Blue Goose Mining Co.	Council, Alaska	Ophir, Alaska	Reconstructed	3½
Bohannon Dredging Co.	Salmon, Idaho	Salmon, Idaho	Special	5
Borden Co., Ivy L	San Francisco, Cal.	Jenny Lind, Cal	Bucyrus	6
Boston & Idaho Gold Dredging Co.	Idaho City, Idaho	Idaho City, Idaho	Yuba	5
Bourbon Dredge	Nome, Alaska	(See Nome Holding Co.)		
Budd Creek Gold Dredging Co.	do	Budd Creek	Special	3
Butte Dredging Co.	San Francisco, Cal.	Jenny Lind, Cal.	do	3½
Canadian Klondike Mining Co. (Ltd.).	Dawson, Alaska	Dawson, Alaska	Marion	7½
Do	do	do	do	16
Do	do	do	do	16
Do	do	do	do	16
Candle Creek Mining Co.	Candle, Alaska	Candle Creek	Union	1¾
Casadepaga Gold Mining & Dredging Co.	Solomon, Alaska	Casadepaga River	Special	2½
Cie. des Mines de la Faleme	Paris, France	Senegambia	Lobnitz	7
Chestate Dredging Co	Gainesville, Ga	Gainesville, Ga	Bucyrus	3½
Chiksan Mining Co.	Chiksan, Korea	Chiksan, Korea	New York Engineering.	10
Compagnie Francaise du Mataroni.	Cayenne, France	French Guiana	Lobnitz	3½
Conrey Placer Mining Co.	Ruby, Mont	Ruby, Mont	Marion	7½
Do	do	do	do	7½
Do	do	do	do	9
Do	do	do	do	16
Deering Dredging & Mining Co.	Deering, Alaska	Deering, Alaska	Special	2¼
Derry Ranch Gold Dredging Co	Leadville, Colo	Malta, Colo	New York Engineering.	6
El Oro Dredging Co.	do	Yreka, Cal	Special	5
Empire Gold Dredging Co.	John Day, Oreg	John Day	Stearn Rodgers.	5
Ernst Alaska Gold Dredging Co.	Nome, Alaska	Nome, Alaska	Special	1¾
Federal Mines Co.	Fitting, Nev	Unionville, Nev	do	5¼
Flodin Gold Mining & Dredging Co.	Chicago, Ill	Solomon, Alaska	do	2½
Flume Dredge Co	Nome, Alaska	Ophir Creek	Straub	2¼
Do	do	Crooked Creek	do	2¼
Freeze Dredging Co	do	Nome, Alaska	Special	2
French Gulch Dredging Co.	Breckenridge, Cal	Breckenridge, Cal	Bucyrus	5

[a] Based on list from Eng. and Min. World, Jan. 6, 1916.

Incomplete list of dredging operations—Continued.

Name of company.	Headquarters.	Situation of dredge.	Make of dredge.	Bucket capacity.
				Cu. feet.
Gardella, Lawrence	Oroville, Cal	Honcut, Cal		5
Do	do	Garvin, near Redding, Cal.		5
Gumaos Placer Co	Manila, P. I	Guamos, P. I	New York Engineering.	6
Hastings Creek Dredging Co	Nome, Alaska	Hastings Creek	Special	1½
Holyoke Gold Dredging Co	do	Nome, Alaska	do	9
Idaho Power Co	Spokane, Wash	Pierce, Idaho	do	3½
Immachuck Dredging Co	Deering, Alaska	Immachuck River	Union Con	3
Isabel Gold Dredging Co	(See Borden Co., Ivy L.)			
Johnson, Iver	Candle, Alaska	Kugruk River	Union Con	3½
Julien Gold Mining & Dredging Co.	Nome, Alaska	Osborne Creek	do	2¾
Kelliher Dredging Co	do	Nome, Alaska	Special	2½
Kentucky Ranch Gold Dredging Co.	Oroville, Cal	(See Gardella, Lawrence.)		
Kimball Dredging Co	Solomon, Alaska	Adams Creek	Special	2½
Kirtley Creek Gold Dredging Co.	Salmon, Idaho	Salmon, Idaho	Yuba	9
Kugruk Dredge (see Iver Johnson).	Candle, Alaska	Kugruk River	Union Con	3½
La Grange Gold Dredging Co	La Grange, Cal	La Grange, Cal	Special	7
Lincoln Dredge	Auburn, Cal	(See Gardella, Lawrence.)		
Lubbe Gold Dredging Co	Council, Alaska	Council, Alaska	Special	2½
Malaguit Dredging Co	Manila, P. I	Parcale, P. I	Yuba	5
Malayan Tin Dredging (Ltd.)	London, England	Btu Gajah, Perak	Lobnitz	3¼
Do	do	do	do	5
Do	do	do	do	5
Do	do	do	do	5
Mambuloa Placer Co	Manila, P. I	Mambuloa, P. I	New York Engineering.	8
Marysville Dredging Co	Marysville, Cal	Marysville, Cal	Marion	8
Do	do	do	do	8
Do	do	do	Union Iron W'ks	16
Merced Dredging & Mining Co	Snelling, Cal	do	Yuba	3¾
Natomas Consolidated of California (Feather River).	San Francisco, Cal	Oroville, Cal	Yuba	8
Do	do	do	do	8
Do	do	do	do	15
Natomas Consolidated of California.	do	Natoma, Cal	do	13½
Do	do	do	Bucyrus	8½
Do	do	do	do	8½
Do	do	do	do	9
Do	do	do	do	9
Do	do	do	do	9
Do	do	do	do	13½
Do	do	do	Yuba	15
Do	do	do	do	15
Do	do	do	Bucyrus	15
Nechi Mines, Colombia	(See Oroville Dredging, Ltd.)		Union and Fraser & Chalmers.	8½
Nicola Pavda Mining District Co.	Petrograd, Russia	Perm	Marion	7½
Nome Holding Co.[a]	Seattle, Wash	Nome, Alaska	Special	10
Do	do	do	do	7
Do	do	do	do	7
Northern Lights Mining Co	Council, Alaska	Ophir Creek	Union Iron W'ks	3
Oroville Dredging (Ltd.)	San Francisco, Cal	Oroville, Cal	Yuba	7
Do	do	do	do	7
Do	London, England	Nechi mines, Zaragosa, Colombia.	Union and Fraser & Chalmers.	8½
Do	do	Pato mines, Zaragosa, Colombia.	Yuba	9
Oroville Union Gold Dredging Co.	San Francisco, Cal	Oroville	Special	6½
Oro Water, Light & Power	do	do	New York machine.	6
Do	do	do	do	5
Do	do	Camanche, near Clements, Cal.	do	9
Do	do	do	do	6
Orsk Gold Fields	London, England	Nikolaievsk, Siberia	Yuba and New York Engineering.	5
Do	do	do	New York Eng.	8

[a] Formerly Nome Consolidated Dredging Co.

Incomplete list of dredging operations—Continued.

Name of company.	Headquarters.	Situation of dredge.	Make of dredge.	Bucket capacity.
				Cu. feet.
Otter Dredging Co.	Iditarod, Alaska.	Otter Creek	Union Con.	3½
Pacific Gold Dredging Co.	New York, N. Y.	Oroville.	Bucyrus	7½
Do	do	Chico, Cal.	do	5
Do	do	Auburn, Cal.	do	7½
Do	do	Carville, Cal.	Marion	9½
Parcale Bucket Dredging Property (Ltd.)	Parcale, P. I.	(See Philippine Dredges, Ltd.)		
Pasadena Dredging Co.	Teller, Alaska.	Teller, Alaska.	Special	3
Pato Mines	(See Oroville Dredging Ltd.)			
Pavda, Nicola, Mining District Co.	Petrograd, Russia.	Perm	Marion	7½
Philippine Dredges (Ltd.)	Parcale, P. I.	Parcale	Australian	5
Do	do	do	do	5
Do	do	do	do	5
Do	do	do	do	5
Do	do	do	do	7
Portuguese American Tin Co.	Lisbon, Portugal.	Portugal	Union Con	4
Powder River Gold Dredging Co.	San Francisco, Cal.	Sumpter, Oreg.	Yuba	7
Do	do	do	do	9
Plain Mining & Dredging Co.	Nome, Alaska.	Nome, Alaska.	Special	3¾
Riverton Dredging Co.	Shoshoni, Wyo.	Shoshoni, Wyo.	No recent report.	
Ropp Tin (Ltd.)	London, England	Bukeru, No. Nigeria.	Bucyrus	2½
Do	do	do	do	2½
Ruby Gold Dredging Co.	Solomon, Alaska	Casadepaga River	Union	2¾
Santa Fe Dredging Co.	Golden, N. Mex.	Golden, N. Mex.	Special	
Saunders Creek Dredging Co.	Nome, Alaska.	Saunders Creek	do	3½
Saupe Placer Mining Co.	Council, Alaska.	Council, Alaska.	do	2¼
Seward Dredging Co.	New York, N. Y.	Nome, Alaska.	Bucyrus	3½
Shasta Dredging Co.	San Francisco, Cal.	Larkin, Cal.	Yuba	5
Sievertson Johnsen Mining & Dredging Co.	Solomon, Alaska	Solomon, Alaska.	Special	2½
Sioux Alaska Mining Co.	Nome, Alaska.	Hastings Creek	Union	2¾
Siskiyou Dredging Co.	Hammonton, Cal.	Fort Jones	Yuba	5½
Tin Bentong (Ltd.)	London, England	Kuala Lumpur, F. M. S.	Bucyrus	7
Tincup Gold Dredging Co.	Denver, Colo.	Tin Cup, Colo.	Marion	3½
Tin Mines of Siam (Ltd.)	London, England	Pong District, Siam.	Bucyrus	5½
Tonopah Placers Co.	Breckenridge, Colo.	Breckenridge, Colo.	do	9½
Do	do	do	do	9½
Do	do	do	do	9
Trinity Gold Dredging Co.	Lewiston, Cal.	Lewiston, Cal.	Special	11
Uplift Mining Co.	Council, Alaska	Council, Alaska.	Union	2
Valdez Dredging Co.	San Francisco, Cal.	Junction City, Cal.	Union Iron W'ks	7
Warm Creek Dredging Co.	do	do	Special	2½
Welch, J. M.	Teller, Alaska.	Windy Creek	Union	3
Wild Goose Mining & Trading Co.	Ophir, Alaska.	Golovin Bay	Yuba	3½
Willow Creek Dredging Co.	San Francisco, Ca	Council, Alaska.	Special	2½
Wonder Dredging Co.	Nome, Alaska.	Nome, Alaska.	(See Nome Holding Co.)	
York Dredging Co.	York, Alaska.	Buck Creek	Union	2½
Yosemite Dredging & Mining Co.	Snelling, Cal.	Merced Falls	Yuba	3¾
Yuba Consolidated Gold Fields.	Hammonton, Cal.	Hammonton, Cal.	Bucyrus and Yuba.	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	15
Do	do	do	Yuba	15
Do	do	do	do	16
Yukon Gold Co.	New York, N. Y.	Ruby, Alaska.	Union Con.	3½
Do	do	Iditarod, Alaska	Special	7½
Do	do	Dawson, Alaska	Marion	7
Do	do	do	Special	7
Do	do	do	Bucyrus	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	5
Do	do	do	do	5
Do	do	do	do	5

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INDEX.

A. Page.

- Accident prevention on dredges, means of . 133-137
- Accounts, running, method of keeping ... 150
- Accounting methods, forms for ... 163, 164
- Air compressor, portable, for riveting buckets, use of ... 125
- Alaska, dredge in, time lost on ... 129
  - dredges in, cost of ... 196
    - number of ... 11
  - dredge wages in ... 162
  - drilling of placer ground in, cost of ... 52
  - first dredges in ... 13
  - gold dredging in ... 16
    - future of ... 206
  - gold dredging districts in ... 14
  - gold from dredges in, value of ... 11
- Alder Creek district, Mont., description of ... 22
- Alder Gulch, Mont., gold dredging at ... 15
- Alternating current motors for dredges, advantages of ... 98
- Amalgamation of colors from pannings ... 37
  - of concentrates ... 149
- Andrada dredge, P. E. A., handling of bowlders on ... 123

B.

- Bannack, Mont., early gold dredging at ... 8, 9
- Bearings for tumblers, types of ... 58
- Belts, limiting speed of ... 135
  - for conveyors, types of ... 79
    - figure showing ... 79
  - *See also* Stacker, conveyor belts, and tailings.
- Belt-conveyor tailing stacker, advantages of. 77
- Berry dredge Alaska, working costs of 158, 195
- Bibliography ... 212, 216
- Biloro dredge, method of washing buckets on. 119
- Bits, drilling, handling of ... 32
- Black sand from drillings, examination of ... 38
- Blasting of dredging ground, method of ... 121
- Boise Basin, Idaho, view in ... 22
- Boston & Idaho Co., Idaho, dredging operations of ... 12
  - working cost of ... 157
- Bowlders, handling of, methods of ... 123
- Bradford, A. H., cited ... 188
- Breckenridge, Colo., dredge, view of ... 10
  - dredging district, description of ... 21
    - gold dredges in ... 15
- Buckets, California type, view of ... 56
  - capacity of, estimation of yardage from .. 151
  - cleaning of, methods of ... 119
  - changing of, practice in ... 125
  - construction of ... 55
- Buckets, "square bottom," advantages of ... 55
  - "three eye," disadvantages of ... 55
  - limitations on size of ... 205
  - on California dredges, capacity of ... 154
  - views of ... 56, 57, 58
- Bucket chain, drives for ... 97
- Bucket dredge, early California, description of. 8
  - first use of ... 5
- Bucket excavators, use of ... 5
- Bucket idler, construction of ... 73
  - view of ... 73
- Bucket line, close connected, advantage of .. 154
  - cost of ... 54
  - main drive of, design of ... 60, 61
  - motor for, design of ... 93
  - motor drive for ... 96
  - repairing of, facilities for ... 135
- Bureau of Mines, investigations of ... 3
- Butte dredge, rebuilding of, cost of ... 108

C.

- Caissons, for prospecting, size of ... 29
- Calaveras County, Cal., gold dredges in ... 14
- California, dredge wages in ... 162
  - dredges in, number of ... 11
    - operating costs of ... 176
  - dredging districts of ... 14-17
    - area of ... 199
    - character of ... 200
  - dredging in, beginning of ... 5, 8
  - future of gold dredging in ... 205
  - gold from dredges in, value of ... 7, 11, 198
  - hydraulic mining in, gold yield of ... 198
  - limitations on gold dredging in ... 199
  - placer gold in, occurrence of ... 18
- California Débris Commission, creation of ... 198
- California dredges, gold-saving tables on, area of ... 146
- California Industrial Accident Commission, safety orders of ... 133
- California type of bucket, view of ... 56
- Carr, George, acknowledgment to ... 4, 10
- Casing, device for pulling, figure showing 35
  - for drill holes, use of ... 31
- Casings for spuds, construction of ... 66
- Castle Creek dredge, rebuilding of, cost of 109
- Cemented gravel, drilling in, precautions in .. 33
- Channel, leaving of, method of, figure showing 115
- Clean-up on Yuba dredges, methods used in .. 149
- Clay cut-out for dredge, figure showing ... 120
- Clutha River, N. Z., dredging on ... 5, 6
- Coil, Charles, acknowledgment to ... 4
- Colorado, dredging fields in, description of ... 21

Page.
Colorado, gold dredges in, number of........ 11
  gold dredging in........................ 15
  gold from dredges in, value of............ 11
Colorado Gold Dredging Co., estimation of value of dredge ground by, differences in........................ 188
Colors, amalgamation of, difficulties in, causes of................................ 37
  classification of........................ 34
  size of.................................. 34
Concentrates from jigs on Natomas Consolidated dredge, value of............ 140
Conrey No. 3 dredge, view of............... 158
Conrey No. 4 dredge, gold saving on, area of. 146
  gold-saving tables and sluices on, view of................................ 98
  sizes and uses of wire rope on............ 85
  washing screen of, figure showing........ 103
  water rheostat on, arrangement for...... 97
Conrey Placer Mining Co., accounting, forms used by........................ 170-172
  dredge crews of, size of.................. 162
  dredges of, digging motors on, types of... 97
    electric current used on, voltage of... 99
    electric power consumed on.......... 95
    general data on...................... 86-89
    gold-saving devices on, recovery by.. 184
    gold-saving tables on, cleaning of.... 147
    operating costs of.................. 153, 173
    percentage of gold recovered by...... 190
    running speed of..................... 118
    running time and stops of........... 118
    save-alls on, gold recovery by........ 44
    time lost on, causes of............... 130
    total working costs on............... 156
    use of undercurrents on............. 143
    yardage handled by................. 151
Conveyor belts, capacity of.................. 81
  life of................................... 80
  types of, figure showing.................. 79
  widths of................................ 80
  See also Belt conveyor.
Core from drill hole, handling of............. 33
Cost accounting for dredges, requirements of. 150
Costs of dredging.......................... 150-165
  variations in.......................... 154, 158
  See also Accounting, operating costs, and various dredges and dredging companies named.
Current-wheel dredge, first use of............ 6
  view of................................. 6
Curtis, R. P., cited......................... 188

D.

Depreciation charges, method of assessing.... 150
Depth chart, figure showing................. 126
  use of.................................. 126
Digging motor, on Conrey dredges, control of. 97
  power consumption of................... 94
Direct-current motors, for dredges, disadvantages of........................... 98
Distributors for screens, types of............ 77
  view of................................. 76

Page.
Dredge, Alaskan, plan of, figure showing.... 106
  bucket-elevator type, first use of......... 5
  capacity of, estimation of, precautions in. 152
  development of........................ 3
  digging with, to leave a channel, method of, figure showing.................. 115
  electrically driven, plan of, figure showing........................... 104, 105
  Philippine Islands, view of.............. 7
  size of, factors determining.............. 54
  yardage, handled by, calculation of, figure showing....................... 127
Dredges, California, gold output of, value of.. 7
  moving of, feasibility of.................. 110
  operation of, factors governing........... 113
  plans of, figures showing............. 80, 82, 127
  rebuilding of, cost of.................. 108-109
  rebuilding of, feasibility of............... 107
  repairing of, methods of.................. 124
  second-hand, value of.................. 111, 112
  selection of, factors determining.......... 51
  size of, factors determining.............. 51
  spare parts for, need of.................. 114
  steam, first use of........................ 6
  See also dredges named.
Dredge buckets, capacity and monthly output of............................. 54
Dredge flotation, diagram of, figure showing.. 85
Dredge hulls, dimensions of................. 70
  wooden, life of.......................... 50
Dredging, gold, in the United States, beginning of.............................. 4
  in New Zealand, beginning of............ 5
  need of expert advice in................. 208
Dredging ground, life of, estimation of....... 50
    examples showing.................... 51
  prospecting of........................... 27
  gold content of, determination of........ 126
Dredging operations of the world, list of..... 209
Dredging projects, minimum profit........... 50
Drill crew at work, view showing............ 24
  members of............................. 42
  wages of............................... 42
Drill, gasoline driven, view of................ 51
  for frozen ground, description of......... 52
Drill hole, beginning, methods of............ 32
  pumping of, method of.................. 37
Drill holes, erratic values from, treatment of. 46
  in placer ground, figure showing......... 47, 48
  material from, computation of value of... 45
  recovery of gold from, determination of.. 44
  values from, plotting of, figure showing.. 46
Drills, prospecting, hand, types of 49
Drills, prospecting with, cost of.............. 30
    errors in............................. 28
    view of.............................. 25
Drilling by contract, example of............ 52
  cost of................................. 42
    per foot............................. 41
  rate of, causes affecting................. 41
Drilling stroke, length and speed of.......... 31
Drilling tools, washing of................... 34
Drillings, determination of value of ground from.............................. 44

Page.

Drillings, measurement of, method for....... 44
  quantity to be taken, factors determining. 43
Drive, main of bucket line, gears for......... 60
Drive shoe, usual size of..................... 31

E.

Electric current, for dredges, voltage of...... 99–103
  use of..................................... 93–106
Electric motors, cost of, variations in......... 158
  See also Motors, electric.
Electric power, cost of, per cubic yard dug.. 95, 96
Electric wiring, proper insulation of......... 135
El Oro No. 1 dredge, rebuilding of, cost of... 109
Empire drill, description of................. 49
  view of.................................. 48, 49
Excavators, orange-peel, for gold dredging.. 5
Expert advice in dredging, need of.......... 208

F.

Fielding L. Graves, dredge, Montana, description of......................... 8
Fire extinguishers on dredges, need of...... 136
Fire pumps, auxiliary, advantages of......... 136
Flotation diagram of dredge, figure showing. 85
Folsom field, Cal., description of............ 20
  view in.................................. 20
Friction clutches for dredge winches, advantages of......................... 65
  for ladder-hoist winch................... 63
Fuel for dredges, cost of.................... 193

G.

Gantry, bow, construction of................ 71
  figure showing......................... 71
Gardner, Charles, acknowledgment to....... 4, 10
Gardner, W. H., on electrical equipment of Yuba No. 14 dredge............. 101, 102
Gears for driving bucket line................ 60
  view of.................................. 60
Gold, in California dredging fields, occurrence of......................... 18
  in placers, determination of............. 181
    distribution of........................ 33
  in dredging ground, recovery of, losses in. 18
  in Montana dredging fields, fineness of.. 22
  losses of, on dredges, causes of.......... 154
    nature of............................ 146
  proportion of recovered by dredges...... 186
  source of................................ 17
  value of from dredges in California...... 7
Gold dredging, development of.............. 12
  features of.............................. 204
  in United States, yield of................ 11
  in Yukon Territory, yield of............. 13
Gold dredges, cost of....................... 13
  development of.......................... 13
  in United States, number of............ 11
Gold grains in California dredging field, size of. 19
  recovered by Conrey dredges, size of..... 144
Gold-saving appliances, on dredges, variety of.............................. 138–149
Gold-saving area on dredge, plan of, figure showing.......................... 183

Page.

Gold-saving tables, efficiency of, factors governing............................ 143
  on Yuba dredges, arrangement of......... 148
  recovery from different parts of........... 182
  view of.................................. 136
  See also Tables, gold saving.
Gravel, compactness of, relation to drill samples.......................... 29
A. E. Graeter, dredge, Mont., description of.. 9
  view of........................... 10
Grease pumps, use of......................... 134
Griffin, Frank, acknowledgment to.......... 4, 10
Griffin, F. W., on gold recovery on Natomas Consolidated dredges.............. 141
Grizzly, construction of...................... 72
  view of................................. 72
Guamos dredge, P. I., fuel costs of.......... 196
Guards for moving parts, use of............ 133

H.

Haley, Charles, acknowledgment to......... 4
Hammon, W. P., acknowledgment to....... 4, 10
Hammondton district, Cal., gold dredging in. 14
Hanford, R. G., acknowledgment to......... 10
High-carbon steel, properties of.............. 92
Hopfield, Louis, acknowledgment to......... 4, 10
Hopkins, E. J., acknowledgment to.......... 4
Hopper, main, elevation of.................. 72
Hull, dredge, construction of................ 68
    dimensions of........................ 70
    repair and maintenance of............ 122
    steel, advantages of.................. 68
    view showing......................... 68
    wooden, life of...................... 69, 121
Hurst, G. L., cited.......................... 201

I.

Idaho, dredging fields of..................... 23
  gold dredges in, number of.............. 11
  gold dredging in........................ 15
  gold from dredges in, value of.......... 11

J.

James, W. H., acknowledgment to........... 4
Janin, Charles, cited............... 176, 191, 193, 201
  on future of gold dredging............... 205
Japin, Hennen, acknowledgment to......... 4
  cited.................................. 172, 173
  on running time of Conrey dredges...... 118
Jig concentrates from Natomas Consolidated dredges, value of................. 140
Jigs, on dredges, arrangement of, view of.... 139
  on dredges, use of...................... 139

K.

Kammerer, Charles, acknowledgment to.... 4, 10
Keys for securing hubs to shafts, use of...... 135
Keystone drill, view of....................... 25

L.

Labor costs on dredges...................... 162
Ladder-hoist winch, construction of.......... 62
Ladders, dredge, rollers on, materials for.... 71

Page.

- Leeson, C. G., acknowledgment to 4
  - on quicksilver losses on Oroville dredges 142
- Lights for dredges 124
- Logs, field and time, types of 40
  - prospecting, keeping of 39
- Loss of gold on dredges, causes of 146

M.

- MacKay, Gordon, acknowledgment to 10
- *Maggie Gibson* dredge, general data on 86 to 89
- *Magpie* dredge, rebuilding of, cost of 109
- Malaguit, P. I., gold-dredging ground in 24
- Mambuloa dredge, P. I., power plant of, details of 197
- Manganese steel, for bucket parts, use of 56
  - for ladder rollers, use of 71
  - for sheaves, use of 84
  - heat treatment of 91
  - properties of 90
  - screen plates of, view of 74
  - use of on dredges 90
- Marysville Gold Dredging Co., delays on dredges of, causes of 128
  - operating costs of 178
- Miles, J. H., cited 121
- Monitors, use of before dredges 120
- Montana, dredge wages in 162
  - dredges in, number of 11
  - dredging fields in 22
    - gold grains in, size of 22
  - early dredging in 5, 6, 8
  - gold from dredges in, value of 11
- Motors, for driving bucket line, design of 93
  - for dredges, design and use of 99
    - power consumption of 94 to 97
  - *See also* Digging motor, electric motors.

N.

- Natomas Consolidated, Cal., dredges of, electric current used on, voltage of 99
    - electric power consumed on 95
    - electrical equipment of, details of 99
    - gold recovery from mills and jigs on 141
    - gold-saving equipment on 139-141
      - operating costs of 164, 174
      - variations in 153
    - operation of, data on 95
    - recovery of gold by, percentage of 186
    - time lost on 128
    - yardage handled by 151
  - operations of 12
  - reclamation of land by 200
- Natoma No. 8 dredge, method of cleaning buckets on 119
  - yardage and costs of 119
- Neill, J. W., cited 139
  - on gold recovery at Snelling, Cal 192
- Neill jig, construction of, figure showing 138
- New South Wales, gold dredging in 12
- New Zealand, gold dredges in, production of 12
- Nickel-chrome steel, for buckets and pins 56, 57
  - for ladder rollers 71
  - properties of 92
  - use of, on dredges 92
- Nickel steel, properties of 92
  - use of on dredges 92

O.

Page.

- Oiling of bearings, facilities for, need of 134
- Oliver, Emery, acknowledgment to 4
- Oregon, gold dredges in, number of 11
  - gold dredging in 16, 23
  - gold from dredges in, value of 11
- Oro Electric Corporation, dredges of, operating costs of 179
- Oroville, Cal., dredge at, recovery of gold by 191
  - operating cost of 153
    - variations in 153
    - quicksilver losses on 142
  - gold dredging near 14
- Oroville Dredging Co., dredges of, gold recovered by, percentage of 187
    - working, costs of 155
- Oroville fields, Cal., description of 19
  - gold grains in, size of 19
  - view of 10

P.

- Pacific Gold Dredging Co., operating costs of 166-196
- Pacific No. 1 dredge, working cost on 155
- Panama Canal, dredges on, operating costs of 160
- Panning of drillings, method of 35
- Paracale, P. I., gold in gravels of 24
- Pato dredge, gold recovered by, percentage of 187
- Pennsylvania dredge, moving of 110
- Perry, O. B., cited 113, 124
- Philippine Islands, dredge fuel in, cost of 193
  - gold dredging areas in 14, 23
    - gold grains from, size of 23
  - gold dredging in 16
    - difficulties of 24
  - future of 207
- Pins for bucket lines, proper size of 57
  - steels used for 57
- Pipe, drill-hole, pulling of 36
- Placer County, Cal., gold dredges in 14
- Placer mining, development of 3
- Placers, averaging of samples from, danger in 46
  - drill holes in, arrangement of, figure showing 47, 48
  - drillings from, quantities to be taken, factors determining 43
  - gold content of, calculation of 47, 48
    - determination of from drill holes 44
    - proportion of recovered by dredges 186
    - source of 17
  - prospecting of, with drills, cost of 42
    - errors in 181
  - sampling of, skill required in 180
  - size of gold grains in 18, 19
  - *See also* Dredges, dredging ground, and dredging districts named.
- Pontoons for insulated cables, view showing 99
- Postlethwaite dredge designs, early use of 7
- Power plants for dredges, fuel consumption of 193
- Prospect holes, placing of, factors determining 43
  - *See also* Drill holes, drills, placers.
- Prospecting, cost of 30, 41, 42
  - drills for, view of 24
  - errors in 188
  - features to be determined in 27
  - methods of 26
    - at Snelling, Cal 192
  - need of 26
- Prospecting records, forms for 39

Page.
Prospecting shafts, advantages of 27
  sinking of 23
Pumpings from drill hole, handling of 37
Pump valves, desirable types of 136
Pumps, auxiliary for fire protection, use of 136
  capacity of 131
  centrifugal, power consumption of 94
    use of 94

Q.

Quicksilver, losses of, on Oroville dredges 142

R.

Ransome, F. L., cited 188
Reclamation of dredging ground 198
  special dredge construction for 203
Records, prospecting, forms for 39
  keeping of 38
Redredging, gold recoveries in 189
Repairs on dredges, methods of making 124
Resoiling of dredging ground, methods of 210
Richards, R. H., cited 139
Riffles for saving quicksilver, use of 149
Riffles for sluices, types of 148
Rivets for buckets, proper size of 56
Rocker, construction of 37
  details of, figure showing 36
  use of 35
Rollers, for conveyor belts, design of 82

S.

Sacramento County, Cal., gold dredges in 14
"Safety-First" rules for dredges 133
Salting of samples, precautions against 28
Sampling of placer deposits, skill required in 180
Sand pump, use of 32, 34
Save-all, gold recovery by 44
Save-all sluices, construction of 72
  view of 71
Screen, revolving, cylindrical, advantages of 74
    driving gears for, view of 72, 74
    figure showing 24
    peripheral speed of 67
    sizes of 74
    step type, advantages of 74
    view of 73
  housing for, views of 74
  idler rollers for, placing of 75
  structure supporting, view of 73
    use of steel for, need of 69
  washing of gravel in, methods of 75
Screen plates, materials for 74
Scott River dredge, rebuilding of, cost of 108
Searchlights for dredges, advantages of 124
Seward Peninsula, Alaska, dredges in, operating cost of 195
    fuel consumption of 193
  drilling of placer ground in, cost of 53
Shafer, N. S., acknowledgment to 10
Shafts, prospecting, advantages of 27
    cost of 30
    size of 29
  See also Prospecting shafts.
Shasta County, Cal., gold dredging in 15
Shepard, W. M., on electrical equipment of Yuba No. 14 dredge 101, 102
Sibbett, G. E., acknowledgment to 4
Siberia, future of gold dredging in 206
Siskiyou County, Cal., gold dredging in 14
Summers, M. L., acknowledgment to 4
Sluices, save-all, construction of 72
    view of 72
Sockets for wire ropes, proper method of forming 84
Spoon dredge, description of 5
Spud, large, transportation of, view showing 66
Spuds, breakage of, cause of 66
  digging on, method of 114, 115
  increasing use of 114
  lengths of 67
  rectangular, construction of 66
    figures showing, sections of 66
  round point, view of 65
  single, disadvantages of 67
  types of 65
Stacker, for tailings, construction of 78
  winch, view of 78
Stacker belt, drive for 82
  idlers for, placing of 82
  pulley for, dimensions of 81
  silent chain drive for, advantage of 83
  view of 76
  See also Belts, conveyor belts, tailings.
Steam dredge, first use of 6
  view of 6
Steam power plant, novel 196
Steam shovel, use of 5
  See also Manganese steel, nickel steel, nickel-chrome steel, and high-carbon steel.
Steels, special, for dredge parts 90-93
Summers, M. L., on rate of digging 117
Swing winch for dredge, construction and use of 63, 64

T.

Tables, gold saving, cleaning of 147
  proportion of material handled on 131
  types of 141
Tailing, gold in, amount of 184, 189
  stacker for, use of 78
    for land reclamation, view of 202
Tin Cup dredge, rebuilding of, cost of 107
Tin dredging, improvement of dredges for 207
Timbers, dredge, preservation of 69
Time losses on dredges, causes of 127
Threee Friends dredge, operating cost of 194
Thunen, E. G., acknowledgment to 4
  on electric motors for dredges 98-100
Tongkah Harbour Tin Dredging Co., Siam, dredging costs of 151
Tonopah Placers Co., dredges of, operating costs of 177
Transformers on dredges, housing of 136
Trinity County, Cal., gold dredging in 15
Tubs for panning samples 37
Tumblers, bearings and shafts for 58, 59
  of bucket ladder, construction of 57

Page.
Tumbler shaft, construction of, view showing ... 58
proper size of ... 58
Turbo-generator, as power plant, details of... 196

U.

Undercurrents on dredges, use of ... 143, 144
figure showing ... 145
United States, gold dredging districts in ... 13

V.

Victoria, Australia, gold dredging in ... 12
Viloro dredge, wreck of, view of ... 136
Voltage of electric current on dredges ... 99-103

W.

Wages of dredge men, rates of ... 162
Water consumption of dredges ... 130, 131
Water rheostat, Conrey No. 4 dredge, arrangement of ... 97
Winch, ladder-hoist, friction clutches for ... 63
view of ... 62
Winch, swing, types of ... 63
view of ... 63
Winches for dredges, views of ... 62
Winchmen, duties of ... 116
regulation of bucket swing by ... 116
Winston, W. B., cited ... 120
Wire rope on dredges, grade and size of ... 84
life of ... 85
sheaves for, dimensions of ... 83
Wood for dredge hulls, creosoting of, cost of.. 69
Wooden hulls, decay of, causes of ... 70
life of ... 121

Y. Page.

Yale, Charles, cited ... 198
Yardage dug, methods of estimating ... 151
Yosemite dredge, tables on, plan of, figure showing ... 140
Yosemite & Mining Co., Cal., recovery by dredge of ... 192
Yuba Consolidated, Cal., dredges, consumption of quicksilver on ... 143
gold recovered by, percentage of ... 187
operating costs of ... 161
operations of ... 14
time lost on ... 128
Yuba County, Cal., new dredges in ... 14
Yuba dredges, changing buckets on, method of ... 125
electric current used on, voltage of ... 99
gold-saving methods on ... 148
plan of, figure showing ... 105
sizes and uses of wire rope on ... 85
Yuba field, Cal., description of ... 20
view in ... 20
Yuba Gold Fields dredge, plan of, figure showing ... 79
Yuba No. 14 dredge, electrical equipment of.. 101
motors for, power consumption of ... 93
Yuba No. 15 dredge, view of ... 98
Yuba River, Cal., early dredge on, description of ... 8
gold dredging on ... 14
Yukon Territory, first dredges in ... 13
dredging in, difficulties of ... 25
gold production of dredges in ... 13
placer deposits of ... 24, 25
repairing of dredges in ... 124

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Bulletin 128 **Mineral Technology 18**

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY
BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

REFINING AND UTILIZATION OF GEORGIA KAOLINS

BY

IRA E. SPROAT

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CONTENTS.

Page.

Preface.......... 5

Introduction.......... 7

Secondary kaolins of Georgia and South Carolina.......... 7
- Principal deposits of secondary kaolins.......... 8
  - Kaolin mining in the Aiken district, South Carolina.......... 8
    - McNamee & Co.......... 8
    - South Carolina Clay Co.......... 8
    - Paragon Kaolin Co.......... 9
    - Immaculate Kaolin Co.......... 9
    - Peerless Clay Co.......... 9
  - Kaolin mining in the Dry Branch district, Georgia.......... 9
    - Georgia Kaolin Co.......... 9
    - American Clay Co.......... 9
    - John Sant Clay Co.......... 10
    - Atlanta Mining & Clay Co.......... 10
- Probable causes of variation in physical properties of the clays.......... 10

Mining methods and treatment.......... 11
- Kaolin mining in the Dry Branch district.......... 11
  - Removal of overburden.......... 11
  - Mining the clay.......... 11
  - Removing clay from pit.......... 12
    - John Sant Clay Co. mine.......... 12
    - American Clay Co. mine.......... 12
    - Georgia Kaolin Co. mine.......... 13
- Kaolin refining in the Dry Branch district.......... 13
- Impurities in Georgia kaolins.......... 14

Effects of electrolytes on clay bodies.......... 15
- Literature.......... 15
- Experiments with kaolins of South Carolina and Georgia.......... 17
  - Preliminary investigation.......... 17
  - Purpose of present tests.......... 18
  - Effect of alkalies, acids, and salts on Georgia kaolin slip.......... 18
    - Effect on viscosity of slip.......... 18
    - Effect upon sedimentation.......... 20
  - Control of the viscosity of clay slip.......... 20
    - Factors affecting the initial viscosity.......... 21
      - Varying clay content.......... 21
      - Degree of blunging.......... 22
      - Predrying.......... 23
      - Use of alum.......... 23
    - The effect of varying viscosity on capacity of washing plant.......... 23
    - Factors affecting minimum viscosity.......... 24
      - Degree of blunging.......... 24
      - Predrying.......... 25
      - Use of alum.......... 25

Effects of electrolytes on clay bodies—Continued.

Experiments with kaolins of South Carolina and Georgia—Continued.

Page.
Refining kaolin by technical control of disperse system.............. 26
Sedimentation process.. 26
The elutriation or sliming process............................. 27
Storage and disintegration..................................... 27
Feeding.. 28
Blurging and dispersion.. 28
Types of washers... 29
Addition of caustic soda....................................... 29
Strength of caustic-soda solution.............................. 31
Sliming.. 31
Coagulation.. 31
Screening.. 33
Concentration.. 33
Filter-pressing.. 33
Drying... 35
Crushing... 35
Layout of plant.. 35
Extra cost of the caustic soda sulphuric acid treatment.............. 35
Mechanical equipment... 36
Chemicals.. 36
Chemist.. 36
Conclusions drawn from results of tests.............................. 36
Utilization of refined Georgia kaolin..................................... 37
Preliminary tests to determine physical properties and color............ 37
Shrinkage.. 37
Color in white-ware bodies... 38
Materials used in making bodies................................ 38
Composition of the bodies used................................. 39
Preparation of the bodies...................................... 39
Preparation of test pieces..................................... 39
Biscuit burn... 39
Gloss burn... 40
Results of tests... 40
Test pieces biscuit-fired at cone 12 and gloss-fired at cone 3.. 40
Test pieces biscuit-fired at cone 4 and gloss-fired at cone 3... 40
Conclusions as to results of color tests............................. 40
Tests in making vitreous china.. 41
Procedure in tests... 41
Materials used... 41
Composition of bodies.. 41
Preparation of bodies.. 43
Biscuit burn... 43
Glazing.. 43
Gloss burn... 44
Results of tests... 44
Series A tests.. 44
Conclusions from results of series A........................ 44
Series B tests.. 45
Conclusions from results of series B........................ 45
Series C tests.. 45
Conclusions from results of series C........................ 45
General conclusions as to practicability of making china from Georgia clays.. 46

Utilization of refined Georgia kaolin—Continued.

Page.
Manufacture of wall tile.......... 46
  Preliminary tests.......... 46
    Materials used.......... 47
    Composition of bodies.......... 47
    Preparation of bodies.......... 48
    Making the tile.......... 48
    Biscuit burn.......... 48
    Gloss burn.......... 49
    Vitrification.......... 49
    Determination of color.......... 50
    Results of tests.......... 50
  Large-scale tests.......... 50
    Materials.......... 50
    Composition of bodies.......... 51
    Preparation of bodies.......... 51
    Making the tile.......... 52
    Burning 52
    Results of tests.......... 52
Applicability of refined Georgia kaolin in wall-tile bodies.......... 53
Acknowledgment.......... 53
Publications on mineral technology.......... 54
Index.......... 57

TABLES.

Page.
Table 1. Results of mechanical analyses of washed Georgia kaolin.......... 18
2. Percentages of clay and water in slip during one month of operation. 21
3. Results of analyses of materials used in tests.......... 38
4. Composition of bodies used.......... 39
5. Composition and physical properties of bodies used in tests.......... 42
6. Results of mechanical analyses of feldspar and flint.......... 47
7. Composition and physical properties of bodies used in tests.......... 48
8. Composition and physical properties of bodies used in large-scale tests.......... 51

ILLUSTRATIONS.

Page.
Plate I. *A*, View in open-cut kaolin mine in the Dry Branch district, Ga., showing overhead tramway; *B*, View in another kaolin mine, Dry Branch district; *C*, Removing overburden with steam shovel.......... 10
II. *A*, Removing overburden with drag scraper; *B*, View in open-cut kaolin mine, showing irregular jointing of clay; *C*, Another view showing jointing of clay.......... 12
III. *A*, Open-air drying sheds for crude clay; *B*, Open-air drying sheds for washed clay; *C*, Kaolin refining plant near Dry Branch, Ga. 14
IV. *A*, Disk clay feeder; *B*, Concrete washer or blunger.......... 28
V. Plan of washing plant.......... 36

Page.
FIGURE 1. Cross section of strata in a kaolin mine in the Dry Branch district.. 11
2. Effect of electrolytes on viscosity of slip containing 20 per cent of Georgia kaolin.... 19
3. Effects of moisture in clay used, blunging, and additions of alum, on Georgia kaolin slip.... 22
4. Effect of caustic soda on viscosity of Georgia kaolin slips under dry-weather conditions.... 24
5. Effect of caustic soda on viscosity of Georgia kaolin slips under wet-weather conditions.... 25
6. Effect of caustic soda on viscosity of Georgia kaolin slips blunged in ball mill.... 26
7. Effect of caustic soda on Georgia kaolin slips made from predried clay.... 27
8. Effect of caustic soda on Georgia kaolin slips containing 0.03 per cent of alum.... 28
9. Design of concrete washer.... 30
10. Design of floating troughs.... 32
11. Design of concentration tank.... 34

PREFACE.

For years American manufacturers of tile and white ware have been dependent on the importation of foreign kaolin, chiefly in the form of English china clay, for their high-grade material. In 1916, for example, 229,093 tons of such kaolin were imported, valued at $1,377,106, at the port from which it was shipped. At the same time there were known to be available in Georgia and South Carolina large quantities of kaolin of a high degree of purity, except that it contained certain color-giving minerals, especially iron and titanium compounds, and had an abnormal shrinkage. Heretofore this material has been available only for the paper trade and for some lower grades of pottery. Investigations carried on by the division of mineral technology of the Bureau of Mines have shown that the impurities may be readily and cheaply separated and the kaolin so modified in its physical characteristics, both as to shrinkage and to the development of color on burning, that it is comparable with the best English fire clay for many pottery purposes.

By the addition of a small amount of sodium hydroxide, this amount being carefully controlled by chemical analysis, the kaolin colloids may be so deflocculated that the clay remains in suspension in water through long periods of time, allowing the small particles of heavier minerals to settle out readily. On neutralizing afterwards with sulphuric acid, the clay readily settles and may then be filtered and dried as usual.

As a result of laboratory experiments, cooperative experiments were conducted with the Georgia Kaolin Co. at Dry Branch, Ga., and a plant built in accordance with the facts discovered in the laboratory. As a result, many tons of kaolin have been purified at a cost of less than 50 cents a ton, producing thereby a material which has a market value nearly twice that which could be obtained for the kaolin for the purposes to which it was previously put. The purified china clay produced has been further investigated in potteries on an extensive scale through the cooperation of the Zanesville Mosaic Tile Co., the Mayer China Co., and the Beaver Falls Art Tile Co. It has been shown that in vitreous chinaware this purified American kaolin can be successfully substituted for all of the ball clay heretofore used and for at least 50 per cent of the English china clay. Better still, in the tile industry whiter and stronger tile can be and are being made from this treated Georgia kaolin and American feldspar by substituting them entirely for the English china clay and Cornwall stone heretofore used.

America should, accordingly, be largely independent, and, with further experiments, probably entirely independent of imports of foreign material for the white-ware industries. The details of the work are described in the following pages.

Charles L. Parsons,
Chief, Division of Mineral Technology.

REFINING AND UTILIZATION OF GEORGIA KAOLINS.

By Ira E. Sproat.

INTRODUCTION.

The question "Why can not American kaolins be substituted for English china clay?" has been asked time and time again, and more often since the beginning of the present war in Europe, which has threatened to cut off the supply of European clays imported from abroad. The answer to this question has invariably been, that the domestic kaolins are not as pure and uniform in composition as the imported clays.

This demand for white-burning American clays is of such economic importance that it was considered advisable to determine whether some of our vast deposits of impure white clays could not be refined sufficiently to permit their being substituted for the foreign materials.

Up to the present time mechanical principles only have been applied in the refining of kaolins, but in order to keep pace with the increasing requirements for better quality and uniformity of product, the application of the principles of colloidal chemistry is necessary.

The investigation described in this report was carried on to determine the practicability of applying technical control of clay disperse systems to the refining of kaolins and the utilization of the prepared clay in the manufacture of vitreous china and wall tile.

It is hoped that this report will prove a stimulus to the clay-washing industry; will point out a method of refining certain American clays, and will lead to the substitution of domestic kaolins for imported china clays by manufacturers of high-grade ceramic wares.

SECONDARY KAOLINS OF GEORGIA AND SOUTH CAROLINA.

Immense deposits of secondary kaolins underlie certain areas in Bibb, Twiggs, Wilkinson, Washington, Glasscock, Jefferson, and Richmond Counties, in Georgia; and Aiken, Edgefield, Lexington, and Kershaw Counties, in South Carolina. As these sedimentary kaolins in the Cretaceous strata of Georgia and South Carolina occur in large deposits which run very high in clay substance, and in some of their physical properties are not unlike the English china clay, it was thought advisable to select these kaolins as a basis for the experiments.

PRINCIPAL DEPOSITS OF SECONDARY KAOLINS.

A general study of the principal kaolin deposits of South Carolina and Georgia was made by Robert Back, formerly of the Bureau of Mines, in 1913, under the direction of C. L. Parsons, chief mineral technologist. The following description of the nature of the different kaolin deposits of Georgia and South Carolina is taken from Mr. Back's field notes, supplemented by the writer's observations in the Dry Branch district of Georgia.

There are two great areas; the Aiken district, in Aiken County, South Carolina, and the Dry Branch district, in Bibb and Twiggs Counties, Georgia, which are being worked extensively to-day. There are a number of other deposits both in Georgia and South Carolina which in quality and extent rival those of the Dry Branch and Aiken areas, but they are located farther from railroads and have not been worked to any great extent. The deposits in Georgia have been described in detail by Veatch,[a] and those of South Carolina by Sloan.[b]

KAOLIN MINING IN THE AIKEN DISTRICT, SOUTH CAROLINA

Kaolin is mined in the Aiken district from open cuts with gravity drainage or from pits which are drained by steam pumps. A number of kaolin mines have been opened in this district, chief of which are those of McNamee & Co., South Carolina Clay Co., Paragon Kaolin Co., Immaculate Kaolin Co., and Peerless Clay Co.

MCNAMEE & CO.

The property of McNamee & Co. is situated about one mile from Bath and near Horse Creek. The kaolin is mined by open cuts having gravity drainage. The overburden, which is about 30 feet in depth, is removed with a steam shovel. The clay bed is known to attain in places a thickness of about 15 feet, the upper 2 feet and lower 7 feet of which are full of grit and sand and are thrown away. This clay is simply air-dried and sold to the paper trade.

SOUTH CAROLINA CLAY CO.

The South Carolina Clay Co. operates an open-cut mine near Horse Creek and not far from that of McNamee & Co. The kaolin deposit is about 8 feet thick and the overburden ranges from 20 to

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[a] Veatch, Otto, Second report on clay deposits of Georgia: Bull. 18, State Geol. Survey, 1909, 453 pp.

[b] Sloan, Earle, A preliminary report on the clays of South Carolina: Bull. 1, ser. 4, State Geol. Survey, 1904, 171 pp.

40 feet thick. The top 8 to 10 inches of kaolin, which is badly stained with iron, is thrown away, and the underlying kaolin, yellowish in color, is dried in the open air and sold to the pigment and crayon trades.

PARAGON KAOLIN CO.

The mines of the Paragon Kaolin Co. are about 1½ miles from Langley Station. This company is operating two open-cut mines with a daily capacity of about 100 tons. The clay bed is about 12 feet thick. The overburden, which varies in thickness from 30 to 40 feet, is removed with mules and carts. The kaolin obtained from these two mines is rather soft and mealy, and the output is sold exclusively to the paper trade.

IMMACULATE KAOLIN CO.

The mine of the Immaculate Kaolin Co. is located about 2 miles southeast of Langley Station. The clay deposit in this mine reaches a thickness of about 25 feet and is very white and free from grit, especially in the lower part of the bed. The overburden, which reaches a thickness of 40 feet, is removed with a cable excavator. The clay is sold directly and exclusively to the paper trade.

PEERLESS CLAY CO.

The Peerless Clay Co. operates a mine 2 miles southeast of Langley Station, the clay deposit being very similar to that mined by the Immaculate Kaolin Co.

KAOLIN MINING IN THE DRY BRANCH DISTRICT, GEORGIA.

In March, 1916, there were three companies actively engaged in the mining and marketing of secondary kaolins within a radius of three miles of Dry Branch. The mines are all open-cut mines.

GEORGIA KAOLIN CO.

The Georgia Kaolin Co. is the only one in this district actively engaged in washing kaolin. The clay as mined is highly plastic, has a high strength, and burns to a good white, but if glazed the color becomes a decided cream. The product is sold to the paper trade, pigment manufacturers, and some ceramic industries. A view of a face in this mine is shown in Plate I, *A*.

AMERICAN CLAY CO.

The American Clay Co. sells its entire output to the paper trade. The clay is not washed and the only mechanical treatment it receives is crushing after being air-dried. A washing and refining plant is being built. This clay has a good plasticity and burns to about the same color as the clay mined by the Georgia company, but is softer and more mealy and has a lower tensile strength. A view of the claybank in this mine is shown in Plate I, *B*.

JOHN SANT CLAY CO.

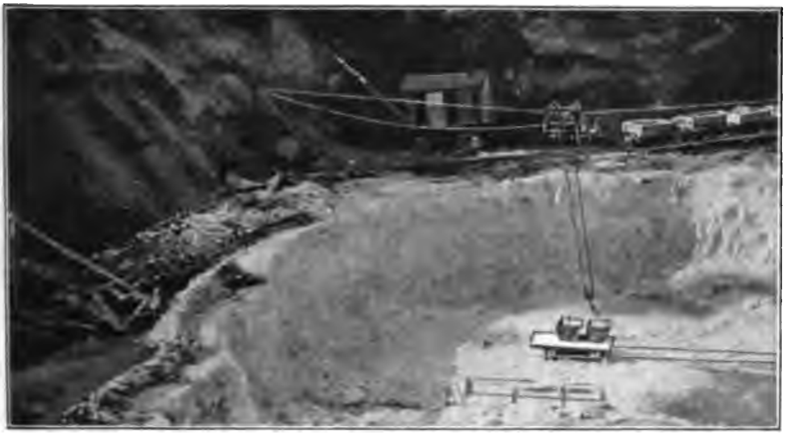
The John Sant Clay Co. mines and pulverizes its clay for use in the manufacture of white-ware pottery, wall tile, and other ceramic wares. The clay from this mine is very hard, fine grained, and burns to a good color, but lacks uniformity in composition, which is partly due to impurities. If this clay were to be refined it would doubtless find a much wider application in the ceramic industries.

ATLANTA MINING & CLAY CO.

The Atlanta Mining & Clay Co. formerly operated a mine and washing plant in this district. The property has been idle for the past year or more.

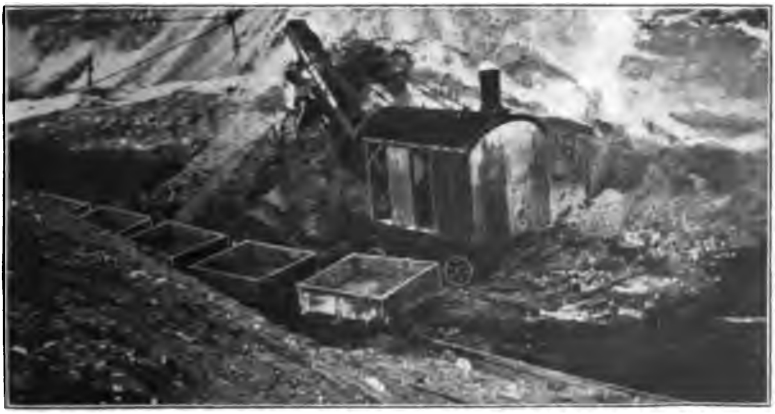
PROBABLE CAUSES OF VARIATION IN PHYSICAL PROPERTIES OF THE CLAYS.

The variation in physical properties of kaolin from different mines, as noted above, is evidently due to the variable conditions under which the deposits were laid down, the denser and harder clays having been formed under conditions of complete deflocculation and disintegration of clay grains, whereas the clays of lesser density and tensile strength have probably been deposited while the grains were in a more flocculated and coagulated condition. In other words, the extent of dispersion at the time of deposition has affected the physical properties of the clay. As a result the clay from one of the mines mentioned is hard and weak and must be finely ground before it will blunge satisfactorily with water; another mine produces soft mealy clay which is easily blunged with water without any preliminary treatment; and still another mine yields a soft, highly plastic, and strong clay which contains irregular layers or pockets of fine-grained flinty clay of low tensile strength, that is so hard that much of it must be thrown away, as it will not blunge in the washers. The harder clays contain less impurities than the softer ones, indicating that the ancient ocean or sea, which acted as a huge washing plant in refining and sorting these sediments, deflocculated some clays and caused the impurities to settle out more readily.



A. VIEW IN OPEN-CUT KAOLIN MINE IN THE DRY BRANCH DISTRICT, GA., SHOWING OVERHEAD TRAMWAY.

B. VIEW IN ANOTHER KAOLIN MINE, DRY BRANCH DISTRICT.



C. REMOVING OVERBURDEN WITH STEAM SHOVEL.

MINING METHODS AND TREATMENT.

KAOLIN MINING IN THE DRY BRANCH DISTRICT.

Open-cut mining is universally practiced at the kaolin mines in the Dry Branch district, owing to the fact that the overburden is so soft that any timbering would have to be water-tight in order to keep out sand and iron-stained silt in rainy weather.

REMOVAL OF OVERBURDEN.

Figure 1 shows graphically the strata in the mine of the Georgia Kaolin Co. and is typical of the strata generally found in mines of this district. However, the thickness of overburden will vary from about 10 feet along the ravines to probably 100 feet below the crest of the hill.

The removal of this overburden is an important part of the mining operations. To remove such large amounts of material economically and with dispatch and dispose of them has been one of the main problems. However, until lately not much attention has been given it.

At present (March, 1916) two of the companies mentioned are using steam shovels for removing the overburden, and it will probably be only a short time until the overburden is stripped back far enough (200 feet) from the clay face or breast to avoid contamination in case of caving during rainy weather. A steam shovel at work is shown in Plate I, *C*. Another company removes the overburden with pick and shovel, removing just enough to expose the face of the clay bank, with the result that sometimes after a rain the overburden caves, filling the pit and staining the clay. The drag scraper which was formerly used by two of the companies is shown in Plate II, *A*.

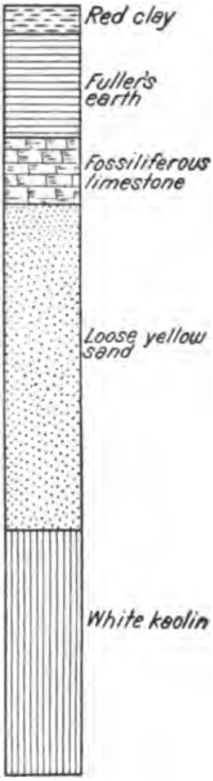


FIGURE 1.—Cross section of strata in a kaolin mine in the Dry Branch district.

MINING THE CLAY.

In the three mines mentioned the kaolin is mined with pick and shovel, the clay being mined economically and easily in this way, owing to the jointing. Veatch [a] in describing the mining of kaolin at the mine of the Georgia Kaolin Co., says:

> A structural feature which has an economic application is the jointing. The joints have no definite system or fixed direction, and may be oblique, vertical, or horizontal,

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[a] Veatch, Otto, Second report on the clay deposits of Georgia: Bull. 18, State Geol. Survey, 1909, p. 127.

and the bed may be minutely jointed or there may be only large cracks at wide intervals. The jointing has the appearance of being due to shrinkage of the clay mass, and not to orogenic movements.

By undermining the clay at the base of the face or breast and driving long wooden or steel stakes down into the clay near the edge of the breast, large chunks are split off, advantage being taken of the jointing of the clay. The clay split down from the bank is broken into lumps of a size that can be conveniently loaded. In the mines of the American Clay Co. and John Sant Clay Co., where the clay is not given a subsequent treatment other than grinding, these small lumps are hand sorted and the iron stains and sand pockets are cut out with hoe-shaped knives. The irregular jointing of kaolin in two different mines is shown in Plate II, *B* and *C*.

REMOVING CLAY FROM PIT.

Each company uses different methods in removing the mined clay from the pit, the method used depending upon the output of the mine, the topography of the surface, and the subsequent treatment of the clay.

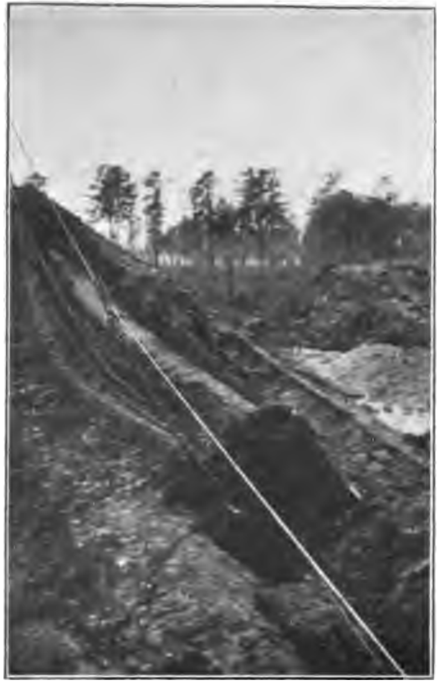
JOHN SANT CLAY CO. MINE.

The mine of the John Sant Clay Co. is located about 1 mile from their drying sheds, which are situated along the Macon, Dublin & Savannah Railroad, and the output is small. The clay is shoveled into wagons and hauled to the open-air drying sheds where it is allowed to dry for several weeks. When dry the clay is ground in a set of corrugated rolls, screened, and sacked for shipment to the pottery trade.

The drying sheds (See Pl. III, *A*) consist of three to six tiers of superimposed racks about 15 feet wide, the clay being piled on the racks to a depth of about 3 feet. The bottom of the racks is formed of movable scantlings or pine railings. As the clay on the lowest tier becomes dry it is dropped on the floor, and the loads on each superimposed tier are dropped in succession. The dried kaolin dumped on the floor is broken with mauls or in roll crushers.

AMERICAN CLAY CO. MINE.

At the mine of the American Clay Co. the clay is elevated out of the pit with an overhead tramway which transports and dumps the clay into small loading bins at the top of the pit. The clay is then dumped into cars, which are drawn from the loading bins to the open-air drying sheds by a wire cable and winding drum. The clay is allowed to dry for several weeks, when it is shaken down to the bottom of the shed, hammered and cut up into small lumps about the size of a man's fist, loaded into cars, and shipped exclusively to the paper trade.



A. REMOVING OVERBURDEN WITH DRAG SCRAPER.

B. VIEW IN OPEN-CUT KAOLIN MINE, SHOWING IRREGULAR JOINTING OF CLAY.

C. ANOTHER VIEW SHOWING JOINTING OF CLAY.

GEORGIA KAOLIN CO. MINE.

In the mine of the Georgia Kaolin Co. the clay is merely broken in pieces small enough to be loaded by two men, as the subsequent treatment of the clay does not require breaking into small pieces at this stage. The clay is elevated from the pit by an overhead tramway or by cars which are pulled out of the pit by a steel rope and winding drum. Both the cars and buckets automatically dump their clay into a large storage shed, which holds enough clay to run the washing plant for a week.

KAOLIN REFINING IN THE DRY BRANCH DISTRICT.

Only one company, the Georgia Kaolin Co., in this district is actively engaged in the refining of kaolin. The method of washing clay at this plant is similar to the refining process used in refining the primary kaolin of the southern Appalachian district, which has been described in detail by Watts,[a] so only a mere outline of the washing process will be given here. It consists of seven different operations, which are as follows:

1. Thorough blunging of the clay with water, in rectangular blungers, which are about 10 feet long, 4 feet wide, and 4 feet deep.
2. Floating the slip through narrow troughs about 50 feet long with an average width of about 2 feet, the width of trough depending on the rate of flow required to settle out the impurities.
3. Screening the slip through a 110-mesh rotary screen.
4. Concentrating the slip in large concrete tanks, which are about 75 feet long, 25 feet wide, and 5 feet deep, where it is allowed to stand for about 48 hours, when the supernatant liquid is drawn off and the thickened clay slip run into a small retaining tank.
5. Filter pressing the thick clay slip, which is pumped from the small retaining tank into the filter press at a pressure of 100 pounds.
6. Drying the filter-press cakes, which contain about 25 per cent of water, in tunnel driers heated by exhaust steam, or in open-air drying sheds.
7. Crushing the thoroughly dried cakes into small pieces in corrugated rolls.

Open-air driers for drying the filter-press cakes are shown in Plate III, *B*. A view of the refining plant is shown in Plate III, *C*.

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[a] Watts A. S., Mining and treatment of feldspar and kaolin in the southern Appalachian region: Bull. 53, Bureau of Mines, 1913, p. 72.

IMPURITIES IN GEORGIA KAOLINS.

The following is a chemical analysis of a sample of kaolin taken from the pit of the Georgia Kaolin Co., as reported by S. W. McCallie:

Composition of sample of kaolin.[a]

	Per cent.
Moisture at 100° C	1.22
Loss on ignition	13.46
SiO_2	44.76
Al_2O_3	38.41
Fe_2O_3	.63
CaO	.20
MgO	.09
Na_2O	.09
K_2O	.35
TiO_2	1.37

From this analysis it is seen that the chief color-imparting oxides are Fe_2O_3 and TiO_2. The source of these oxides is best shown by the following results of a microscopic examination of a sample of clay from the mine of the Georgia Kaolin Co., by F. B. Laney, of the United States Geological Survey:

Associate minerals found by microscopic examination of kaolin.

Quartz.
*Partly decomposed feldspar, often stained with iron oxide.
*Wad—small pisolites of manganese oxide.
*Limonite.
Muscovite.
*Magnetite.
*Hematite.
*Ilmenite.
Zircon.
*Rutile.
Apatite.
*Tourmaline.
Corundum.
Monazite.
*Iron—minute particles broken off from the crushing machinery.

The minerals marked with an asterisk are the minerals that tend to give a cream color to a burned china body when Georgia kaolin is one of the constituents of the body.

Only a small part of these impurities is removed by the present methods of refining, owing to the fact that the most of the color-imparting minerals are present in a very fine state of subdivision, as the coarser impurities settled out when the clay was being deposited. In fact, most of the color-imparting particles will pass a 260-mesh sieve, and some are much finer. Besides being finely subdivided, these impurities are completely coated with clay, which prevents their being settled out, regardless of their higher specific gravity, without a large loss of clay. This clay coating is not removed to any great extent on prolonged blunging. Blunging for two hours will not disperse the clay particles. Grinding in a ball mill has a greater

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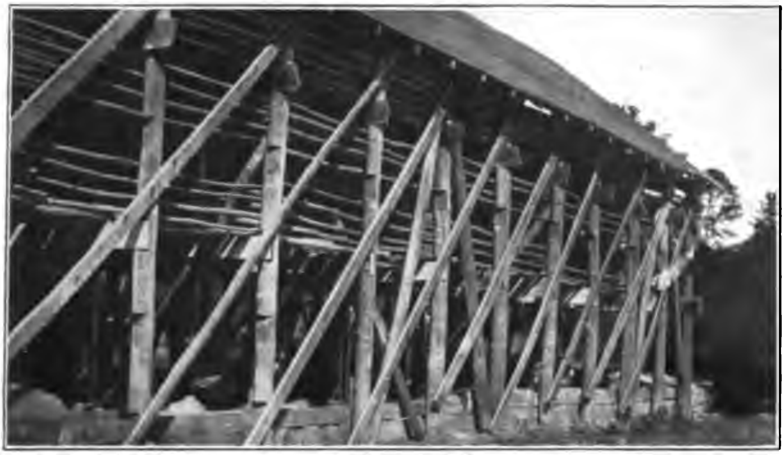
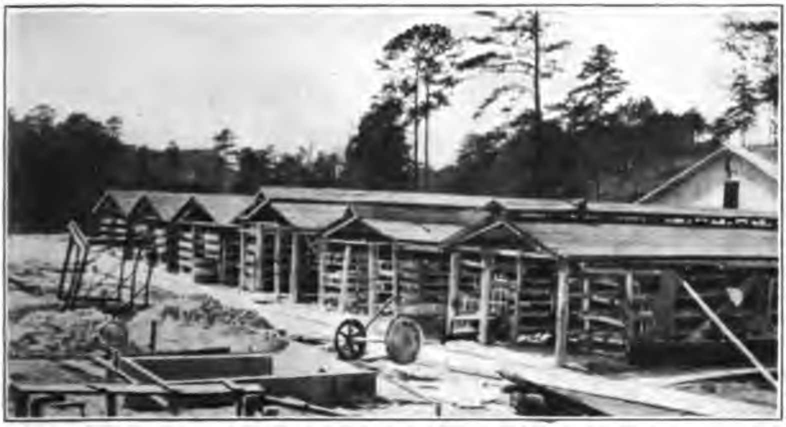
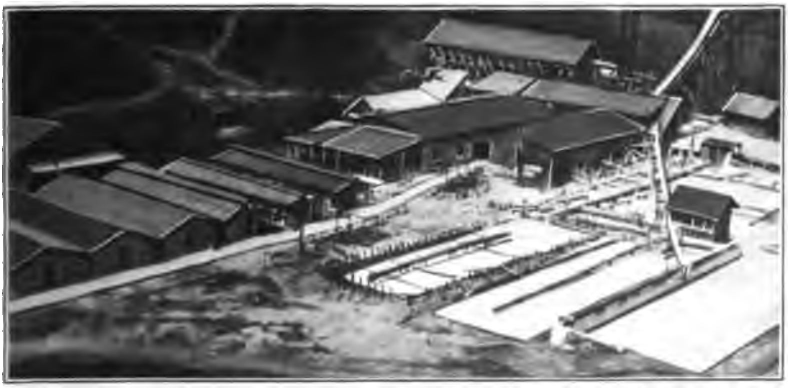
[a] Veatch, Otto, Second report on clay deposits of Georgia: Bull. 18, State Geol. Survey, 1909, p. 128.



A. OPEN-AIR DRYING SHEDS FOR CRUDE CLAY.



B. OPEN-AIR DRYING SHEDS FOR WASHED CLAY.



C. KAOLIN REFINING PLANT NEAR DRY BRANCH, GA.

tendency to produce complete disintegration, but such treatment is not economical to use in the washing of kaolins. Therefore, in order to obtain complete dispersion of the clay grains and the resulting separation of the coloring minerals from the clay it is necessary to consider the colloidal nature of clay, and apply chemical principles thereto.

EFFECTS OF ELECTROLYTES ON CLAY BODIES.

LITERATURE.

Probably one of the first persons who noticed the effect of alkalies on clay was Brongniart,[a] who in 1844 claimed that adding 3 per cent of potash improved the pottery bodies with which he worked.

Shortly after this, Hartmann [b] described the use of alkalies in casting.

Seger [c] also noticed some effects of alkalies in ceramic bodies.

Zebisch,[d] who carried on one of the first scientific investigations of the effect of bases on clays, noticed that basic substances have an angular sharp-cornered form while in a solution of water, and that acid substances have a round smooth form. He offers this as a possible explanation of the effect of bases on clays.

Goetz [e] in his German patent advises the use of soda with or without cinnabar in the casting of porcelain and stoneware.

In 1895 Hagedorn[f] claimed that a mixture of clay, chalk, quartz, and ground glass made a body requiring a smaller amount of water than certain other ceramic mixtures. This is probably due to the fact that alkalies dissolved from the ground glass and caused deflocculation of the clay.

Kosmann [g] claims that a certain amount of silicic acid is dissolved from the clay substance by soda, and that the silicic acid thus dissolved causes the clay slip to become more liquid.

New developments in colloidal chemistry about this time helped Lottermoser[h] to throw light on some of the present-day views of the effect of alkalies on colloidal gels.

Weber [i] claims in his patent that by using a plastic bond clay in a body together with soda, ammonia, potash, sodium silicate, molasses, and soap it is possible to cast with nonplastic ingredients a body as thick as 10 mm. in diameter. He states that such a slip will cast as well as the finer grained porcelain and stoneware slips.

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[a] Brongniart, A., Traité des arts céramiques ou des poteries, 1844, pp. 468–469.

[b] Hartmann, C., Die Tonwarenfabrikation (1850): Sprechsaal, Jahrg. 27, 1894, p. 1028.

[c] Seger, Hermann, Japanisches Porzellan und dessen Dekoration: Tonindustrie Ztg., Jahrg. 15, 1891, p. 813; Seger's Gesammelte Schriften, 1897, p. 573.

[d] Zebisch, A., Sprechsaal, Jahrg. 27, 1894, p. 105; Jahrg. 28, 1895, p. 303.

[e] Goetz, Karl, German patent 76247, 80b, Oct. 23, 1891; Ueber Sodaschlicher, Sprechsaal, Jahrg. 27, 1894, pp. 637, 685, 738.

[f] Hagedorn, Robert, and Hagedorn, Hermann, German patent 90697, Oct. 23, 1895.

[g] Kosmann, Dr., Ueber die Verflüssigung von Thonbrei durch Soda: Tonindustrie Ztg., Jahrg. 19, 1895, p. 382.

[h] Lottermoser, Alfred, Ueber anorganische Colloïde: Sammlung Chem. und chem.-techn. Vorträge, Bd. 6, 1901, pp. 161–240.

[i] Weber, E., German patent 158496, 80b, Aug. 14, 1902.

L. J. Briggs[a] and others, of the U. S. Bureau of Soils, used ammonia to dissolve the colloids in soils. It was found that ammonia was most effective in deflocculating soils containing a low percentage of soluble salts. For instance, little effect was noticed on soils containing more than 1 per cent of soluble salts. The investigators mentioned also noticed that when the soil contains a considerable amount of lime or magnesium carbonate, flocculation occurs, usually becoming more pronounced on the addition of ammonia.

Förster[b] notes that the viscosity of clays is decreased by small additions of alkalies. He claims that the negatively charged OH ion repels similarly charged clay particles. He also notes that the OH ion is absorbed by the clay. Spangenberg[c] claims that the OH ion is absorbed by the clay particles and the so-charged clay particles repel each other, causing a liquefaction of the clay slip.

Förster[d] claims that the Ca ion dissolved by clay slips from the plaster molds prevents to a certain extent the deflocculating action of the OH ion. In order to prevent this he advises the use of Na_2CO_3 instead of NaOH, thereby changing the Ca ion to harmless $CaCO_3$.

The German patent 201404[e] issued in 1906, states that the plasticity of kaolin, clay, and ceramic mixtures can be increased by the addition of tannic acid. It also states that tannic acid with an excess of alkali when added to clay causes a certain fluidity in the slip, which on the addition of more tannic acid, again thickens. Certain other organic acids may be used in the place of tannic acid.

The introduction of alum[f] to accelerate sedimentation was another important step in the clay-washing industry. Hirsch,[g] Simonis,[h] Mellor,[i] Rieke,[j] and others studied the effect of soluble sulphates on clays. Most important is probably Rieke's work, in which he showed that $MgSO_4$, $CaSO_4$, $FeSO_4$, $ZnSO_4$, $CuSO_4$, $CoSO_4$, $Al_2(SO_4)_3$, and MgO increased the viscosity of a Zettlitzer kaolin slip.

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[a] Briggs, L. J., Martin, F. O., and Pearce, J. R., Centrifugal method of mechanical soil analysis: Bureau of Soils Bull. 24, 1904, p. 24.

[b] Förster, F., Ueber das Giessen des Tons: Chem. Ind., Jahrg. 28, 1905, pp. 733-740.

[c] Keppeler, Gustav, and Spangenberg, Albert, Verfahren Kaoline, Tone und aus hergestellte keramische Massen plastischer zu machen: Chem. Ind., Bd. 31, 1908, p. 592.

[d] Förster, F., Loc. cit.

[e] Keppeler, Gustav, and Spangenberg, Albert, German patent 201404, 80b, Aug. 29, 1906.

[f] Ries, Heinrich, Clays of the United States east of the Mississippi River: Prof. Paper 11, U. S. Geol. Survey, 1903, p. 35.

[g] Hirsch, H., Verhalten von Ton in Salzlosüngen: Tonindustry Ztg., 1904, p. 491.

[h] Simonis, M., Weitere Beiträge zum Verhalten von Tonen und Magerungsmitteln gegen Elektrolyte: Sprechsaal, Jahrg. 36, 1906, p. 1186.

[i] Mellor, J. W., Studies on clay slips: Trans. Eng. Cer. Soc., vol. 6, 1906-7, pp. 161-170.

[j] Rieke, Reinhold, Ueber die Wirkung löslicher Sulfate auf Kaoline und Tone: Sprechsaal, Jahrg. 43 1910, p. 709.

Alkalies were also used by Schwerin[a] in his electrical method of purification of clays. Before the clay slip was treated with electricity it was deflocculated with an electrolyte to permit the coarser impurities to settle. The finer impurities were then removed by electrolysis.

Ashley[b] found that the color of clays in which iron is one of the bases would be improved if the clay was blunged with hydrochloric acid, settled, and the overlying water drawn off.

Ashley[c] also reports Keppler's patent on improving clays by deflocculating with alkali, making an addition of colloid matter from humus, and reprecipitating all together by the addition of an acid.

Bleininger,[d] in summarizing his results on the electrical separation of clay, says the use of caustic soda and other electrolytes brings about a condition of minimum viscosity which greatly assists in the separation of quartz and feldspar from the clay.

Brown and Howat[e] found that by deflocculating kaolins with caustic soda or silicate of soda their color was improved and the drying shrinkage decreased, but the burning shrinkage increased.

Bleininger[f] states:

> The washing of kaolinitic clays may be decidedly improved as far as the quality of the product is concerned by adding to the water in the blunger a definite small amount of caustic soda or a mixture of caustic soda and sodium silicate. This treatment improves the color of most clays, enriches their content in clay substance, decreases the drying shrinkage, but tends to increase the burning shrinkage.

EXPERIMENTS WITH KAOLINS OF SOUTH CAROLINA AND GEORGIA.

PRELIMINARY INVESTIGATION.

The preliminary tests on the different secondary kaolins of South Carolina and Georgia were made by Robert Back in 1913–14 at the Washington laboratory of the bureau. Back investigated different methods of refining these secondary kaolins, and compared the color of the refined product when burned and glazed at cone 10 to that of English china clay burned under the same conditions. He found that by thoroughly deflocculating kaolin from the Dry Branch district with a small measured quantity of caustic soda the impurities could be settled out with greater ease. The product showed sufficient improvement in color when made up into hard, fire-glazed porcelain bodies to justify the washing of several tons of the Dry Branch (Ga.) kaolin at one of the washing plants operating in that district. The results of Back's work have been published by permission of the bureau in the Transactions of the American Ceramic Society.[a]

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[a] Stoermer, M., Ein neues elektrisches Tonreinigungsverfahren: Tonindustrie-Ztg., Jhrg. 36, 1912, pp. 1283-1284, Schwerin, —, Electrical process for the purification of clays; Trans. Eng. Cer. Soc., vol. 12, 1912-1913, pp. 36-65; Bleiniger, A. V., Use of sodium salts in the purification of clays and in the casting process, Tech. Paper 51, Bureau of Standards, Sept. 25, 1915, pp. 14–19.

[b] Ashley, H. E., The technical control of the colloid matter in clays: Trans. Am. Cer. Soc., vol. 12, 1910, p. 802.

[c] Ashley, H. E., Op. cit., p. 803.

[d] Bleininger, A. V., Note on the electrical separation of clay: Trans. Am. Cer. Soc., vol. 15, 1913, p. 343.

[e] Brown, G. H., and Howat, W. L., The use of deflocculating agents in the washing of clays and the effect of the process upon the color: Trans. Am. Cer. Soc., vol. 17, 1915, p. 87.

[f] Bleininger, A. V., Use of sodium salts in purification of clays and in the casting process: Tech. Paper 51, Bureau of Standards, Sept. 25, 1915, p. 38.

PURPOSE OF PRESENT TESTS.

In order to determine whether some practical method of refining the secondary kaolins of Georgia on a commercial scale could be devised, the Bureau of Mines arranged a cooperation agreement with the Georgia Kaolin Co., Dry Branch, Ga., and this work was begun by the writer in March, 1915.

EFFECT OF ALKALIES, ACIDS, AND SALTS ON GEORGIA KAOLIN SLIP.

Much work has been done in this country and abroad to determine the effect of electrolytes on clay in suspension in water, and in nearly every investigation carried on in this country Georgia kaolin was included among the clays tested. Therefore only a brief summary of the effect of alkalies, acids, and salts on Georgia kaolin suspensions, as determined in experiments by the writer, will be given here.

EFFECT ON VISCOSITY OF SLIP.

It is well known that alkalies in small amounts tend to increase the state of dispersion of clay-water systems, the result being a decrease in viscosity of the system. This breaking up of the larger clay particles into smaller ones is plainly shown by the following mechanical analyses of two samples of washed Georgia kaolin, one of which was washed in a solution containing 0.075 per cent NaOH. These analyses were made with a Schultze elutriation apparatus having an overflow of 80 c. c. per minute.[b]

TABLE 1.—*Results of mechanical analyses of washed Georgia kaolin.*

Size of clay particles.	Washed Georgia kaolin.	Georgia kaolin washed with 0.075 per cent NaOH.
Mm.	*Per cent.*	*Per cent.*
0.110	0.27	0.28
.0765	.42	.45
.0578	.33	.30
.0340	4.40	.92
.0187	2.36	2.30
.0100	15.70	9.04
	76.52	86.61

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[a] Back, Robert, Effects of some electrolytes on clays: Trans. Am. Cer. Soc., vol. 76, 1914, pp. 515–546.

[b] For description of this apparatus see Watts, A. S., Mining and treatment of feldspar and kaolin in the southern Appalachian region: Bull. 53, Bureau of Mines, 1913, p. 46.

This table shows that the particles that were larger than 0.034 mm. were little affected by hydroxide; therefore they were chiefly impurities and not colloidal clay.

On the other hand, many acids and salts have the opposite effect to that of alkalies, causing a coagulation of the clay particles and a resulting increase in the viscosity of the system.

Therefore the effect of the addition of different amounts of electrolytes to a clay-and-water system can be satisfactorily studied by the determination of viscosity. This is best done by the use of a brass efflux tube or viscosimeter. The viscosimeter used by the writer was similar to the one described by Bleininger,[a] with the exception of the size of the efflux, which was $\frac{3}{32}$ instead of $\frac{5}{32}$ inch, as recommended by Bleininger. This reduction in size of efflux is necessary in determining the viscosity of slips containing a small percentage (less than 25 per cent) of solids. All viscosity determinations made by the writer are based on the volume of water that ran through the $\frac{3}{32}$-inch efflux tube in 117 seconds, which is to be understood when mention is made of viscosity determinations throughout this report. The viscosities of all clay suspensions are computed in terms of water by dividing their observed time of flow by 117.

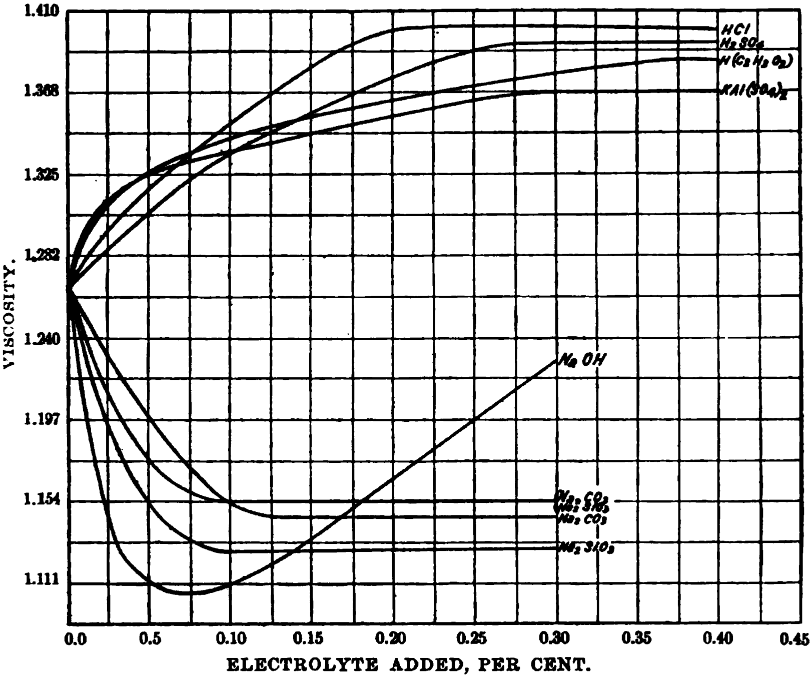

FIGURE 2.—Effect of electrolytes on viscosity of slip containing 20 per cent of Georgia kaolin.

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[a] Bleininger, A. V., Use of sodium salts in purification of clays and in the casting process: Tech. Paper 51, Bureau of Standards, Sept. 25, 1915, p. 20.

Figure 2 shows the effect of alkalies, acids, and salts upon the viscosity of a Georgia kaolin slip containing 20 per cent clay. From these curves it is seen that caustic soda caused the greatest drop in viscosity of any of the reagents used, the viscosity reaching a well-defined minimum with an addition of 0.075 per cent of this reagent. On further additions of caustic soda the viscosity at once increased. In the case of sodium silicate and sodium carbonate the decrease in viscosity was not as great as that caused by additions of caustic soda. Also the minimum viscosity is not reached until 0.10 per cent of sodium silicate, or 0.125 per cent of sodium carbonate, is added to the clay-and-water system. A mixture consisting of 50 per cent Na_2CO_3 and 50 per cent Na_2SiO_3 caused a smaller drop in viscosity than either of these reagents alone, the minimum viscosity being reached with 0.10 per cent of the mixture. Neither sodium carbonate, sodium silicate, or a mixture of the two showed any tendency to increase the viscosity.

In the case of the coagulating agents hydrochloric acid showed the greatest coagulating tendency, the mixture reaching a well-defined maximum viscosity upon the addition of 0.2 per cent of the reagent. Acetic acid and alum did not increase the viscosity as much as hydrochloric acid, the maximum viscosity produced by these reagents being 1.384 and 1.376, respectively, as compared to 1.401 produced by hydrochloric acid; but they were much more active in coagulating the clay particles below 0.05 per cent additions. The addition of 0.4 per cent of acetic acid, or 0.3 per cent of alum, was necessary to reach the maximum viscosity.

In the case of sulphuric acid the maximum viscosity of 1.393 was reached with the addition of 0.275 per cent of the reagent.

EFFECT UPON SEDIMENTATION.

It is also a well-known fact that additions of small amounts of alkalies tend to hold the clay in suspension or in a state of deflocculation which corresponds to decreasing viscosity, while acids and salts have the opposite effect upon the slip, causing a more rapid settling of the clay particles. The slip is then said to be in a state of flocculation, which corresponds to an increase of viscosity.

CONTROL OF THE VISCOSITY OF CLAY SLIP.

The fact that alkalies when added to clay-and-water systems increase the dispersion of the system with resulting decrease in viscosity and increased suspension of the clay particles makes this principle directly applicable to the purification of Georgia kaolins, for the impurities in the Georgia kaolins are coated with clay, which fact makes it impossible to settle the impurities out without a great loss of clay, but when the clay-and-water system is dispersed they are then free to settle. Also the consequent increased suspension of the clay particles gives a longer time in which impurities may settle before the process becomes uneconomical owing to the settling of clay. The success of any kaolin-refining process making use of these facts depends upon the technical control of the viscosity of the clay-water system. Therefore a study of the factors affecting both the initial and the minimum viscosity was made under the conditions of a commercial washing plant.

FACTORS AFFECTING THE INITIAL VISCOSITY.

It is a well-known fact that present-day kaolin-refining methods are not only wasteful but very inefficient, which is shown clearly by viscosity tests performed by the writer at Dry Branch, Ga. These tests not only point out the cause of inefficiency, but show why better results have not been obtained.

VARYING CLAY CONTENT.

Without some mechanical means of controlling the amount of clay going into the washer, wide variations in the percentage of clay in the slip will occur. These variations are sometimes very great, as is shown in the table following, which shows the variation in clay-and-water content of the slip during a period of one month. From the data in this table it is seen that the percentage of clay varied from 0.6 to 12.7 per cent, with an average of 7.64 per cent for the month. These tests were made at all hours of the day and under varying conditions of weather, temperature, mining operations, and other changing factors, so that the average of 7.64 per cent of clay represents the average conditions found in the washing plant. It can be readily seen that this wide range in composition of slip will affect both the quantity and the quality of the product.

TABLE 2.—*Percentages of clay and water in slip during one month of operation.*

Date.	Hour.	Water.	Clay.	Date.	Hour.	Water.	Clay.
(1916)		*Per cent.*	*Per cent.*	(1916)		*Per cent.*	*Per cent.*
Mar. 20	9.00 a.m.	90.0	10.0	Apr. 7	8.15 a.m.	95.4	4.6
24	10.00 a.m.	91.6	8.4		9.45 a.m.	93.9	6.1
	11.00 a.m.	92.7	7.3		10.00 a.m.	96.0	4.0
25	9.00 a.m.	93.6	6.4		11.00 a.m.	94.4	5.6
	10.00 a.m.	90.2	9.8	8	6.45 a.m.	91.8	8.2
	1.00 p.m.	93.6	6.4		8.15 a.m.	88.9	11.1
26	8.00 a.m.	92.4	7.6		8.30 a.m.	94.2	5.8
	10.00 a.m.	93.6	6.4		8.45 a.m.	94.5	5.5
	12.00 a.m.	94.0	6.0		1.30 p.m.	91.5	8.5
27	8.00 a.m.	92.0	8.0	9	8.30 a.m.	88.9	11.1
	10.00 a.m.	91.8	8.2		10.00 a.m.	92.7	7.3
29	8.00 a.m.	94.8	5.2		2.00 p.m.	91.8	8.2
31	11.30 a.m.	95.6	4.4	10	8.00 a.m.	90.5	9.5
Apr. 1	8.00 a.m.	91.7	8.3		9.00 a.m.	87.3	12.7
2	9.30 a.m.	93.0	7.0		10.30 a.m.	93.9	6.1
3	2.45 p.m.	95.5	4.5		11.30 a.m.	89.4	10.6
5	7.30 a.m.	99.4	0.6	12	7.15 a.m.	86.5	13.5
	8.00 a.m.	92.2	7.8	14	6.30 a.m.	93.4	6.6
	9.00 a.m.	94.2	5.8		8.00 a.m.	91.4	8.6
	10.00 a.m.	88.7	11.3	15	5.00 p.m.	88.6	11.4
	1.30 p.m.	90.9	9.1	16	10.00 a.m.	89.6	10.4
	1.45 p.m.	94.7	5.3		10.30 a.m.	95.2	4.8
	5.00 p.m.	91.8	8.2		1.00 p.m.	91.8	8.2
7	7.15 a.m.	91.8	8.2		4.00 p.m.	91.2	8.8

Average, 7.64 per cent clay.

DEGREE OF BLUNGING.

The effect of different degrees of blunging on the initial viscosity is shown in curves *1*, *3*, and *4* of figure 3. Curves *1* and *3* were plotted from viscosity tests of slips taken direct from the washer or blunger of the plant at which the tests were made, the tests of curve *3* being made in dry weather while those of curve *1* were made in wet weather. From these two curves it is seen that there is a decided increase in viscosity of clay treated during wet weather. This increase is due to the fact that the clay is wet and sticky as it enters the washer, which in turn decreases the blunging action of the washer, causing a decrease in the dispersion of the clay particles with consequent increase in viscosity. It is a well-known fact at kaolin-refining plants that the clay will not settle as quickly in wet weather as in dry weather, and alum is used to precipitate the clay. This fact seems to be contradictory to the above statements, but the reason for the slow rate of settling is not due to increased dispersion but to the decreased percentage of clay in the slip.

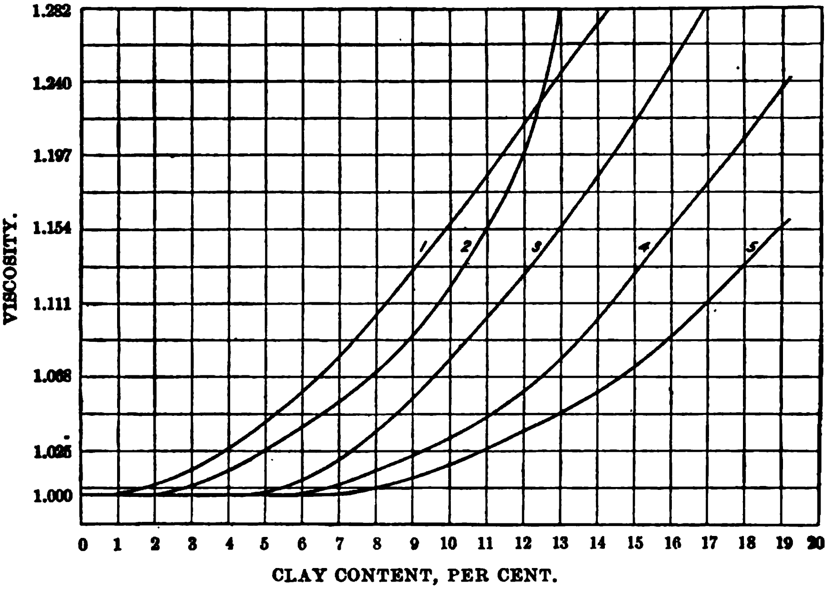


FIGURE 3.—Effects of moisture in clay used, blunging, and additions of alum, on Georgia kaolin slip. *1*, test made during wet weather; *2*, water contained 0.03 per cent alum; *3*, test made during dry weather; *4*, undried clay blunged 2 hours in ball mill; *5*, bone-dry clay blunged in small churn.

The effect of increased blunging action is shown in curve *4* of figure 3. The slips for this test were made from clay taken direct from the mine during dry weather, blunged in a ball mill for two hours, screened through a 120-mesh sieve, and then tested. From this curve it is seen that increased blunging causes an increased drop in the viscosity, owing to increased dispersion.

PREDRYING.

Curve *5* of figure 3 shows the effect of drying clay before blunging with water. These tests were made by blunging bone-dry clay with water in a small household churn. The curve shows that drying produces still greater dispersion with resulting low viscosity.

USE OF ALUM.

The tests represented by curve *2* of figure 3 were made in dry weather, but the feed water contained about 0.03 per cent of alum. From this curve it is evident that small percentages of alum will greatly increase the viscosity, the effect of the alum becoming more pronounced as the percentage of solids in the slip increases.

THE EFFECT OF VARYING VISCOSITY ON CAPACITY OF WASHING PLANT.

The capacity of a washing plant depends on the maximum flow of water through the washer and the percentage of clay held in suspension at the maximum viscosity at which precipitation of impurities is not retarded.

This maximum permissible viscosity will vary to some extent with clays from different deposits and can be determined only by close observation of a number of settling tests of slips of varying viscosities. In most cases a viscosity of about 1.094 will be the maximum. However, it is advisable to run at a viscosity a little lower than the maximum, a viscosity of 1.068 being safe practice.

The maximum amount of water that will flow through the ordinary rectangular washer, 10 feet long, 4 feet high, and 4 feet wide, without overflowing and produce maximum dispersion under given blunging conditions depends on the condition of the clay entering the washer, whether wet or dry, lumpy or disintegrated. Under the present-day conditions of kaolin refining, where the clay goes into the washer in large lumps, the maximum flow is about 1,200 cubic feet per hour.

It has been shown that a number of factors affect the percentage of clay contained in a slip of a given viscosity. Therefore these factors tend to increase or decrease the capacity of a washing plant, according to whether they decrease or increase the viscosity of the slip. For example, suppose the plant is operating under dry-weather conditions and maintaining a constant viscosity of 1.094; the slip will then contain 10.8 per cent of solids, but under wet-weather conditions the slip will contain only 7.4 per cent of clay, a decrease in capacity of 31 per cent, whereas if the clay has first been dried the slip will contain 16.2 per cent solids, an increase in capacity of 50 per cent over dry-weather conditions.

FACTORS AFFECTING MINIMUM VISCOSITY.

As the previous tests showed that caustic soda produces the greatest dispersion in Georgia kaolin slip, it was used in commercial tests of kaolin refining, and it was found that not all of the factors affecting the initial viscosity had an effect on the degree of dispersion or the minimum viscosity produced by small additions of alkalies.

DEGREE OF BLUNGING.

It is seen upon comparing the curves of figure 4, plotted from tests made in dry weather, with those of figure 5, plotted from tests made in wet weather, that the degree of dispersion or minimum viscosity of slips having the same clay content is the same, regardless of the wide difference in initial viscosity. For example: From the curve in figure 4, representing a slip which contained 9.6 per cent of clay, it is seen that the initial viscosity is 1.059 and the minimum 1.025, whereas the curve shown in figure 5, representing a slip containing 9.4 per cent solids, shows an initial viscosity of 1.145, but the same minimum of 1.025.

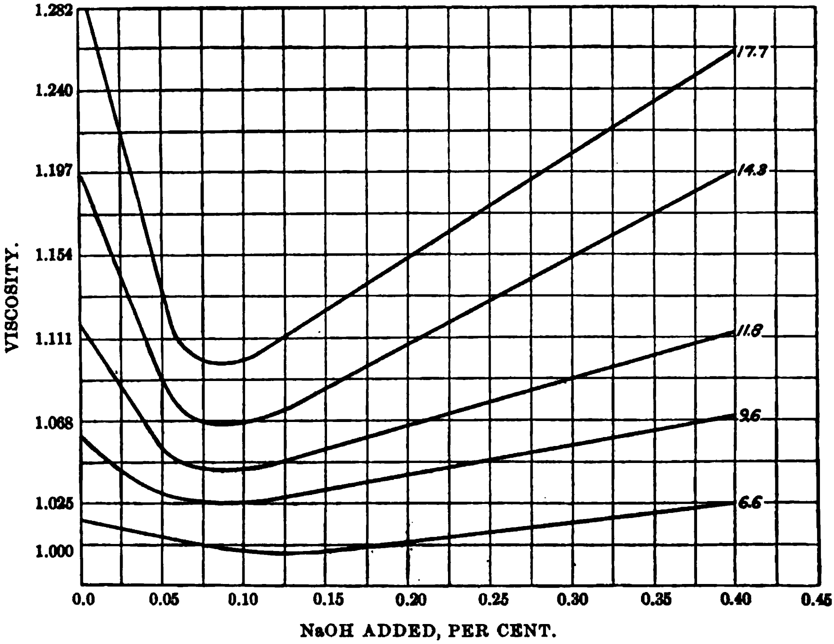


FIGURE 4.—Effect of caustic soda on viscosity of Georgia kaolin slips under dry-weather conditions. Figures on right show percentage of clay in slip.

Comparison of results shown by the curves of figure 6, which were plotted from tests of slips blunged in a ball mill, with those of the curves shown in figure 4, indicates that similar conditions prevailed. Although there was a large drop in initial viscosity, the minimum viscosity is the same for slips containing the same percentage of clay. Therefore the amount of blunging has no effect on the degree of dispersion or minimum viscosity of a slip of given clay content.

PREDRYING.

The curves shown in figure 7 were plotted from tests with slips of Georgia kaolin which had been previously dried to "bone dryness" on the top of steam pipes. By comparing these curves with those of figure 4, it is seen that predrying the clay decreases the initial viscosity—in slips containing less than 15 per cent clay—to that of the minimum viscosity. In other words, when the slip contains 15 per cent or less of clay, predrying is just as effective in increasing dispersion as additions of caustic soda, but predrying is far more expensive than using the electrolyte. However, on adding caustic soda to such slips the viscosity is decreased still further, but the drop is very small.

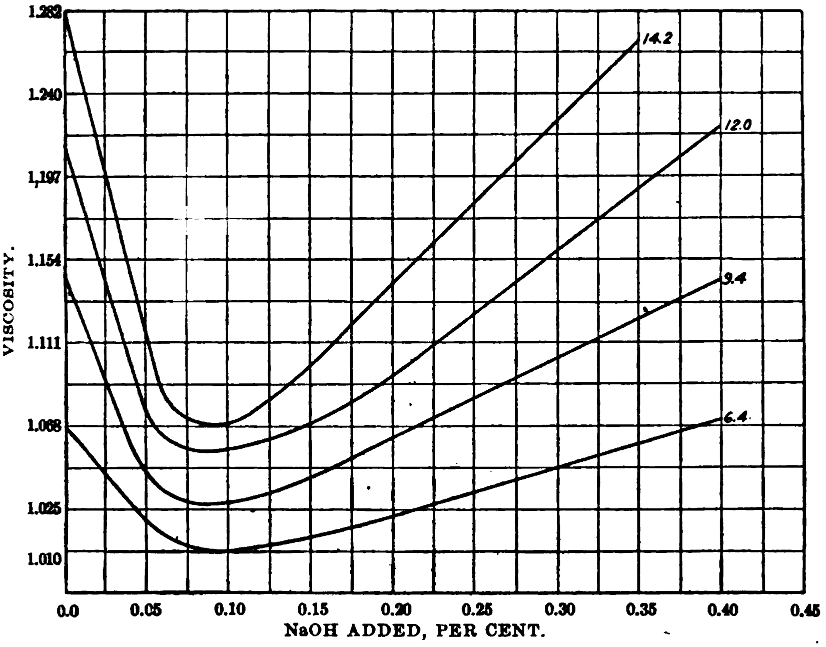


FIGURE 5.—Effect of caustic soda on viscosity of Georgia kaolin slips under wet-weather conditions. Figures on right show percentage of clay in slip.

As the minimum viscosity is not changed by the amount of blunging, and only slightly by severe predrying conditions, the capacity of a plant using a dispersing agent will be constant under such changing operating conditions.

USE OF ALUM.

The results of tests made with slips containing a small percentage (0.03 per cent) of alum are shown by the curves in figure 8. By comparing these curves with those of figure 4 it is seen that alum raises the initial viscosity, increases the amount of caustic soda necessary to produce minimum viscosity, and decreases the degree of dispersion. Therefore the presence of a flocculating agent such as alum in the feed water is very objectionable to a successful application of the technical control of disperse systems in kaolin refining.

REFINING KAOLIN BY TECHNICAL CONTROL OF DISPERSE SYSTEM.

SEDIMENTATION PROCESS.

From laboratory experience and from suggestions made by Ashley, Bleininger, and others, it appears that the best method of utilizing caustic soda as a deflocculating agent in kaolin refining is as follows:

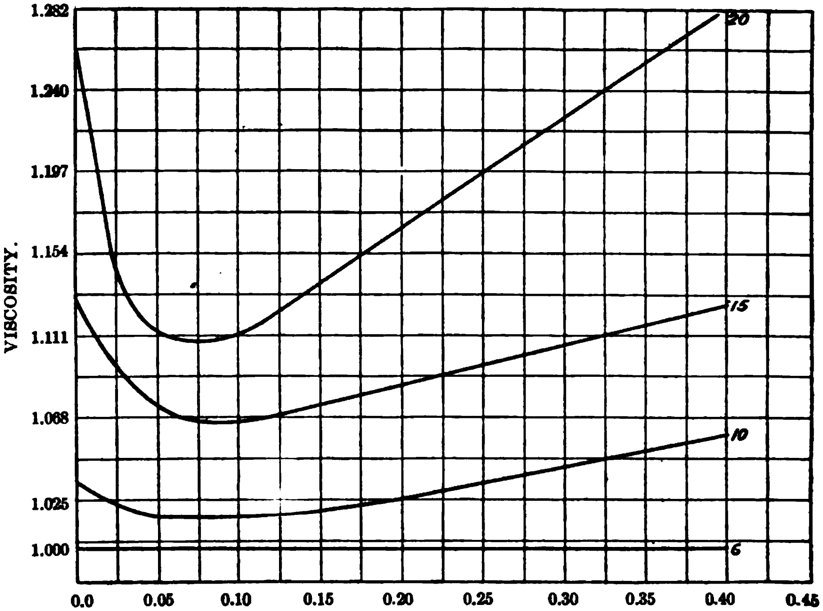


FIGURE 6.—Effect of caustic soda on viscosity of Georgia kaolin slips blunged in ball mill.

Sufficient caustic soda to cause maximum deflocculation or minimum viscosity is added to the clay and water in the washer. The deflocculated slip is run through the settling troughs, screened, and collected in the concentration tank, where it is allowed to stand long enough for the impurities to settle out. The slip is then siphoned into another tank and a flocculating agent added in order to coagulate the clay particles and concentrate the slip for filter pressing.

However, the results of a number of tests conducted in a small concrete tank 6 feet square and 2 feet deep showed that settling out the impurities in large tanks was not an economical process. The time required to settle out the impurities through a large quantity of slip was comparatively long, being about two hours, and in the meantime much clay had also settled, which had to be thrown away with the impurities. Also several inches of slip had to be left in the bottom of the settling tank when the slip was drawn off to prevent some of the impure material being sucked up. Aside from the large loss of clay caused by use of such a method, the continuity of the refining process is stopped, which adds to the operating costs.

THE ELUTRIATION OR SLIMING PROCESS.

Experiments on settling out the impurities in troughs were next carried on. A new set of troughs was built for this purpose and after a number of changes in width and length of the last trough, good results were obtained. The elutriation process consists of 11 different operations which are as follows: (1) Storage and disintegration, (2) feeding, (3) blunging, (4) dispersion, (5) sliming, (6) coagulation, (7) screening, (8) concentration, (9) filter pressing, (10) drying, and (11) crushing.

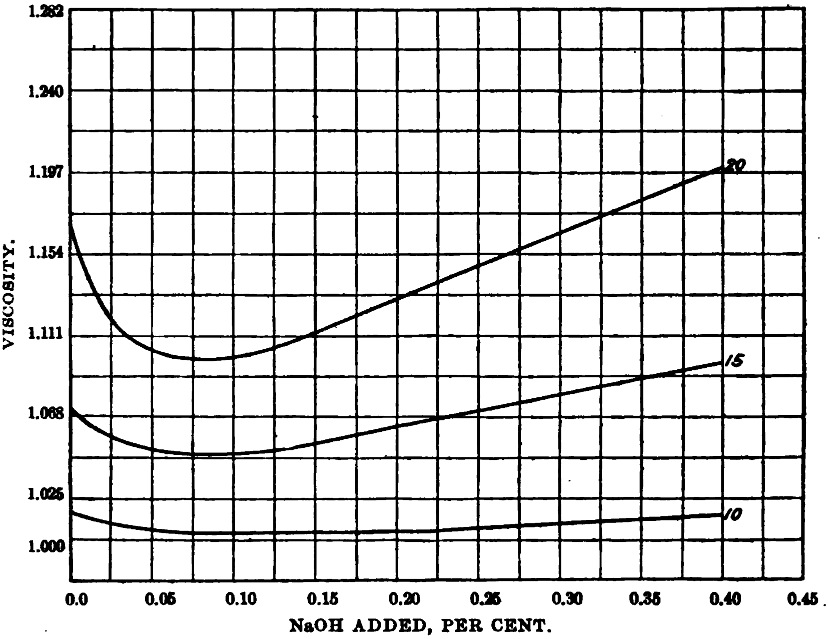


FIGURE 7.—Effect of caustic soda on Georgia kaolin slips made from predried clay.

STORAGE AND DISINTEGRATION.

The clay should be broken into lumps no larger than a man's fist and most of the clay should be much finer. In order to disintegrate the clay in this manner it is necessary to have a storage shed of at least 500-ton capacity, because during rainy weather the clay as mined from the pit is too sticky to be broken up.

All washing plants should have a storage shed for the crude clay regardless of the method of washing employed, for it has been shown that the capacity of the plant will vary greatly with the condition of the clay entering the washer unless it is dispersed with an alkali. However, few kaolin refining plants have a storage shed; therefore the writer wishes to emphasize the fact that the first thing to be considered in constructing a washing plant, if constant conditions are to be maintained with resulting uniform product, is a well-constructed storage shed of sufficient capacity. The disintegrator should be of the centrifugal roll type, as this type works best with highly plastic clays.

FEEDING.

It has been shown that the quantity of caustic soda required to produce maximum dispersion depends upon the percentage of clay in the slip; therefore it is expedient to have a constant flow of clay into the washer. The best way to obtain a constant flow is by the use of a clay feeder of the disk type. No feeder will deliver clay to a washer with any uniformity unless the clay has been previously crushed into small lumps; hence the necessity of using a disintegrator. One type of feeder is shown in Plate IV, *A*.

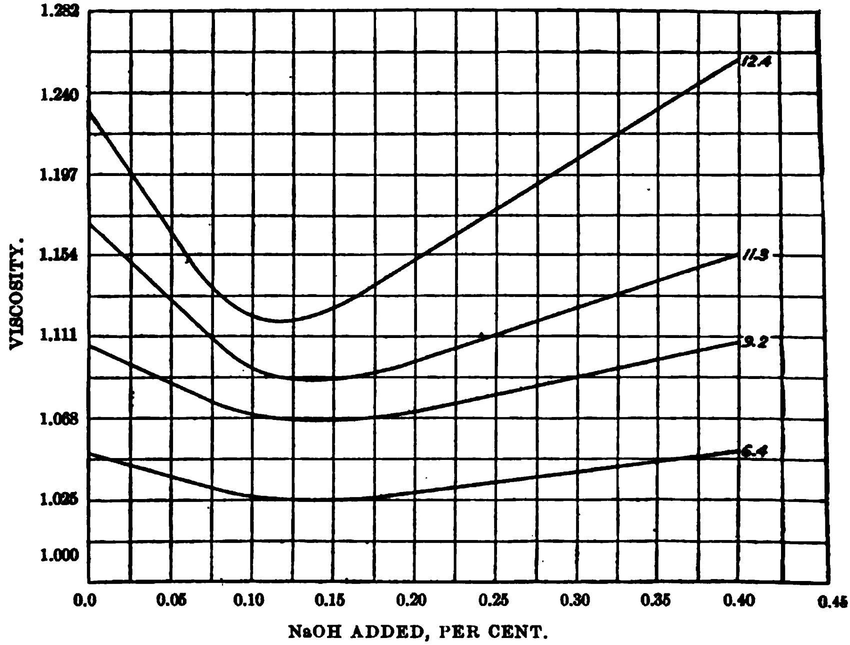
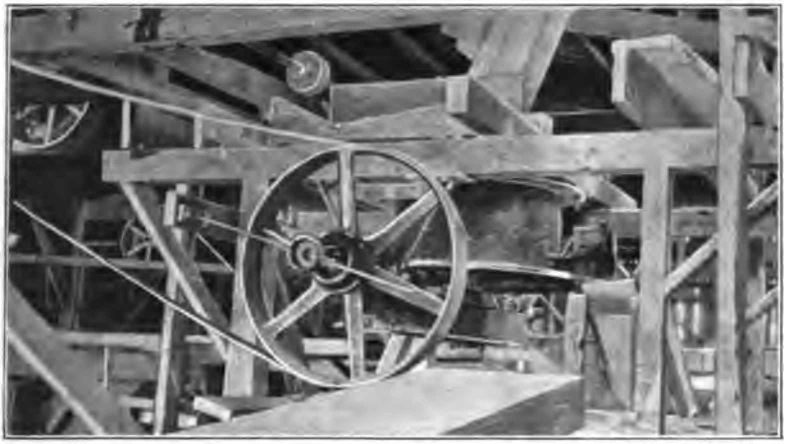


FIGURE 8.—Effect of caustic soda on Georgia kaolin slips containing 0.03 per cent of alum.

BLUNGING AND DISPERSION.

Blunging and dispersion will be discussed together as they are very closely related.

Blunging the clay, water, and alkali is one of the most important steps in the washing process, and too much emphasis can not be



A. DISK CLAY FEEDER.

B. CONCRETE WASHER OR BLUNGER.

laid upon this operation. Double blunging or washing is necessary to obtain complete dispersion and in the case of some very tough and tenacious clays that resist blunging it will be necessary to use three washers in order to get complete separation of the clay particles and impurities.

It is advisable to add the caustic soda to the slip in the second blunger, because if the flow of clay or water should be increased or decreased the clay content of the slip can be calculated from viscosity or specific-gravity tests of the slip leaving the first washer, and the flow of standard caustic-soda solution can be so regulated as to produce maximum dispersion under the changed flow of clay or water. In case the maximum permissible viscosity is exceeded the process should be stopped. In order that there will be no serious stoppage of the flow of clay or water into the washer there should be a storage bin of about 8-ton capacity, placed above the clay feeder, and the water fed into the washer by a No. 5 centrifugal pump. This pump should be set below the level of the bottom of a large concrete water-supply tank, about 30 by 15 by 8 feet, so that the water will run directly into the pump and prevent it from sucking air, which necessitates priming the pump.

TYPES OF WASHERS.

The rectangular washer used in nearly all washing plants has proved very successful in blunging the clay, but the power consumption has been very high, owing to faulty construction. The concrete washer illustrated in figure 9 and shown in operation in Plate IV, *B* will give efficient results, and the cost of repairs on this type of washer will be much lower than on the rectangular wooden box ordinarily used, as it can be easily relined without the necessity of rebuilding the whole washer.

ADDITION OF CAUSTIC SODA.

A standard solution of caustic soda is made up, and this solution is run into the second washer at a constant rate of about 10 cubic centimeters per second, which is obtained by allowing the solution to run through an efflux tube one-eighth of an inch in diameter. This efflux tube is placed about 2 inches from the bottom of the caustic soda supply tank so that it will not be easily clogged by any material that has settled to the bottom of the tank. The supply tank should also be equipped with two auxiliary efflux tubes of smaller diameter for regulating the quantity of caustic soda if the flow of clay is changed. The tank, which should have a capacity of 150 gallons, should be made from alkali-resisting material, such as enameled steel, and should contain an agitator. The head or top of the tank should be made air-tight in order that the effect of the varying head due to the lowering of the level of the solution can be overcome by the application of Mariotte's principle of the flow of liquid from an immersed tube, a constant flow of standard solution into the washer being thus obtained.

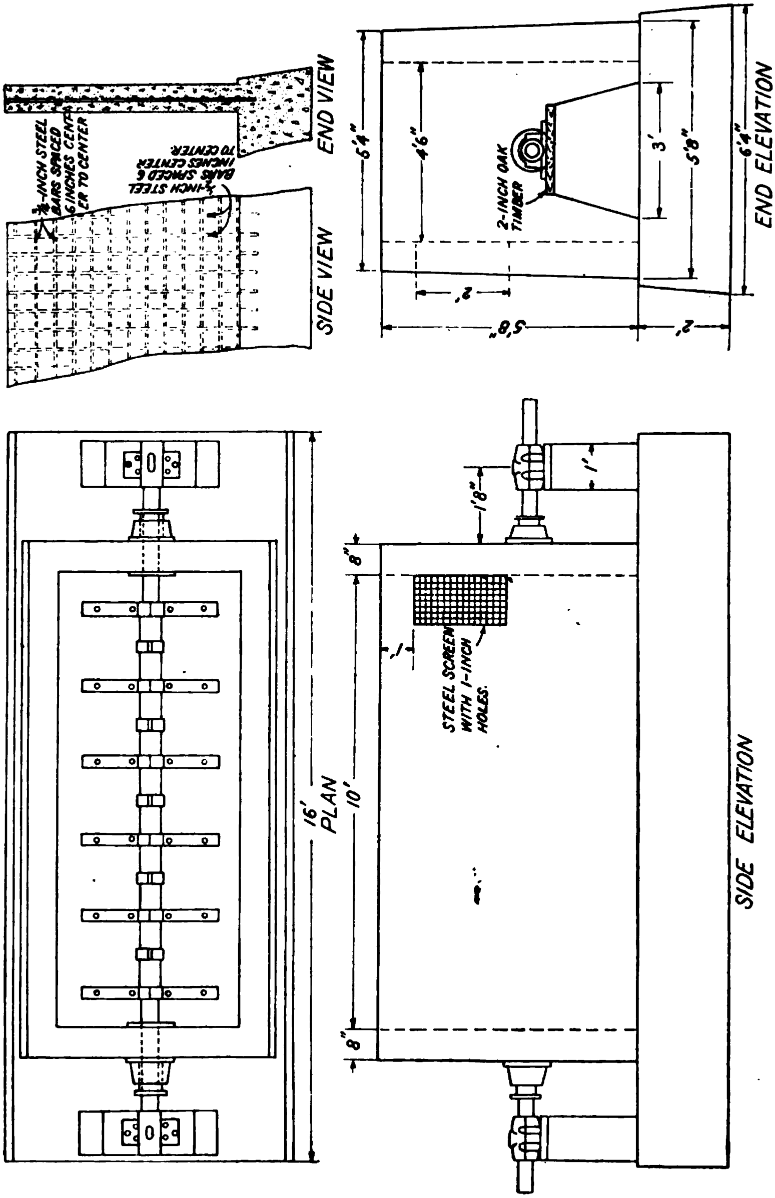

END VIEW
SIDE VIEW
¾-INCH STEEL BARS SPACED 6 INCHES CENTER TO CENTER
¼-INCH STEEL BARS SPACED 6 INCHES CENTER TO CENTER
6'4"
4'6"
2-INCH OAK TIMBER
3'
5'8"
6'4"
END ELEVATION
2'
5'8"
2'
16'
PLAN
10'
8"
8"
1'8"
1'
1"
STEEL SCREEN WITH 1-INCH HOLES.
SIDE ELEVATION

FIGURE 9.—Design of concrete washer.

The figures given above for the rate of flow of standard caustic soda solution and for the size of the main efflux tube are somewhat arbitrary and can be changed to suit different conditions.

STRENGTH OF CAUSTIC-SODA SOLUTION.

The proper strength of caustic soda solution to use will depend on the rate at which the solution is allowed to flow into the washer and the rate at which the clay is fed to the washer. For example, if 5 tons of clay is being fed to the washer every hour, then $5 \times 2000 \times 0.00075$, or 7.5, pounds, of caustic soda will have to be added every hour to produce complete dispersion. Now, if the rate of flow of the solution is 10 cubic centimeters per second, every cubic centimeter of standard solution will have to contain $\frac{7.5 \times 454.5}{3600 \times 10} = 0.094$ gram of caustic soda.

Therefore the normal solution is $\frac{0.094 \times 1000}{40} N = 2.35N$.

SLIMING.

After thorough blunging and complete dispersion the slip is run through the elutriation troughs. The troughs used by the writer with good success do not differ from the ordinary mica troughs used at present in most washing plants, except that a long fan-shaped trough is added at the end of the series of troughs.

This trough should be at least 125 feet long, 2 feet wide at the inlet, and the width at the discharge end should be such as to produce a depth of slip of not over 1 inch. The trough should be constructed with great care, for the bottom must be perfectly level both lengthwise and crosswise and the sides very smooth, as too great a frictional resistance to the flow causes coagulation of the slip along the sides and results in faster flow through the center of the trough. The bottom of the trough should be rough and uniform in texture, because on a smooth bottom the slip tends to roll the impurities along, and on an uneven bottom the slip tends to thicken around the spots of most resistance, causing it to flow too fast in other parts of the trough. The best material of which to construct these troughs is concrete, and by proper sizing of the aggregate and proper troweling the required surfaces can be obtained.

There will be less tendency for the clay particles to coagulate around spots of greater resistance if the slip is agitated before it enters the last trough. This can be easily done by having the fan-shaped trough lower than the others and then running the slip over riffles as it passes from the narrow troughs into the fan-shaped one. (See fig. 10.)

COAGULATION.

As the slip comes from the troughs it is still in a state of maximum deflocculation and therefore can not be concentrated as the exceedingly fine particles would not settle even after weeks of standing.

Furthermore, any attempts at filter-pressing the slip as it comes from the sliming troughs would be futile for, aside from the fact that the quantity of water to be removed would be so great that the process would not be practicable, the alkaline condition of the slip with the resulting dispersion of the clay particles would make filter-pressing almost impossible, as the exceedingly fine particles would tend to clog the filter cloth and what water did run through would be very turbid. Therefore it is necessary to add some flocculating or coagulating agent. Alum is usually used for this purpose, but in case the washing plant is using the water from the concentration tanks and filter presses over and over again alum can not be used, as a very small proportion (0.02 per cent or less) in the feed water decreases the effect of caustic soda and consequently increases the amount of soda

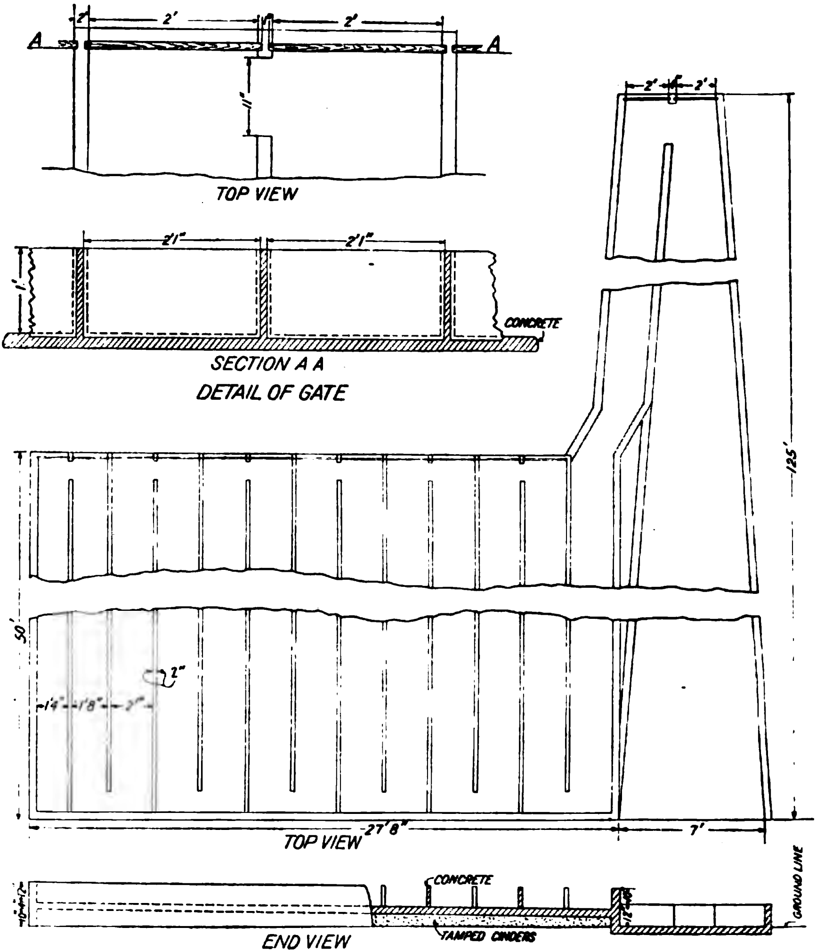


FIGURE 10.—Design of floating troughs.

necessary to cause maximum dispersion. If just enough sulphuric acid is added to neutralize the caustic soda, the slip is brought back to its normal state.

The sulphuric acid solution should be the same strength as the caustic soda solution, and it should be added to the slip in the same manner. Under these conditions perfect neutralization will result and there will not be an excess of caustic soda left to cause trouble in settling and filter-pressing or an excess of sulphuric acid left to cause trouble in blunging and dispersion. The sulphuric acid solution is added to the slip at the point where it leaves the sliming troughs. After the addition of the acid the slip is run through a long narrow trough, say 50 feet by 1 foot by 2 feet, containing a number of riffles for mixing the acid solution with the alkaline slip. The bottom of this trough should have a drop of at least 1 foot in 50 in order to cause a rapid flow.

SCREENING.

After the slip is neutralized by adding sulphuric acid it is advisable to run it through a rotary screen of about 120 mesh. This screen will remove any chips of wood or other foreign materials that float on the top of the slip; it also acts as an agitator in case some of the alkaline slip has not been completely neutralized. Care should be taken that practically all of the acid has combined with the caustic soda before reaching the screen, as any excess of acid will act upon the screen cloth and shorten its life.

CONCENTRATION.

The concentration of the slip is best accomplished by running the neutral slip into large concrete tanks (see fig. 11) 60 feet long, 30 feet wide, and 6 feet deep, where it is allowed to stand long enough for the clay to precipitate; then the supernatant liquid is drawn off, the time of settling governing the thickness of the slip. However, after about 96 hours the amount of precipitation is exceedingly small and in general 72 hours is sufficient for the concentration of the slip. This concentrated slip is then run into a small retaining tank, or, better still, is pumped into the filter presses direct from the concentration tanks.

FILTER-PRESSING.

Under present conditions of kaolin refining the filter-pressing of the thickened clay slip has given more trouble than all the rest of the washing operations. The troubles have been numerous and varied, and have been partly due to the type of filter press used, poor or clogged filter cloth, too thin a slip, condition of clay particles, whether dispersed or coagulated, and other factors. In view of these facts

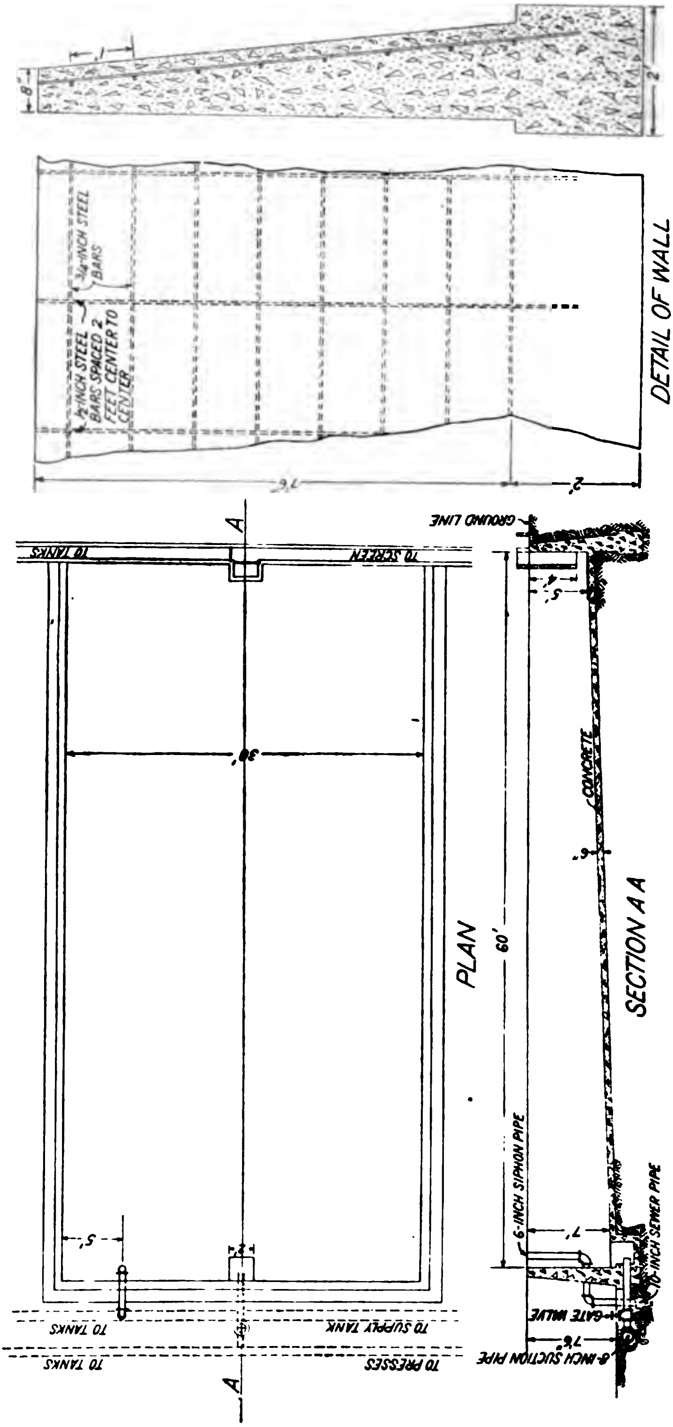


FIGURE 11.—Design of concentration tank.

numerous attempts have been made to devise other methods of drying the clay, but to the writer's knowledge none has proved more efficient than the filter press. The osmose purification process used in England[a] eliminates the filter press, but it is doubtful whether the apparatus would meet conditions in this country.

Bleininger[b] says:

> If the electrical continuous separation can replace the filter press economically its use would be justified. To a large extent it would be a matter of the comparative economy of the electrical endosmose apparatus and the filter press. If the latter should be improved, as has been done in the so-called clamshell press, it is quite possible that the economic advantage might remain with the filter press.

One of the chief sources of trouble in filter-pressing highly plastic clay slips is a slip that has not been sufficiently concentrated. In other words, the clay has not been allowed time to settle, owing, perhaps, to lack of tank capacity. The slip for filter-pressing should contain at least 30 per cent of clay. It must be remembered that in order to force this thick clay slip into the presses the pumps must be below the bottom of the tank from which they are drawing the slip; as the ordinary filter-press pump will not successfully lift a thick clay slip.

DRYING.

The drying of the filter-press cakes is best accomplished in a tunnel drier heated by exhaust or live steam. During the summer months open-air driers can be used to good advantage.

CRUSHING.

The filter-press cakes after being thoroughly dried are crushed into small pieces in a set of rolls.

LAYOUT OF PLANT.

The plan of a washing plant equipped for using the chemical process is shown in Plate V.

COST OF THE CAUSTIC SODA AND SULPHURIC ACID TREATMENT.

The extra cost over present-day methods of the caustic soda and sulphuric acid refining of kaolins may be divided into three factors: (1) First cost, maintenance, and operation of extra mechanical equipment required; (2) cost of chemicals; (3) chemist's salary. These costs, on the basis of 50 tons of clay treated per day, follow.

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[a] Ormandy, W. R., British clays under the osmose purification process: Trans. Eng. Cer. Soc., vol. 13, 1913-1914, pp. 35-48.

[b] Bleininger, A. V., Use of sodium salts in purification of clays and in the casting process: Tech. Paper 51, Bureau of Standards, Sept. 25, 1915, p. 19.

MECHANICAL EQUIPMENT.

The extra equipment necessary for the successful operation of the process is: One disk clay feeder, one washer, extra settling trough, and two 200-gallon tanks. The cost of operating and maintaining this extra equipment will be about $0.12 per ton of clay treated.

CHEMICALS.

With caustic soda at $0.03 per pound and sulphuric acid $0.025 per pound, the cost of chemicals will be $0.0825 per ton of clay.

CHEMIST.

Successful operation will depend upon careful technical control of every step of the process. It will therefore be necessary to have a trained chemist available at all times. The services of such a chemist will approximate $0.15 a ton.

The extra total cost will therefore be about $0.35 a ton. However, it must be remembered in comparing the cost of the new with the old method that the caustic soda and sulphuric acid process increases the capacity of the plant. For example, slip with a maximum permissible viscosity of 1.094 contains after complete dispersion 17 per cent of clay, whereas the slip that is not dispersed contains 10.8 per cent during dry weather and 7.6 per cent during wet weather. As the average percentage of clay content, under the present-day methods, is 7.64 per cent, the increase in capacity will be 123 per cent, which will more than offset the increased cost of operation.

CONCLUSIONS DRAWN FROM RESULTS OF TESTS.

The application of the new process of treatment to kaolin refining on a commercial scale is thoroughly practical and the product has a purity impossible to attain under present methods of refining. In order to utilize the caustic soda sulphuric acid process with clay such as that experimented with it must be remembered that:

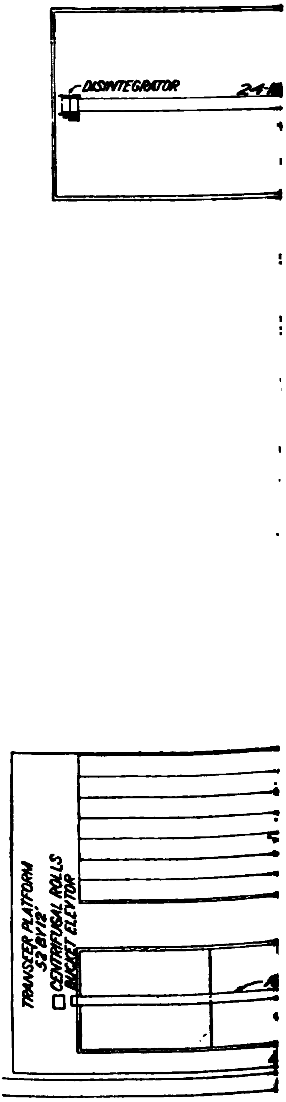
1. The clay and water must be fed into the washer at a uniform rate.

2. Double blunging or washing is necessary. Without thorough blunging the process is doomed to failure.

3. The caustic soda must be added in definite quantities, the amount depending upon the character of the clay and the quantity being fed to the washer.

4. A constant viscosity of about 1.068 must be maintained at the head of the sliming troughs in order to obtain a uniform product. The viscosity should never exceed 1.094.

5. It is advisable to agitate the slip before it is allowed to run through the fan-shaped trough.


DISINTEGRATOR
24-
TRANSFER PLATFORM 52' BY 12'
CENTRIFUGAL ROLLS
BUCKET ELEVATOR

6. The last sliming or settling trough should be fan-shaped, at least 125 feet long, 2 feet wide at the inlet, 7 feet wide at the discharge end, and perfectly level both lengthwise and crosswise. The sides of this trough should be very smooth; the bottom should be comparatively rough but uniform in texture.

7. Sufficient sulphuric acid to neutralize the caustic soda should be added, and no more.

8. It is absolutely necessary to have a chemist, one familiar with clay disperse systems, present at all times, for it is impossible to make the process fool-proof.

UTILIZATION OF REFINED GEORGIA KAOLIN.

After an improved process of kaolin refining has been worked out, the question, "Will the treated clay be sufficiently improved so that it can be utilized in the manufacture of the finer grades of ceramic wares by substituting it for the English china clay?" at once arises. In order to answer this question several tons of clay was refined by the new process at the plant of the Georgia Kaolin Co., to be used for test purposes. Owing to the fact that during the refining of this clay the flow of clay into the washer was not uniform because a clay feeder had not been installed and the amount of clay had to be approximately determined by viscosity tests taken every five minutes, it was not as pure as would be obtainable under standard conditions. However, on making a number of settling tests it was thought that sufficient improvement had been made to warrant some preliminary tests of the burned color of the clay.

PRELIMINARY TESTS TO DETERMINE PHYSICAL PROPERTIES AND COLOR.

SHRINKAGE.

The average linear shrinkage of the washed Georgia, treated Georgia, and English china clays are given in the table following. The shrinkage was determined by measuring the dry length and burned length of a 5-centimeter mark placed upon 10-centimeter disks in the green state. These disks were made by pressing the plastic clay into plaster molds. The disks were dried at 110° C. and the dry lengths measured, then fired in saggers at cone 10 in the kilns of the Mayer China Co., and the burning shrinkage determined.

Drying shrinkage and burning shrinkage of kaolin disks.

Kaolins tested.	Drying shrinkage, per cent.	Burning shrinkage, per cent.
English china clay	2.32	4.48
Washed Georgia kaolin	4.38	8.12
Treated Georgia kaolin	3.24	8.16

The table shows that the new process decreases the drying shrinkage about 25 per cent, but the burning shrinkage is about the same.

COLOR IN WHITE-WARE BODIES.

Some preliminary tests on the use of the treated Georgia kaolin in white-ware bodies were made in the ceramic laboratories of the Ohio State University, Columbus, Ohio. The purpose of these tests was to determine how much improvement in burning color the treated clay would show over that of the washed Georgia kaolin when made into glazed white-ware bodies, and also the approximate percentage of treated Georgia clay that could be substituted for the English china clay in a white-ware body without impairing the color of the glazed body.

MATERIALS USED IN MAKING BODIES.

The commercial materials used in this investigation were as follows:

Feldspar.—Canadian feldspar, Pennsylvania Feldspar Co.

Flint.—Pulverized quartz sand, Ohio Silica Co.

Florida kaolin.—Edgar Plastic Kaolin Co.

English china clay.—Moore & Munger's M. W. M. No. 1.

Ball clay.—Tennessee No. 9, Johnson & Porter Co.

Washed Georgia kaolin.—Georgia Kaolin Co.

Treated Georgia kaolin.—Georgia Kaolin Co.

Whiting.—Gilders.

The analyses of these materials are given in Table 3 following:

TABLE 3.—*Results of analyses of materials used in tests.*

Constituent.	Feldspar.	Flint.	Florida kaolin.	English china clay.	Ball Clay.	Washed Georgia kaolin.[a]	Whiting.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Silica (SiO_2)	65.15	99.18	45.50	47.37	51.20	45.53	1.30
Alumina (Al_2O_3)	18.65	.20	39.30	37.85	32.10	37.97	2.14
Iron oxide (Fe_2O_3)	.10	.06	.05	.31	.73	.78	.10
Titanium oxide (TiO_2)	.00	.01	.00	.04	1.40	1.55	.00
Lime (CaO)	.18	.01	.00	.00	.13	.05	54.40
Magnesia (MgO)	.00	.01	Trace.	Trace.	.00	.13	.00
Potash (K_2O)	10.36	....	.42	1.40	.69	....	.00
Soda (Na_2O)	4.31	....	.44	.14	.10	....	.00
Ignition loss	.32	.11	14.20	12.60	13.75	13.69	.00
Carbon dioxide	.00	.00	.00	.00	.00	....	42.68
Total	99.87	99.58	99.91	99.71	100.10	99.65	100.62

[a] The treated Georgia kaolin was not analyzed.

COMPOSITION OF THE BODIES USED.

The composition of the seven bodies that were compounded from the above white-burning materials is given in Table 4.

Table 4.—*Composition of bodies used.*

Material.	Proportion used in body No.—						
	1	2	3	4	5	6	7
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Feldspar	15	15	15	15	15	15	15
Flint	38	38	38	38	38	38	38
Florida kaolin	7	7	7	7	7	7	7
Tennessee ball clay	3	3	3	3	3	3	3
Whiting	2	2	2	2	2	2	2
English china clay	35	23	12	0	23	12	0
Washed Georgia kaolin	0	12	23	35	0	0	0
Treated Georgia kaolin	0	0	0	0	12	23	35
	100	100	100	100	100	100	100

PREPARATION OF THE BODIES.

The materials for each body, consisting of 8,000 grams, were weighed out and the total weight checked by weighing the batch. Each body was ground wet for five hours in a porcelain-lined ball mill, sieved through a 120-mesh screen, and divided into two parts, one part being used for casting small cups, and the other part being dried to a plastic state in plaster molds lined with cheesecloth, and after the clay had been wedged small pots like flower pots were made from it on a jigger.

PREPARATION OF TEST PIECES.

The bodies were made up into 48 test pieces each—24 by casting and 24 by molding on a jigger. These test pieces were first air-dried and then the drying was completed in a gas-fired drier.

BISCUIT BURN.

In order that the range of vitrification of the test pieces would cover the range of vitrification of commercial white-ware bodies, from the high porosity of wall tile to the low porosity of vitreous china, one half of the pieces were biscuit burned at cone 4 and the other half at cone 11.

GLOSS BURN.

All pieces were gloss fired at cone 3. The glaze and frit used had the following molecular formula:

Molecular composition of glaze.

$$\left.\begin{array}{l}0.25\ \mathrm{PbO}\\0.15\ \mathrm{K_2O}\\0.05\ \mathrm{Na_2O}\\0.45\ \mathrm{CaO}\\0.10\ \mathrm{ZrO_2}\end{array}\right\}0.25\ \mathrm{Al_2O_3}\left\{\begin{array}{l}2.0\ \mathrm{SiO_2}\\0.4\ \mathrm{B_2O_3}\end{array}\right.$$

Molecular composition of frit.

$$\left.\begin{array}{l}0.10\ \mathrm{Na_2O}\\0.90\ \mathrm{CaO}\end{array}\right\}0.1\ \mathrm{Al_2O_3}\left\{\begin{array}{l}2.0\ \mathrm{SiO_2}\\0.4\ \mathrm{B_2O_3}\end{array}\right.$$

The glaze was applied in a thick coat to one half of the trials by dipping and in a thin coat to the other half by spraying.

RESULTS OF TESTS.

TEST PIECES BISCUIT-FIRED AT CONE 12 AND GLOSS-FIRED AT CONE 3.

No black specks were observed in bodies 5, 6, and 7, containing treated Georgia kaolin, whereas a number of specks appeared in bodies 2, 3, and 4, especially in body 4, which contained 35 per cent of washed Georgia kaolin. The treated Georgia kaolin bodies had a better color than the washed kaolin. Body 5 had just as good a color as body 1, which contained English china clay, while body 6 had a slight creamy tinge and body 7 a decided cream color, when compared with body 1.

TEST PIECES BISCUIT-FIRED AT CONE 4 AND GLOSS-FIRED AT CONE 3.

A few black specks showed in body 4, but the difference in color between the treated and washed kaolin bodies was only slight at this heat treatment. Bodies 5 and 6 had just as good color as body 1, but body 7 showed a very slight creamy color when compared with 1.

CONCLUSIONS AS TO RESULTS OF COLOR TESTS.

Glazed white-ware bodies containing the treated Georgia kaolin have a whiter color than those containing the washed Georgia kaolin.

Treated Georgia kaolin can be substituted for at least 40 per cent of the English china clay in vitreous china, and for all the English china clay in wall-tile bodies without affecting the color of the product.

TESTS IN MAKING VITREOUS CHINA.

These preliminary tests gave indications of the possible commercial uses of the treated Georgia clay and justified further tests of substituting it for English china clay. In order that the tests might be made under commercial conditions of white-ware manufacture, they were conducted at the plants of the Beaver Falls Art Tile Co., the Mayer China Co., and the Mosaic Tile Co., through the courtesy of these firms.

The commercial tests of the use of treated Georgia kaolin as a substitute for English china clay in vitreous china bodies were made at the Mayer China Co. plant at Beaver Falls, Pa. The main object of these tests was to determine whether a practicable vitreous china body composed entirely of American materials could be made.

PROCEDURE IN TESTS.

MATERIALS USED.

The materials used in this investigation, with the exception of the treated Georgia kaolin, were taken from the stock bins of the china company. The following materials were used:

Feldspar.—Eureka Flint & Spar Co.
Flint.—Ohio Silica Co.
Florida kaolin.—Edgar Plastic Kaolin Co.
English china clay.—Moore & Munger's M. G. R. No. 2.
Ball clay.—Moore & Munger's English ball clay.
Treated Georgia kaolin.—Georgia Kaolin Co.

COMPOSITION OF BODIES.

The percentage composition of each body used in the three series of tests comprising this investigation are given in Table 5 following:

TABLE 5.—*Composition and physical properties of bodies used in tests.*

COMPOSITION.

Item.	Series A, body No.—						Series B, body No.—				Series C, body No.—					
	20	21	22	23	24	25	26	27	28	29	29b	30	31	32	33	34
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
Feldspar	16.0	16.0	16.0	16.0	16.0	16.0	17.0	18.0	16.0	16.0	16.0	16.0	16.0	16.0	17.0	16.0
Flint	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5
Ball clay	7.5	7.5	7.5	7.5	7.5	7.5	7.5	7.5	7.5	7.5	7.5	3.5	......	......	......	......
Florida kaolin	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0
English china clay	29.0	24.2	20.3	18.4	14.5	18.4	16.0	15.0	14.5	13.5	13.5	17.5	19.5	17.5	13.5	19.5
Treated Georgia kaolin	......	5.8	8.7	10.6	14.5	......	12.0	12.0	13.5	13.5	13.5	13.5	15.0	17.0	20.0	15.0
Washed Georgia kaolin	......	......	......	......	......	10.6	......	......	......	......	......	......	......	......	......	......
Whiting	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	3.0	4.0	4.0	4.0	4.0	3.0	2.0	2.0
Magnesite	......	......	......	......	......	......	......	......	......	......	......	......	......	1.0	2.0	2.0

PHYSICAL PROPERTIES.

Item.	20	21	22	23	24	25	26	27	28	29	29b	30	31	32	33	34
Total shrinkage	11.46	12.00	12.30	12.54	12.78	12.74	12.60	12.96	12.64	13.34	12.70	12.06	11.34	13.60	13.40	12.84
Drying shrinkage	2.42	2.60	2.78	3.00	3.17	3.16	2.84	3.00	3.20	3.16	3.08	2.24	2.50	3.14	3.22	2.58
Burning shrinkage	9.06	9.40	9.52	9.54	9.61	9.58	9.76	9.96	9.44	10.18	9.72	9.82	8.84	10.46	10.18	10.26
Absorption	0.05	0.41	1.00	1.08	1.30	1.02	0.26	0.03	0.60	0.02	0.16	1.19	2.26	1.05	0.76	1.04
	Cm.	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*
Translucency	0.65	......	......	......	......	......	0.61	0.64	0.61	0.68	0.40	0.45	0.46	0.51	0.53	0.54

PREPARATION OF BODIES.

The weight of each ingredient taken was determined from the percentage composition, figured on the basis of an 8,000-gram batch for each body. After weighing and checking the total weight by weighing the batch 1.2 grams of cobalt sulphate was added for a stain, and the bodies blunged in the ordinary wet way in porcelain-lined ball mills. The bodies in series A were ground for only five hours, but after the bodies of this series were biscuited it was found that the blunging was insufficient to give the desired uniform texture to each body; therefore the bodies of series B and C were blunged for 20 hours. After thorough grinding, the bodies were passed through a 120-mesh sieve and dried to a plastic condition in plaster molds lined with cheesecloth. As soon as the bodies were sufficiently dry they were wedged on a marble slab, and then each body was made up into 10 disks 10 centimeters in diameter for shrinkage and absorption tests, and twelve 5-inch plates for color comparisons. Twenty-four wedge-shaped pieces for translucency tests were also made from each body of series B and C. These pieces were allowed to dry for about 12 hours and then fettled.

BISCUIT BURN.

The finished pieces were placed in saggers in the ordinary commercial way, and placed in the second ring of the biscuit kiln. The pieces in series A and B were burned to cone 10, but those in series C were burned only to cone 9, owing to the lower temperature in the kiln in which the test pieces of this series were burned.

GLAZING.

The glaze and frit used had the following molecular formula:

Molecular composition of glaze.

$$\left.\begin{matrix}0.058\ K_2O\\0.207\ Na_2O\\0.530\ CaO\\0.205\ PbO\end{matrix}\right\}0.275\ Al_2O_3\left\{\begin{matrix}2.540\ SiO_2\\0.414\ B_2O_3\end{matrix}\right.$$

Molecular composition of frit.

$$\left.\begin{matrix}0.272\ Na_2O\\0.031\ K_2O\\0.697\ CaO\end{matrix}\right\}0.272\ Al_2O_3\left\{\begin{matrix}2.550\ SiO_2\\0.544\ B_2O_3\end{matrix}\right.$$

Eight kilograms of the above glaze was ground in the ordinary wet way for 60 hours in a porcelain-lined ball mill, passed through a 120-mesh sieve, and flocculated or thickened by the addition of 0.13 per cent $CaCl_2$, calculated on the basis of dry weight. The thickened glaze was then applied to the biscuited plates by dipping in the ordinary commercial way.

GLOSS BURN.

The glazed pieces were gloss fired at cone 1 in the gloss kilns of the company.

RESULTS OF TESTS.

The physical and pyrophysical properties of the different bodies shown in Table 5 were compared with those of body 20, which was represented to the writer by Mr. Ernest Mayer as being a practicable vitreous china body. The physical and pyrophysical properties studied were as follows: Working properties, drying shrinkage, burning shrinkage, absorption, transluscency, and color.

The working properties of the bodies were studied as regards the ease which the ware was formed in the jigger without flawing.

The drying shrinkage and total shrinkage were determined by measuring the dry and burned lengths of 5-centimeter marks placed on the 10-centimeter disks while in the green state. The drying and total-shrinkage data given in Table 5 are expressed in percentage of green length. The burning shrinkage was calculated by subtracting the percentage of drying shrinkage from the percentage of total shrinkage.

The absorption data were obtained by weighing the dry disks and then soaking them in distilled water for 48 hours and again weighing. The average absorptions given in Table 5 are expressed in percentages of dry weight.

The translucency was determined by measuring the thickest part of the wedge-shaped piece through which light would pass. The average translucencies are given in centimeters in Table 5.

SERIES A TESTS.

The purpose of the tests in series A was to study the direct substitution of treated Georgia kaolin for English china clay in a practical vitreous china body

CONCLUSIONS FROM RESULTS OF SERIES A.

As the proportion of treated Georgia clay was increased the working properties of the body were improved, the drying and burning shrinkages and the absorption were increased. This decreased vitrification of the Georgia-kaolin bodies is not due to the difference in fineness of grain of the English and Georgia clays, as is commonly thought, for the Georgia kaolin is much finer grained than the English china clay, but is probably due to the low alkali content of the Georgia kaolins.

The colors of bodies 21, 22, and 23 were just as good as that of body 20, whereas body 24 showed a slight cream color, therefore only about 50 per cent of the English china clay in an ordinary vitreous china body can be replaced by this treated Georgia kaolin before the color of the glazed product is impaired.

SERIES B TESTS.

The tests in series B were designed to obtain a body of low absorption and still containing the maximum percentage of treated Georgia kaolin. For this purpose the feldspar content in bodies 26 and 27, and the whiting content in bodies 28 and 29 were increased.

CONCLUSIONS FROM RESULTS OF SERIES B.

The working properties of the four bodies of this series were very good. The burning shrinkage was increased, with resulting decreased absorption and increased translucency. The color of all the bodies of this series was just as good as that of body 20. Body 29 was a good vitreous china body of high translucency.

SERIES C TESTS.

The purpose of the tests of series C was to study the effect of substituting treated Georgia kaolin for ball clay, and magnesite for part of the whiting. Owing to the decreased biscuit heat the bodies of this series had an abnormally high absorption and low translucency, as may be seen by comparing the data for bodies 29 and 29b, Table 6. These two bodies had the same composition but were biscuit-burned with different heat treatments.

CONCLUSIONS FROM RESULTS OF SERIES C.

Substituting treated Georgia for ball clay. decreased the drying and burning shrinkage and increased the absorption. The translucency was increased regardless of the decrease in vitrification.

Magnesite had a greater fluxing action than whiting, as is seen by comparing the absorption of body 31 with that of 34.

A direct substitution of Georgia clay for ball clay made the body too short, and therefore greater amounts of Georgia kaolin must be used, as compared with ball clay, to produce a practicable vitreous china body. Bodies 32 and 33, containing 17 per cent and 20 per cent of Georgia kaolin, respectively, had good plasticity, and worked well in the jigger, whereas bodies 30 and 31. which contained 13.5 and 15 per cent, were too short.

Bodies 31, 32, 33, and 34 of series C had a much better color than bodies 20 and 29 of series B, which contained ball clay. Bodies 33 and 34 had the best color of those in series C, which was evidently due to the substitution of magnesite for 50 per cent of the whiting.

GENERAL CONCLUSIONS AS TO PRACTICABILITY OF MAKING CHINA FROM GEORGIA CLAYS.

A commercial vitreous china body could not be obtained by the direct substitution of refined Georgia kaolin for English china clay, but the fluxes had to be increased to obtain the desired degree of vitrification.

The substitution of treated Georgia kaolin for all the ball clay and 50 per cent of the English china clay produced a body (No. 33) of good working qualities and as good a color as the whitest commercial ware and of a far better color than most vitreous chinaware on the market.

Body 33 contained 8 per cent of Florida kaolin, 13.5 per cent of English china clay, and 20 per cent of treated Georgia kaolin. As far as color is concerned, more than 20 per cent of treated Georgia kaolin can be used in bodies containing no ball clay, but the writer is of the opinion that if more than 20 per cent were added the temperature of the biscuit heat would have to be increased in order to obtain the desired vitrification, as an increase in percentage of fluxes over that in body 33 would reduce the toughness of the ware. As toughness is an important factor in high-grade vitreous china or hotel china, 20 per cent of treated Georgia kaolin is about the maximum proportion that can be used in this class of ware. In order to produce a vitreous china body containing only American clays it would be necessary in a body such as No. 33 to substitute for the 13.5 per cent of English china clay a domestic kaolin that is as easily vitrified as the English china clay and having as good a burning color.

MANUFACTURE OF WALL TILE.

The commercial tests of the substitution of treated Georgia clay for English china clay in wall tile were made at the factories of the Beaver Falls Art Tile Co., Beaver Falls, Pa., and the Mosaic Tile Co., Zanesville, Ohio. As the tests at the Beaver Falls plant were of a preliminary nature they will be discussed first.

PRELIMINARY TESTS.

The purpose of the preliminary tests at Beaver Falls was to determine the maximum proportion of Georgia clay that could be used in a commercial wall tile. The tests made at the Ohio State University showed that at least comparatively large proportions could be used in bodies of high absorption before the color was affected. What effect a large proportion would have on other pyrophysical properties of wall tile had next to be determined.

MATERIALS USED.

The materials used in this investigation were as follows:

English china clay.—Moore & Munger's M. W. M. No. 2.
Ball clay.—Kentucky No. 4.
Florida kaolin.—Edgar Plastic Kaolin Co.
Flint.—Pennsylvania Pulverizing Co.
Feldspar.—Pennsylvania Feldspar Co.
Treated Georgia kaolin.—Georgia Kaolin Co.

As these tests were to form the basis of larger commercial tests at the Mosaic Tile Co. plant, it was thought advisable to determine the fineness of grain of the flint and feldspar in order that the shrinkage and absorption could be controlled on changing to a different feldspar and flint. The mechanical analyses were made with a Schultze elutriation apparatus[a] having an overflow of 80° c. c. per minute, and are given in Table 6.

TABLE 6.—*Results of mechanical analyses of feldspar and flint.*

Residue.	Average diameter of grain.	Feldspar in residue.	Flint in residue.
Residue on sieves:	*Mm.*	*Per cent.*	*Per cent.*
On 120-mesh	0.186	9.46	0.60
On 150-mesh	0.157	0.90	0.42
On 200-mesh	0.110	7.50	1.56
On 260-mesh	0.0765	10.70	9.30
On 330-mesh	0.578	9.36	4.32
Residue from elutriator jars:			
In No. 1 jar	0.034	20.84	25.74
In No. 2 jar	0.0187	9.60	18.00
In No. 3 jar	0.010	10.80	19.06
Overflow from No. 3 jar	....	20.84	20.40

COMPOSITION OF BODIES.

The percentage composition of the bodies compounded from the above white-burning materials, and the physical properties of the bodies are shown in Table 7.

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[a] For description of apparatus, see Watts, A. S., Mining and treatment of feldspar and kaolin in the southern Appalachian region: Bull. 53, Bureau of Mines, 1913, p. 46.

TABLE 7.—*Composition and physical properties of bodies used in tests.*

COMPOSITION.

Item.	Series A, body No.—					Series B, body No.—			Series C, body No.—		
	10	11	12	13	14	15	16	17	18	19	20
	Per ct.	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*
Feldspar	15.0	15.0	15.0	15.0	15.0	13.0	12.0	11.0	12.0	12.0	12.0
Flint	40.0	40.0	40.0	40.0	40.0	40.0	40.0	40.0	40.0	40.0	40.0
Florida kaolin	10.0	10.0	10.0	10.0	10.0	11.0	11.0	12.0	......	......	......
Kentucky ball clay	15.0	15.0	15.0	15.0	15.0	16.0	16.0	15.0	15.0	15.0	......
English china clay	20.0	15.0	10.0	5.0	......	......	5.0	......	33.0	......	......
Treated Georgia kaolin	......	5.0	10.0	15.0	20.0	20.0	16.0	20.0	......	33.0	48.0
Whiting	......	......	......	......	......	......	......	1.0	......	......	......
Magnesite	......	......	......	......	......	......	......	1.0	......	......	......

PHYSICAL PROPERTIES.

Item.	10	11	12	13	14	15	16	17	18	19	20
Shrinkage	9.70	9.57	9.37	9.15	8.51	6.39	6.10	5.76	6.74	4.59	3.41
Absorption	6.75	7.52	7.98	8.70	9.65	12.44	12.65	12.27	10.02	13.86	17.70
Shade (color) [a]	D	D	D	D	D	B	B	B	C	C	A

[a] Comparative, A is the whitest shade, D the darkest.

PREPARATION OF BODIES.

The weight of each ingredient to be taken was determined from the percentage composition, figured on the basis of a 10-kilogram body, and the total weight was checked by mixing the ingredients and weighing the batch. The bodies were blunged for five hours in a small blunger constructed from an ice-cream freezer; then the slip was passed through a 120-mesh sieve and filter-pressed in a small laboratory press. The press cakes were dried at 110° C. and then ground to pass a 20-mesh sieve. Thirteen per cent of water was added to the ground body, which was then thoroughly mixed by hand, and the moist clay reground in a small dust mill to pass a 30-mesh sieve. The finely ground clay was stored in stoneware jars and allowed to stand overnight.

MAKING THE TILE.

The bodies were pressed into standard 6 by 3 inch tile on an ordinary hand screw press. In order to obtain uniform pressing, all bodies were pressed by the same experienced pressman. Thirty tile were pressed from each body and divided into six sets of five tile each for distribution in different sections of the kiln.

BISCUIT BURN.

The tile were set in saggers in the ordinary manner. Each sagger contained five tile of each body composition, and were distributed in the kiln as follows: A, bottom of first ring, just above bog wall; B, top of first ring; C, bottom of second ring; D, top of second ring; E, bottom of third ring; G, top of third ring.

A plate containing Seger cones Nos. 9, 10, 11, and 12 was placed in each sagger. The heat treatment that the bodies in different sections of the kiln received was as follows:

Temperatures in different sections of kiln, as shown by Seger cones.

Kiln section.	Cone No.	Kiln section.	Cone No.
A	13	D	10
B	11	E	9
C	11	G	12

The burn was made in an ordinary down-draft Rivers kiln. The variation in temperature shown in the above table was due to a change in burners.

GLOSS BURN.

One tile of each body composition from each section of the biscuit kiln was glazed with the regular glaze used by the company. The glaze was applied by dipping.

VITRIFICATION.

A study of the vitrification was made by the determination of average absorptions and shrinkages. The absorption was determined by soaking the biscuited tiles in distilled water for 48 hours and calculating the percentage of water absorbed by each tile on the basis of the dry weight of the tile. The shrinkage was determined by measuring the length of the burned tile and expressing the difference between the green length (length of die) and the burned length in terms of percentage of green length. The different heat treatments which the test pieces in different sections of the kiln received had a decided effect on the absorption and shrinkage. This may be emphasized by the actual results of tests with body 18, which were as follows:

Results of absorption and shrinkage tests with body 18.

Tile in kiln section—	Average absorption.	Average shrinkage.
	Per cent.	*Per cent.*
A	7.94	7.43
B	10.19	6.90
C	9.75	6.88
D	10.66	6.55
E	12.30	5.83
G	9.29	6.88

The average shrinkage and absorption for each body have been given in Table 7.

DETERMINATION OF COLOR.

As there is no standard method of measuring the shade or color of a wall tile, the shade of the glazed trials was determined by comparison. The results are given in Table 7, A being the whitest shade and D the darkest.

RESULTS OF TESTS.

The purpose of the tests in series A, Table 7, was to study the effect of substituting treated Georgia kaolin for English china clay. The results show that as the content of Georgia kaolin was increased the vitrification was decreased, but no change in color was noted. All the bodies of this series had too high a shrinkage and too low an absorption for practical wall tile, as most commercial wall tile have a shrinkage of about 4.5 per cent and an absorption of 14 per cent.

The tests in series B, Table 7, were designed to decrease the vitrification shown by members 13 and 14 of series A. The data show that the shrinkage was reduced and the absorption increased, but not sufficiently to approximate those of commercial wall tile. An improvement in color, owing to decreased vitrification, was noted. Introducing 1 per cent of whiting and 1 per cent of magnesite did not improve the color of the tile.

The purpose of series C, Table 7, was to further reduce the vitrification by substituting treated Georgia clay for Florida kaolin and ball clay. The results speak for themselves. In bodies 18 and 19 33 per cent of treated Georgia kaolin was substituted for 33 per cent of English china clay without impairing the color of the glazed product. Another interesting fact was the improvement in color caused by substituting treated Georgia kaolin for ball clay in body 20, and in spite of the fact that the content of Georgia kaolin was 48 per cent of the body composition. The shrinkage and absorption of body 19, consisting of 12 per cent feldspar, 40 per cent flint, 15 per cent Kentucky ball clay, and 33 per cent treated Georgia kaolin, approximated that of commercial wall tile, and was therefore used as a basis for further investigations of wall-tile bodies.

LARGE-SCALE TESTS.

In the preliminary tests at the Beaver Falls plant the treated Georgia kaolin gave such promise of being an ideal clay for wall-tile manufacture that large commercial tests were thought advisable. These tests were carried on at the plant of the Mosaic Tile Co., Zanesville, Ohio.

MATERIALS.

The materials used in this investigation were as follows:

Flint.—Pennsylvania Pulverizing Co.
Feldspar.—Pennsylvania Pulverizing Co.

Florida kaolin.—Edgars Plastic Kaolin Co.
Ball clay.—Cooley ball clay.
Treated Georgia kaolin.—Georgia Kaolin Co.

COMPOSITION OF BODIES.

The percentage composition and physical properties of the bodies included in this investigation are given in Table 8.

TABLE 8.—*Composition and physical properties of bodies used in large-scale tests.*

COMPOSITION.

Item.	Series A, body No.—		Series B, body No. —									Series C, body No. —
	43	44	45	46	47	48	49	50	51	52	53	54
Flint............per cent..	40.0	40.0	40.0	40.0	40.0	40.0	40.0	40.0	37.5	35.0	35.0	35.0
Feldspar............do....	13.0	00.0	13.0	14.0	14.0	14.0	15.0	16.0	14.0	14.0	14.0	13.0
Cornwall stone.......do....	0.0	14.0	0.0	......	......	......	......	......	......	......	......	......
Florida kaolin.......do....	12.0	11.0	12.0	11.0	6.0	......	......	......	......	......	......	......
Ball clay............do....	0.0	0.0	0.0	......	5.0	11.0	5.0	......	13.5	16.0	10.0	14.0
Treated Georgia kaolin, per cent................	35.0	35.0	35.0	35.0	35.0	35.0	40.0	14.0	35.0	35.0	41.0	38.0

PHYSICAL PROPERTIES.

Shrinkage........per cent..	3.00	3.00	5.28	5.44	5.92	6.40	6.56	7.20	6.88	7.84	7.52	4.67
Absorption..........do....	17.70	18.00	17.09	17.19	15.10	14.25	14.19	15.19	14.74	14.21	15.37	14.40
Breaking strength.pounds..	45.0	52.0	55.1	55.5	68.5	81.0	71.7	76.5	85.0	100.8	87.0	70.0
Relative shade (color).....	A	A	......	......	......	......	......	......	......	......	......	D

PREPARATION OF BODIES.

The bodies in series A and C, consisting of 3,000-pound batches, were prepared as follows: The total batch was blunged in a double blunger for one hour in the ordinary wet way and then the slip was passed through a 120-mesh vibrating screen into an agitator, from which it was pumped into the filter presses. The filter-press cakes were dried in a tunnel drier heated by the waste heat from cooling kilns and then crushed into small pieces in a set of rolls. About 11 per cent of water by weight was added to the crushed body and after standing for several hours it was finely ground in an ordinary dust mill of the fan type. The finely ground clay was collected in a large storage bin; from this bin it was taken to the presses.

The batch weight of the bodies in series B was only 8,000 grams, and they were prepared as follows: (1) The ingredients were weighed out and the total weight checked by weighing the batch. The clay was (2) blunged in porcelain-lined ball mills for 1 hour, (3) sieved through a 120-mesh screen, and (4) the slip concentrated in plaster absorption bowls lined with muslin, and dried completely in a drier. The dried clay was (5) crushed to pass a 20-mesh screen, (6) tempered by the addition of 11 per cent of water by weight, and the tempered clay was (7) ground to pass a 20-mesh screen, and (8) allowed to stand for several hours in a covered jar.

MAKING THE TILE.

The bodies were pressed into 3 by 6 inch and 6 by 6 inch tile on the power presses of the tile company.

BURNING.

The tile after being allowed to dry for about four days were set in saggers and biscuit burned at cone 11. The biscuited tile were then glazed with the regular glaze used by the company and gloss burned at cone 02.

RESULTS OF TESTS.

The absorption, shrinkage, and color data given in Table 8 were obtained in the same manner as in the preliminary tests. The breaking strengths given in this table were obtained by determining the weight that would break the biscuited tile with one end gripped between two steel plates. The weight was suspended 2½ inches from the point of support.

The object of the tests in series A, Table 8, was to compare the effects of feldspar and Cornwall stone in wall-tile bodies. The data in the table show that it takes 1 per cent more Cornwall stone to produce the same vitrification, but the stone increases the strength of the body about 15 per cent. The bodies of this series had too high an absorption, too low a shrinkage, and too low a breaking strength for good commercial wall tile.

The purpose of the tests in series B, Table 8, was to obtain a body of higher breaking strength. By comparing body 45 of this series with body 43 of series A it is seen that grinding a body in a ball mill, instead of blunging it, increased the shrinkage but had little effect on the absorption. The breaking strength was greatly increased by substituting ball clay for Florida kaolin, and by increasing the clay content at the expense of the flint a further increase in strength was obtained, reaching a maximum of 100.8 pounds in body 53. As none of the bodies of this series was gloss fired no relative shade or color is given for these bodies.

A large commercial test was made on a body of the composition of No. 54, in series C of Table 8. This body proved to be a good commercial wall-tile body. The low breaking strength, 70 pounds, is due to the fact that the finely ground clay was too dry when pressed. The color of the glazed tile was as good as that of some of the leading wall tile on the market.

APPLICABILITY OF REFINED GEORGIA KAOLIN IN WALL-TILE BODIES.

Treated Georgia kaolin can be substituted for all the English china content and a part of the ball-clay content in clay bodies without impairing the color of the burned product.

Cornwall stone makes a stronger body than feldspar, but a feldspar wall-tile body can be made that is as strong as any commercial wall tile on the market, provided that the total clay content is at least 52 per cent, of which 10 per cent is ball clay and the other 42 per cent is treated Georgia kaolin.

ACKNOWLEDGMENT.

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INDEX.

A. Page.

- Absorption of vitreous china bodies......... 42
  - of wall tile bodies......... 48, 51, 52
    - determination of......... 49
- Acetic acid, as coagulating agent......... 20
    - curves showing......... 19
- Acids. *See* acids named.
- Alkalies, as deflocculating agents......... 15-17
  - *See also* alkalies named.
- Alum, as coagulating agent......... 16, 23, 25, 32
    - curves showing......... 19, 22, 28
- American clays, white-burning, demand for.. 7
  - *See also* Georgia; South Carolina.
- Ammonia, as deflocculating agent......... 16
- Ashley, H. E., on deflocculating agents...... 17, 26
  - tests of kaolins by......... 8, 17

B.

- Ball clay, analysis of......... 38
  - in wall tile bodies......... 47
    - effect on breaking strength......... 52
    - proportion of......... 48, 51
  - in white-ware bodies......... 38, 39, 41, 42
- Bath, S. C., kaolin near, mining of......... 8, 9
- Beaver Falls, Pa., wall tile made at, tests of. 46-50
- Biscuit burning of wall tile......... 48, 49, 52
  - of white-ware bodies......... 39, 40, 43, 44
- Bleininger, A. V., cited......... 19, 26
  - on purification of clays......... 17, 35
- Blunger. *See* Washer.
- Blunging, description of......... 28, 29
  - double, need of......... 29, 36
  - effect of, on viscosity......... 22, 24, 25
    - figure showing......... 22
- Briggs, L. J., investigations of......... 16
- Brongniart, A., on potash in pottery bodies.. 15
- Brown, G. H., on purification of clays....... 17
- Bureau of Mines, investigation of......... 7
- Burning of white-ware bodies. *See* Biscuit burning; Gloss burning.

C.

- Caustic soda, as deflocculating agent......... 17, 19, 20, 24, 26, 28-30
    - curves showing......... 19
    - factors governing......... 36
  - cost of......... 36
- Caustic-soda solution, proper strength of..... 31
  - rate of flow of......... 30
  - supply tank for, construction of......... 29
- Caustic soda and sulphuric acid process of refining kaolins, cost of......... 35, 36
  - factors in......... 36, 37
- Chemist in kaolin refining, need of......... 36, 37
- China, vitreous, tests in making......... 41-46

Page.

- China bodies, vitreous, breaking strength of. 51, 52
    - effect of Georgia kaolins in......... 44, 46
    - physical porperties of......... 48
    - preparation of......... 48, 51
    - results of tests of......... 44
- China clay. *See* Clay; English china clay; Kaolin.
- Clay in slip, proportion of, factors determining 23
  - tests of......... 21
  - variations in......... 21
- Clays, physical properties of, variation in.... 10
  - predrying of, effect of......... 25
    - figure showing......... 22
  - purification of, methods for......... 15-17
  - removal from pit, factors governing...... 12
  - *See also* Georgia kaolin; Kaolin.
- Coagulating agents. *See* Acetic acid; Alum; Hydrochloric acid; Silicic acid; Sulphuric acid; Tannic acid.
- Color of Georgia kaolins, tests of......... 38-40
  - of vitreous china body, effect of Georgia kaolins on......... 45, 46
  - of wall-tile bodies......... 48, 51
    - determination of......... 50
- Concentration of slip, method for......... 33
- Concentration tank, design of, figure showing. 34
- Cornwall stone in wall-tile bodies, proportion of......... 51

D.

- Deflocculating agents. *See* Ammonia; Caustic soda; Potash; Sodium carbonate; Sodium silicate.
- Disintegration of clay, need of......... 27, 28
- Disintegrator, type of......... 28
- Dispersion, description of......... 28, 29
- Drag scraper for removal of overburden...... 11
  - view of......... 12
- Drying sheds, description of......... 12
  - views of......... 14

E.

- Electrolytes. *See* Electrolytes named.
- Elutriation apparatus, use of......... 18
- Elutriation process for refining kaolin....... 27-35
- Elutriation troughs. *See* Troughs.
- England, purifying of kaolins in......... 35
- English china clay, analysis of......... 38
  - in vitreous china bodies......... 48
  - in wall-tile bodies......... 47
  - in white-ware bodies......... 38, 39, 41, 42
  - shrinkage of......... 38

F. Page.

- Feeder, type of 28
  - view of 28
- Feeding of clay and water, uniform, need of. 36
- Feldspar, analysis of 38
  - in vitreous china body, proportion of 45
  - in wall-tile bodies 48, 51, 53
  - in white-ware bodies 38, 39, 41, 42
  - mechanical analysis of 47
- Filter-press cakes, crushing of 35
  - drying of 35
- Filter-pressing of slip, difficulties in 33, 35
- Flint, analysis of 38
  - vitreous-china bodies 48, 51
  - in wall-tile bodies 47
  - in white-ware bodies 38, 39, 41, 42
  - mechanical analysis of 47
- Flocculating agents, need of 32
  - *See also* Acetic acid; Alum; Hydrochloric acid; Silicic acid; Sulphuric acid; Tannic acid.
- Flocculation, effect of alkalies on 20
- Florida kaolin, analysis of 38
  - in wall-tile bodies 48, 51
    - effect of, on breaking strength 52
  - in white-ware bodies 38, 39, 41, 42
- Förster, F., on effect of alkalies on clays 16
- Frit in white-ware bodies, composition of 43

G.

- Georgia, kaolin in, extent of 7, 8
  - *See also* localities named.
  - mechanical analyses of 18
- Georgia kaolin, treated, in vitreous-china body, effect of 44-46
  - in wall-tile body 47, 48, 50, 51
  - in white-ware bodies 38, 39, 41, 42
  - shrinkage of 38
  - tests of 37, 38, 40, 41
  - value of 53
  - washed, analysis of 38
    - in white-ware bodies 38, 39, 42
    - shrinkage of 38
    - tests with 40
- Glaze on white-ware bodies, composition of .. 40, 43
- Gloss burning of wall tile 49, 52
  - of white-ware bodies 40, 44
- Goetz, Karl, on use of soda in casting porcelain 15
- Ground glass, effect of, on clays 15

H.

- Hagedorn, Robert, on effect of alkalies on clay 15
- Hartmann, C., on use of alkalies in casting .. 15
- Hirsch, H., cited 16
- Howat, W. L., on purification of clays 17
- Hydrochloric acid as flocculating agent 17, 20
  - figure showing 19

J.

- Jointing of clay, effect of, on mining methods. 11, 12
  - irregular, view of 12

K.

- Kaolin, composition of 14
  - minerals associated with 14
    - removal of 14
- Kaolin. *See also* Clay; Georgia kaolin.
- Kaolin mine, strata in, figure showing 11
- Kosmann, —., on effect of alkalies on clay ... 15

L.

- Laney, F. B., on minerals associated with kaolin 14
- Langley Station, S. C., kaolin near, mining of. 9
- Lottermoser, Alfred, on effect of alkalies on clay 15

M.

- McCallie, S. W., on composition of kaolin 14
- Magnesite in vitreous china body, effect of ... 45, 48
- Mellor, J. W., cited 16
- Moisture in clay, effect of, figure showing 22

O.

- Overburden, removal of 11
  - view showing 10, 12

P.

- Parsons, C. L., acknowledgment to 53
- Pick and shovel, mining of kaolin with 11
- Pit, mine, removal of clay from 12
- Potash, as deflocculating agent 15

R.

- Refining of clays. *See* Washing.
- Refining plant for kaolin, view of 14
- Rieke, Reinhold, cited 16

S.

- Salts. *See* salts named.
- Schultze elutriation apparatus. *See* Elutriation apparatus.
- Schwerin, —, purification of clays by 17
- Screening of slip, need of 33
- Sedimentation, effect of alkalies on 20
- Sedimentation process for refining kaolin 26, 27
- Shrinkage of kaolins, determination of 37, 38
  - of vitreous china bodies 42
  - of wall tile bodies 48, 49, 51, 52
- Silicic acid, as flocculating agent 15
- Simonis, M., cited 16
- Sliming process. *See* Elutriation process.
- Sloan, Earle, cited 8
- Soda, effect of, on clays 15
- Sodium carbonate, as deflocculating agent ... 16, 20
  - curves showing 19
- Sodium hydroxide. *See* Caustic soda 19
- Sodium silicate as deflocculating agent 17, 20
  - curves showing 19
- South Carolina, kaolin deposits in, extent of. 7, 8
  - *See also* localities named.
  - kaolins from, tests of 17
- Spangenberg, Albert, cited 16
- Steam shovels for removing overburden 11
  - view of 10
- Storage shed for clay, need of 27, 28
- Sulphates, effect of, on clays 16
- Sulphuric acid as flocculating agent 33, 37
  - curves showing 19
  - cost of 36
  - treatment of kaolins by, cost of 25, 36

T. Page.

Tannic acid, as flocculating agent............ 16
Tennessee ball clay. *See* Ball clay.
Tile. *See* Wall tile.
Tramway, overhead, for removal of clay..... 13,14
    view of.............................. 10
Troughs, elutriation, construction of......... 31
  floating, design of, figure showing........ 32
  settling, construction of................. 37

V.

Veatch, Otto, cited.......................... 8,11
Viscosimeter, description of.................. 19
Viscosity of slip, constant, need of........... 36
  determination of, factors governing...... 19
  factors affecting.................... 18,23-25,28
  permissible, determination of............ 23
  See also Deflocculating and flocculating agents named.
Vitreous china. *See* China.
Vitreous china bodies. *See* China bodies.

W.

Wall tile, burning of...................... 48,49,52
  manufacture of.......................... 48,52
    materials used in..................47,50,51
    tests in.............................. 46-50

Page.

Washer, concrete, design of, figure showing.. 30
    view of.............................. 28
  feeding of clay and water into............ 36
  types of................................ 29
Washing of clay, operations in............... 13
  See also Blunging.
Washing plant, capacity of, factors determining.. 23
  plan of, view showing.................... 36
Watts, A. S., acknowledgment to............ 53
  cited................................... 13
Weber, E., on effect of alkalies on clays...... 15,16
White-ware bodies, composition of........... 42
  materials used in..................... 38,41,42
  physical properties of.................... 42
  preparation of......................... 39,43
Whiting, analysis of......................... 38
  in vitreous china body................... 45
  in wall tile body......................... 48
Whiting in white-ware bodies................ 38,49

Z.

Zanesville, Ohio, wall tile made at, tests of 46,50-52
Zebisch, —., on effect of bases on clays....... 15

Bulletin 129

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BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

THE FUSIBILITY OF COAL ASH AND THE DETERMINATION OF THE SOFTENING TEMPERATURE

BY

ARNO C. FIELDNER, ALBERT E. HALL
AND
ALEXANDER L. FEILD

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CONTENTS.

	Page.
Introduction	1
Purpose of investigation	1
Scope of investigation	1
Historical review	2
Eugene Prost, 1895 and 1897	2
Henry Le Chatelier, 1902	3
J. W. Cobb, 1904	3
E. G. Bailey and W. B. Calkins, 1910 and 1911	4
L. S. Marks, 1910	5
E. J. Constam, 1913	6
L. S. Marks, 1914	7
F. C. Hubley, 1914	9
O. W. Palmenberg, 1915	11
E. B. Ricketts, 1915	12
Composition of ash and distribution of ash-forming constitutents in coal	13
Intrinsic ash	13
Extraneous ash	14
Nature of fusion of ash	15
Relation of melting points of ash constituents to fusion in the mixture	16
Formation of compounds and eutectics as illustrated in lime-silica system	17
Determination of the melting interval	18
Relation of melting interval to clinker formation	18
Determination of softening temperature	19
Factors influencing softening temperature	20
Influence of iron oxides	20
Experiments with Seger cones	22
Description of furnaces	22
Description and analyses of coal samples used in tests	25
Preparation of ash	29
Preparation of cones	29
General method of heating	30
Temperature measurements	30
Summary of results obtained in the different furnaces	32
Variations due to size and shape of cone	32
Influence of fineness of ash	33
Influence of inclination of cones	33
Influence of rate of heating	35
Influence of dextrin binder	36
Influence of various atmospheres	37
Appearance of cones softened in various atmospheres	42
Platinum furnace, atmosphere of air	42
Molybdenum furnace, atmosphere of hydrogen	43
Northrup furnace, atmosphere of carbon monoxide and nitrogen	43

Page.

- Standard methods for determining softening temperature of ash 44
- Fusibility of coal ash in mixtures of hydrogen and water vapor 44
  - Theoretical consideration 44
  - Outline of experiments 46
    - Description of furnace 46
    - Description of "saturator" 48
    - Preparation of test piece 49
    - Measurement of temperature 49
  - Results of experiments 49
- Fusibility of coal ash in mixutres of carbon monoxide and carbon dioxide 53
  - Theoretical considerations 53
  - Method of conducting experiments 54
    - Preparation of gas mixtures 55
    - Results of experiments 56
    - Discussion of results 62
- Relation of softening temperatures in atmospheres of hydrogen and water vapor, and of carbon monoxide and carbon dioxide to clinker formation in furnaces 63
  - Ferrous, ferric, and metallic iron in fused ash cones and clinker slags 64
- Relative merits of $CO-CO_2$ and H_2-H_2O atmospheres as standard atmospheres for fusion tests 67
- Modified molybdenum furnace and saturator for softening-point determinations in H_2-H_2O atmosphere 74
  - Description of furnace and saturator 74
- Experiments with vertical and horizontal placing of cones in determination of softening temperature 79
  - Method of conducting experiments 80
  - Results of experiments 81
- Effect of dextrin binder or residual carbon on softening temperature of ash, and its relation to intumescence of softening cone 86
  - Swelling of melting ash cones 86
  - Experiments to determine effect of unburned carbon in ash on softening of cone 86
- Molybdenum furnace as standard furnace for fusion tests 89
- Fusibility of coal ash as determined in gas furnace 90
  - Theoretical considerations 90
  - Description of furnace 90
  - Method of operation 92
  - Composition of atmosphere surrounding cones 93
  - Testing for magnetic slag and metallic iron 95
  - Temperature measurements 95
  - Results of tests 96
  - Comparison of results obtained in two different types of gas furnaces operated under maximum reducing conditions 96
  - Comparison of results obtained with natural and with artificial gas 98
  - Ignition of ash in oxygen 101
  - Influence of size and shape of cone 102
    - Results of experiments 104
  - Influence of rate of heating 107
  - Determination of fluid temperature 108
  - Discussion of results obtained in gas furnace with reducing atmosphere 110

Page.
Recommended procedure for standard gas-furnace method — 111
  Preparing the ash — 111
  Preparation of cones — 111
  Method of heating — 112
Summary and conclusions — 114
Acknowledgments — 117
Bibliography — 119
  Fusibility and clinkering of coal — 119
  Contributions to the physical chemistry of slag and silicates — 123
  Melting and softening temperatures of oxides, sulphides, silicates, mixtures, and slags — 130
  Pyrometric cones; their use in measuring temperature — 134
  Pyrometers and the measurement of high temperature — 135
  Laboratory furnaces for high-temperature work — 136
Publications on the utilization of coal and lignite — 139
  Publications available for free distribution — 139
  Publications that may be obtained only through the Superintendent of Documents — 140
Index — 142

TABLES.

Table 1. Character and source of coal samples — 26
2. Analyses of ash and coal — 27
3. Results of tests showing influence of size of cone on softening temperature — 32
4. Influence of fineness of ash on softening temperature — 33
5. Influence of inclination of cones on softening temperature — 33
6. Results of tests showing comparative softening temperatures in molybdenum furnace No. 2 at two different rates of heating — 36
7. Results of experiments to determine influence of dextrin binder on softening temperature of ash cones — 37
8. Results of tests to determine influence of various atmospheres on softening temperatures of ash cones in different furnaces — 42
9. Results of experiments to determine softening temperatures of ash in various mixtures of hydrogen and water vapor — 50
10. Results of tests to determine softening temperatures of certain ash samples in an atmosphere containing equal parts of CO and CO_2 — 61
11. Results of tests to determine comparative effects on softening temperatures of ash of atmospheres of air, of water vapor, of carbon dioxide, of carbon monoxide, of equal parts of hydrogen and water vapor, and of equal parts of carbon monoxide and carbon dioxide — 63
12. Results of analyses of ash cones fused in various atmospheres and of furnace clinkers — 65
13. Results of tests to determine softening temperatures of coal ashes containing various percentages of ferric oxide in various atmospheres — 68

Page.

TABLE 14. Results of tests to determine softening temperatures of standard pyrometric cones in H_2–H_2O and CO–CO_2 atmospheres, and in air ---- 72
15. Results of softening-temperature tests, showing relative effects of vertical and horizontal placing of cones ---- 82
16. Results of analyses of coal ashes and synthetic mixtures tested with the cones used in the vertical and in the horizontal position ---- 82
17. Results of analyses of coal ashes used in experiments to determine influence of carbon on the softening temperature and the intumescence of ash cones ---- 87
18. Results of tests to determine influence of carbon on the softening temperature and on the inteumescence of ash cones ---- 88
19. Results of analyses of gas samples taken at various temperatures from No. 3 melter's furnace ---- 93
20. Composition of furnace atmosphere at various temperatures ---- 94
21. Results of tests to determine softening temperatures in No. 3 melter's furnace and in modified Meker furnace ---- 98
22. Results of tests to determine softening temperatures in No. 3 melter's furnace with four different sizes of cones ---- 104
23. Results of tests to determine softening temperatures in No. 3 melter's furnace at four different rates of heating ---- 107
24. Results of tests to determine initial-deformation, softening, and fluid temperatures of 12 samples of ashes in No. 3 melter's furnace ---- 110

ILLUSTRATIONS.

PLATE I. *A*, Typical forms of cones fused in air in the platinum furnace; *B*, Typical forms of cones fused in hydrogen in the molybdenum furnace; *C*, Typical forms of cones fused in carbon monoxide in the Northrup furnace ---- 42
II. Arrangement of silica-tube furnace and accessories ---- 46
III. *A*, Clinker No. 20452, from grate of Roney stoker; *B*, Typical forms of cones fused in the No. 3 melter's furnace ---- 66
IV. *A*, Molybdenum furnace and saturator; *B*, No. 3 melter's furnace arranged for fusing tests ---- 74
FIGURE 1. Concentration-temperature diagram for the lime-silica system ---- 17
2. Sections through platinum-wire resistance furnace ---- 22
3. Meker muffle furnace No. 29 ---- 22
4. Molybdenum furnace ---- 23
5. Granular-carbon resistance furnace ---- 24
6. Arrangement for softening temperature test in Northrup furnace ---- 25
7. Brass cone mold ---- 29
8. Appearance of ash cones at various stages of softening ---- 31
9. Curves showing results of tests to determine influence of rate of heating on the softening temperature of ash cones in an atmosphere of hydrogen ---- 34

Page.

FIGURE 10. Curves showing results of tests to determine influence of rate of heating on the softening temperature of ash cones in an atmosphere of air 35
11. Curves showing results of tests to determine softening temperatures in Meker furnace, muffle furnace No. 2, and platinum furnace 38
12. Curves showing results of tests to determine softening temperatures in Northrup, carbon-resistance, and platinum furnaces 40
13. Curves showing results of tests to determine softening temperatures in Northrup, platinum, and molybdenum furnaces 41
14. Equilibrium diagram of the system H_2, H_2O, Fe, FeO, and Fe_3O_4 46
15. Vertical longitudinal section through silica-tube furnace 46
16. Front elevation of silica-tube furnace and vertical section through saturator 47
17. Curves showing softening temperatures (final deformation points) of coal ash in various mixtures of hydrogen and water vapor 51
18. Curves showing initial deformation temperatures of coal ash in various mixtures of hydrogen and water vapor 52
19. Equilibrium diagram of the system CO, CO_2, Fe, FeO, and Fe_3O_4 54
20. Curves showing softening temperatures of ash sample 16 in mixtures of CO and CO_2 and of H_2 and H_2O 57
21 Curves showing softening temperatures of ash sample 5 in mixtures of CO and CO_2 and of H_2 and H_2O 58
22. Curves showing softening temperatures of ash sample 12 in mixtures of CO and CO_2 and of H_2 and H_2O 59
23. Curves showing softening temperatures of ash sample 9 in mixtures of CO and CO_2 and of H_2 and H_2O 60
24. Curves showing softening temperatures of coal ash in gas furnace No. 3, in atmospheres of equal parts of H_2 and H_2O, and of equal parts of CO and CO_2, and in air, with special reference to the percentage of Fe_2O_3 in the ash 70
25. Curves showing initial deformation points of coal ash in gas furnace No. 3, in atmospheres of equal parts of H_2 and H_2O, equal parts of CO and CO_2, and in air, with special reference to the percentage of Fe_2O_3 in the ash 71
26. Vertical longitudinal section through molybdenum furnace No. 3 75
27. Front elevation of molybdenum furnace No. 3 and vertical section through saturator 76
28. Appearance of vertical and horizontal ash cones at various stages of softening 83
29. Appearance of vertical and horizontal ash cones at various stages of softening 84
30. Appearance of vertical and horizontal ash cones at various stages of softening 85
31. Section of No. 3 melter's furnace arranged for fusion tests 91
32. Electromagnet used to determine magnetic properties of fused cones 95

Page.
FIGURE 33. Modified No. 29 Meker furnace 97
34. Curves showing softening temperatures determined in No. 3 gas furnace and in modified No. 29 Meker furnace 99
35. Electric furnace for igniting ash in atmosphere of oxygen 102
36. Curves showing softening temperatures obtained in No. 3 melter's furnace with four different sizes of cones 105
37. Curves showing softening temperatures obtained in No. 3 melter's furnace at four different rates of heating 106
38. Curves showing temperatures of initial deformation and softening and fluid temperatures of 12 ashes in No. 3 melter's furnace 108

THE FUSIBILITY OF COAL ASH AND THE DETERMINATION OF THE SOFTENING TEMPERATURE.

By Arno C. Fieldner, Albert E. Hall, and Alexander L. Feild.

INTRODUCTION.

PURPOSE OF INVESTIGATION.

As a safeguard against excessive clinker troubles, specifications for the purchase of coal can be drawn to include the "softening" or "fusing" temperature of the ash. The value of such information has been recognized by the Bureau of Mines and has induced the bureau to investigate laboratory methods of determining the fusibility of coal ash, and the bearing of the results on clinker formation in fuel beds. The bureau realized that the fusibility of a mixture of oxides and silicates such as is comprised in coal ash varies according to the conditions under which tests are made, and that in the absence of any generally accepted method, no agreement is to be expected among tests made in different laboratories. Indeed, Marks[a] has recently called attention to differences as great as 390° C. that were obtained with the same sample of ash by two different laboratories. Therefore if a fusion-temperature clause is to have any place in specifications for coal, a standard method of testing should be adopted in order to insure comparable results by different laboratories and to obtain the softening or fusing temperature of the ash under conditions similar to those of a fuel bed. After a consistent method has been devised for obtaining comparable results at different laboratories, there will remain the correlation of these results with those from the burning of coal in furnaces before the value of fusibility tests in coal specifications can be finally determined.

SCOPE OF INVESTIGATION.

The present publication reviews the literature on the subject and gives in detail the effect of various oxidizing, reducing, and neutral atmospheres such as are found in various parts of the fuel bed on the softening temperature of ash when molded in the form of Seger cones. As a result of this study a practical method of determining fusibility has been developed whereby the ash is caused to soften and form slags in which the iron exists in approximately the same state of oxidation as the iron in fuel-bed clinkers.

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[a] Marks, L. S., The clinkering of coal: Power, vol. 40, 1914, pp. 932–934.

Although it is believed that this method will indicate the probable clinkering characteristics of a coal better than any fusion tests heretofore described, the bureau does not recommend the general use of this method in coal specifications until it is justified by actual trial in furnace tests of different coals. As no fusibility test of a well-mixed sample of the average ash of a coal takes into account the physical distribution of the impurities in the coal as burned, much additional investigation is needed to establish the exact relation of these laboratory fusibility tests to clinker formation. In order to correlate this proposed test of minimum softening temperature with clinker formation, the bureau is now making clinkering tests in a specially designed experimental furnace, in which different coals having ashes that differ in fusibility are burned under similar and known conditions of combustion. The results of this investigation will be published in a subsequent report. In the meantime it is hoped that other investigators will obtain similar correlation data.

HISTORICAL REVIEW.

EUGENE PROST, 1895 AND 1897.

Prost[a] analyzed the ash and determined the temperature of fusion of 23 different types of Belgian coals. The fusion tests were made by placing 3 grams of finely pulverized ash in a small porcelain crucible and then heating it for definite periods of time at four different temperatures, namely, 1,100°, 1,250°, 1,400°, and 1,500° C. The temperatures were controlled by using gold and platinum alloys of known melting points. After each period of heating the ash was examined. The progress of fusion was described as "barely fritted," "fritted," "strongly fritted," "incipient fusion," or "fused."

To obtain the different temperatures Prost used three different furnaces, as follows: (1) A Perrot gas furnace which gave a maximum temperature of 1,100° C.; (2) a coke furnace which gave a maximum temperature of 1,250° or 1,400° C., according to the method of operation; (3) a small blast furnace of fire brick which gave a maximum temperature of 1,500° C.

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[a] Prost, Eugene, Récherches sur les relations existant entre le degré de fusibilité et la composition des cendres de houille: Rev. Univ. Min., t. 31, 1895, pp. 87–98; Monit. Sci., t. 46, 1895, pp. 560–565; Sur les relations entre la composition et la fusibilité des cendres de houille: Bull. Assoc. Belge chim., t. 11, 1897, pp. 119–126; The fusibility and composition of coal cinders: Coll. Guard., vol. 70, 1895, p. 796; The relation between composition and fusibility in coal ash: Coll. Guard., vol. 74, 1897, p. 602; Fusibility of coal ash determined by its composition: Coll. Guard., vol. 75, 1898, p. 473.

In furnaces 2 and 3 the crucibles containing the ash samples and the alloy-temperature indicators were placed in a fire-clay crucible with a wall 1 cm. thick. Coke was packed around this crucible. Prost calculated from the chemical analysis the refractory quotient:

$$Q = \frac{\dfrac{\text{O in Al}_2\text{O}_3}{\text{O in CaO} + \text{O in Fe}_2\text{O}_3 + \text{O in MgO}}}{\dfrac{\text{O in SiO}_2}{\text{O in Al}_2\text{O}_3}}$$

for 23 Belgian coal ashes; he found that in general the fusing temperature increased with the value of Q. There were, however, several exceptions to the calculated results, especially for ash of high iron content. Prost's method of determining relative fusibility was merely approximate; for the conditions of reduction or oxidation undoubtedly varied in his different furnaces. Also his estimate of the degree of fusion on mere observation of the appearance of the melt in a crucible would necessarily be rather indefinite, for judgment of different observers would vary.

HENRY LE CHATELIER, 1902.

Le Chatelier[a] used the Seger-cone method for determining the relative fusibility of different coal ashes. He burned the coal in a muffle at a low temperature, ground the ash fine, and then, using starch as binder, molded the ash into triangular pyramids 50 mm. (2 inches) high and 15 mm. ($\frac{9}{16}$ inch) wide at the base. These pyramids were ignited to destroy the organic binder, placed in a gas furnace, with a suitable number of Seger pyrometric cones, and slowly heated until they collapsed. The temperature at which the vertex of a pyramid became half the original height of the cone was taken as the softening temperature. Le Chatelier gives no data on the possibility of reducing gases from the furnace atmosphere acting on the iron oxide of the ash.

J. W. COBB, 1904.

Cobb[b] ground coal ash to pass a 100-mesh sieve or to an impalpable powder, and, using dextrin solution for binder, made the ash into four-sided pyramids 3 inches (76 mm.) high and 1 inch (25 mm.) square at the base. Each pyramid was mounted with one face vertical, and placed in a circular, down-draft, regenerative furnace, heated by eight burners on the periphery.

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[a] Le Chatelier, Henry, Étude sur la fusibilité des cendre des combustibles: Bull. Soc. L'Ind. Nat., t. 102, 1902, pp. 223–229. Abstract (Fusibility of the ashes of fuels) in Jour. Soc. Chem. Ind., vol. 21, 1902, p. 459.

[b] Cobb, J. W., Coal ash: Jour. Soc. Chem. Ind., vol. 23, 1904, pp. 11–13.

Temperature measurements were taken with Seger pyrometric cones and with a thermocouple. The temperature of the furnace was raised slowly until the tips of the cones started to bend. The rate of heating was then checked to almost zero to allow the cones to bend slowly through a small range of temperature. Cobb gives no exact figures for rate of heating, but calls attention to the necessity of a uniform rate to insure comparable results, and also states that an oxidizing atmosphere must be maintained in the space surrounding the cones.

E. G. BAILEY AND W. B. CALKINS, 1910 AND 1911.

In 1910 Bailey and Calkins[a] published a paper giving experimental data on the relation between the fusing temperature of ash and the formation of clinker. Tests were made in two hand-fired furnaces with several coals having ash of different fusing temperatures. All clinkers larger than about 1 inch were picked out of the refuse and the ratio of clinkers to the total ash in the coal burned was determined. The results, plotted in curves, showed a rapid increase in the ratio of ash fused to clinkers as the fusing temperature fell below 1,371° C. (2,500° F.). The clinker made from a lower fusing ash obstructed more grate area than the same weight of clinker formed from an ash of higher fusing temperature.

In a subsequent paper Bailey[b] describes two 24-hour boiler tests with two coals A and B. The fusing temperatures of the ashes of these coals were 1,510° C. (2,750° F.) and 1,310° C. (2,390° F.). On the basis of the total ash in the coal, coal A formed 20.2 per cent and coal B 53.3 per cent of clinker. Also, the clinker from coal B was much more troublesome in that the clinkers were thinner, thus obstructing about one-third more grate area than the same weight of clinker from coal A. In several boiler tests of coals, made by Prof. E. A. Hitchcock at the Ohio State University, two coals, C and D, having ash that fused at 1,260° C. (2,300° F.) and at 1,450° C. (2,640° F.), were burned on a plain grate of 50 per cent air space, under a Dutch oven. The ash with the lower fusing temperature had a greater proportion fused to clinker than did the ash with the higher fusing temperature; also the clinker from coal C was thin and vitreous, thus offering more obstruction to the passage of air through the fuel bed than the fragile and porous clinker resulting from coal D.

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[a] Bailey, E. G., and Calkins, W. B., Fusing temperatures of coal ash: Power, vol. 32. 1910, pp. 1978–1979. Fuel Testing Co. Bull. 5, 1911, 15 pp.

[b] Bailey, E. G., The fusing temperature of coal ash: Power, vol. 34, 1911, pp. 802–806; 1910, pp. 1978–1979.

Coals E and F, having ash that fused at 1,210° C. (2,210° F.) and 1,400° C. (2,550° F.), were burned on shaking grates under a firebrick arch of a special horizontal return tubular boiler setting. These two coals did not show the difference in clinker formation that was expected from the difference in the fusing temperatures of the ashes. This result is ascribed to a probable difference in fuel-bed conditions and a difference in the burning characteristics of the two coals. However, the lower fusing ash from coal E formed a thinner clinker than coal F, obstructing approximately 0.21 square foot of grate area per pound, as compared to 0.13 square foot for the clinker from coal F.

Bailey concludes that the "main factor in the formation of clinker is the difference between the fusing temperature of the ash and the temperature to which it is subjected. Any factor, such as excess air, rate of combustion, and thickness of fuel bed which will affect the temperature may cause a corresponding change in the amount and nature of the clinker from a given coal."

L. S. MARKS, 1910.

Marks[a] used the Seger-cone method for determining the effect on the fusibility of ash of adding various proportions of Al_2O_3, CaO, Fe_2O_3, FeS_2, Fe, CaS_2, and $CaSO_4$. The finely ground mixtures were molded into pyramids similar in shape and size to standard pyrometric cones, then mounted in a vertical position and slowly heated in a gas muffle. The temperatures were measured with a thermocouple. Sketches were made of the cones as they melted, and the corresponding temperatures were noted. The viscosity of the softening material was judged from the rapidity and manner of deformation of the cone. Certain mixtures melted to a fluid of the viscosity of water, others formed a spongy, viscous mass. Marks obtained the following results on adding various substances to the ash of a nonclinkering West Virginia coal, which had a softening temperature of 1,400° C. (2,550° F.):

Results of tests by Marks with the ash of a nonclinkering West Virginia coal in a gas muffle.

Substance added.	Effect on softening temperature of ash of adding—			Fluidity of melt.
	25 per cent.	50 per cent.	75 per cent.	
Al_2O_3	Raised	Raised	Raised	Very fluid.
CaO	Lowered considerably.		do	
Fe_2O_3	Lowered slightly	Lowered slightly	Lowered slightly	
FeS_2	Lowered considerably.	Lowered considerably.	Lowered considerably.	Viscous.
Fe	do	Lowered somewhat	Lowered slightly	
CaS_2	Lowered	Lowered considerably.	Lowered considerably.	Very fluid.
$CaSO_4$	do	do	do	Do.

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[a] Marks, L. S., The clinkering of coal; results of tests for effect of various constituents in the coal: Eng. News, vol. 64, 1910, pp. 623–626.

Similar clinkering tests were then made in a specially designed furnace, analogous results being obtained. Substances that lowered the softening temperature of the ash tended to increase the clinker formation. The addition of 3 per cent of pyrite (FeS_2) to a nonclinkering West Virginia coal containing 6 per cent ash produced a hard, heavy clinker, which had a strong tendency to stick to the grate bars, thus choking the draft. Adding 2.2 per cent of ferrous sulphide (FeS) caused a similar degree of clinker formation. The addition of 4.4 per cent of ferrous sulphide made a very fusible and extremely troublesome clinker, which flowed over and "froze" to the grate bars, seriously checking the draft. Addition of iron oxide with an iron content equivalent to that in the 3 per cent pyrite produced a somewhat less troublesome clinker than did the pyrite.

The tests seemed to indicate that, other things being equal, the sulphides of iron are more troublesome than the oxides.

E. J. CONSTAM, 1913.

Constam,[a] in 1913, reported the "melting temperatures" of the ash from 200 different samples of coal and coke from various European mines. His criterion of melting temperature was the deformation or softening point of the ash when molded in the form of a small pyramid. The pyramids were heated in a carbon-tube electric furnace having a magnesia tube closed at one end in which the ash pyramid was placed. The temperatures of incipient softening, intumescing, and complete fusion were measured with a Holborn-Kurlbaum optical pyrometer. Constam gives no information on the rate of heating, exact size and shape of the test piece, or the character of the atmosphere in which it was heated, in his published report. At the temperatures necessary to fuse coal ash the magnesia tube would not protect the ash from the reducing action of the carbon monoxide evolved from the heating element, unless a vigorous current of air was circulated through the interior of the tube. In the absence of complete information as to whether oxidizing or reducing atmospheres were used in the fusion tests, Constam's results can not be properly classified, but they gave excellent agreement of the softening temperatures reported for different samples of coal from the same source, and for samples of ash from a given coal and from its resulting coke.

As a result of his investigations, Constam concludes that (1) the proportion of ash does not influence the softening temperature;[b] (2) the softening temperature is characteristic of the seam; (3) the softening temperature is not affected by coking the coal or by the temperature of coking; (4) the softening temperatures of different ashes ranged from 1,150° to 1,700° C.

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[a] Constam, E. J., Ueber Schmelztemperaturen von Kohlenaschen: Ztschr. Ver. Gas- und Wasserfachmänner in Oesterreich-Ungarn, Oct. 15, 1913. Abstract in Jour. Gas Lighting, vol. 124, 1913, p. 522.

[b] This conclusion does not hold when the ash contains an excess of a single constituent, as pyrite or slate.

Regarding various fuels, he recommends temperatures above which the ash of each fuel should fuse, as follows: Coke for a central heating plant, 1,300° C.; anthracite for the same purpose, 1,400° C.; coal for use under boilers, 1,400° C.; coal for locomotives, 1,500° C.; and coal for gas producers, about 1,500° C., or below 1,200° C.

L. S. MARKS, 1914.

Following the publication of Bailey's papers in 1910 and 1911, the growing demand for the fusibility test in connection with the purchase of coal under specifications led to the development of such tests in different laboratories. Although most of these tests depended on some modification of the well-known Seger-cone method used by Le Chatelier and by Cobb, highly discordant results were obtained by different laboratories with samples of the same ash. In 1914, Marks[a] called attention to differences as high as 390° C. in the fusing temperatures reported by different testing laboratories for samples of the same ash. An investigation of the causes of these variations led to the conclusion that the most important factor was the atmosphere surrounding the ash. Using the Seger-cone method, Marks determined the softening temperatures of 11 samples of ash in a Hoskins carbon resistance furnace and in a Meker muffle furnace; for each sample the Hoskins furnace gave the higher softening points, the difference ranging from 120° to 255° C. This variation in fusibility was attributed to the Hoskins furnace containing a reducing atmosphere of carbon monoxide and the Meker furnace having an oxidizing atmosphere from the air circulating through the muffle. Marks considers it essential that fusion tests be made in an oxidizing atmosphere.[b]

As less important causes for variations in the results of Seger-cone tests, Marks enumerates the following: (*a*) Size of cone; (*b*) position of cone; (*c*) nature of binder; (*d*) rate of heating; (*e*) location of cone in furnace; (*f*) material on which the cone is supported; (*g*) unburned carbon in the ash.

The use of a 10 per cent dextrin solution for binder and slight variations in the size of the cone had no appreciable effect on the results.

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[a] Marks, L. S., The clinkering of coal: Jour. Am. Soc. Mech. Eng., vol. 37, 1915, pp. 205–208; Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 801–834.

[b] Combustion tests made for the Bureau of Mines by Henry Kreisinger, engineer, have shown that at ordinary rates of combustion almost all the oxygen in fuel-bed gases has disappeared at 3 inches above the grate bars, the gases at this point containing approximately 15 per cent CO_2 and 14 per cent CO, the remainder being nitrogen and other combustible gases. It is therefore highly probable that clinker forms in a slightly reducing atmosphere, and that fusion tests of coal ash in air do not parallel furnace conditions.

The deformation temperatures of 11 different samples of ash tested in horizontally mounted[a] cones were from 5° to 50° C. lower than when similar samples were tested in vertically mounted cones. The average difference was 21° C. The smaller differences were observed with samples showing the greatest fluidity. Cones mounted horizontally were more sensitive to heat than those mounted in the usual manner, and gave indication that could be duplicated more closely.

Comparative tests of the effect of rate of heating with two samples of ash showed deformation points 40° and 35° C. lower at 6° increase per minute than at 2° per minute. This difference was ascribed to the increased lag of the thermocouple and its protecting porcelain tube at the higher rate of heating.[b]

Unburned carbon in the ash tended to increase the softening temperature.

Marks finally adopted as the most satisfactory modification of the Seger-cone method the molding of the ash into a cone 11 by 52 mm. ($\frac{7}{16}$ inch by 2 inches), placed in a horizontal position on a quartz plate, with the apex projecting over the side of the support. The quartz plate was blocked up near the center of the muffle of a No. 29 Meker furnace. A stream of air provided an oxidizing atmosphere. A thermocouple, protected with a porcelain tube, was inserted through a hole in the back of the furnace, the cone being placed as near it as possible. The furnace was heated rapidly to within about 200° C. of the expected fusing temperature; the rate of temperature increase was then reduced to 2° C. per minute. The temperature of initial and of final bending (apex of the cone pointing vertically downward) was noted, and also the final appearance of the cone—whether it showed a fluid constituent that ran down and formed a knob on the vertex of the cone. This seemed to be somewhat characteristic of clinkering coals.

The results of tests in which the method of determining fusibility, described above was used, were compared with the clinkering results observed when 10 different coals were burned under normal power-house conditions. Marks, concludes that "a general relation was shown between the two, but not definite enough to be reliable. For the particular boiler plant investigated, it would seem that an ash with a fusing temperature (final bending) below 1,400° C. will probably give trouble if the ash cone shows a fusible constituent; whereas it will not give trouble with a fusing temperature above 1,380° C. if the ash is viscous. This conclusion would require further investigation with many other coals before it could be accepted even for this particular plant; naturally it can not be applied to plants with different operating conditions."

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[a] As recommended by J. P. Sparrow of the New York Edison Co.

[b] Higher rates of heating should give higher softening temperatures owing to the time lag in the slag-forming reactions. This relation has been shown in experiments with Seger pyrometric cones by numerous investigators. See Zimmer, W. H., Practical experience with pyrometers: Trans. Am. Cer. Soc., vol. 1, 1899, pp. 23–38; Geijsbeck, S., The fusing points of Seger cones expressed in degrees: Trans. Am. Cer. Soc., vol. 6, 1904, pp. 94–99; Geijsbeck, S., The melting points of pyrometric cones under various conditions: Trans. Am. Cer. Soc., vol. 14, 1912, p. 849; Hoffman, G., Prüfung der Segerkegel: Sprechsaal, Bd. 44, 1911, pp. 143–145; Reike, R., Die Schmelzpunkte der Segerkegel 022–15; Sprechsaal, Bd. 44, 1911, pp. 726–728; Brown, G. H., and Murray. G. A., The function of time in the vitrification of clays: U. S. Bur. Standards Tech. Paper 17, 1913, p. 23.

F. C. HUBLEY, 1914.

Various commercial laboratories that have developed individual modifications of the fusibility test seem to find a somewhat definite relation between the indications of their particular laboratory test and the clinkering characteristics of the fuel used in the power plants under their observation. Hubley,[a] after two years' experience with the Seger-cone method, considers it of value in predetermining the clinkering properties of a coal, and states that in most instances the clinkering of a coal in the boiler furnace agreed closely with the results of the laboratory tests.

Hubley used cones 2 inches high and 1 inch in diameter at the base. He ground the ash to pass a 60-mesh sieve; mixed it with 10 per cent by weight of dry dextrin and water enough to make a plastic mass, which was pressed in wooden molds, dried, and heated in a vertical position in a cylindrical Fletcher crucible furnace. Illuminating gas was used for heating to temperatures up to 2,700° F., and Blau gas for temperatures to 3,000° F. The temperatures were measured with a thermocouple, having the bare hot junction close to the cone. The temperature was gradually increased until signs of fusing appeared, as when the sides began to lose their sharp outlines, or when the body of the cone began to swell. The furnace was held at this temperature for three minutes, and then allowed to cool. Other cones from the same ash sample were in turn tested, the final temperatures for the lot being regulated to give a series at 25° or 50° F. intervals on either side of the temperature obtained in the preliminary test. Enough cones were tested to indicate a series from a temperature of no deformation to one at which the ash flowed readily. From this series the progress of fusion at various temperatures was determined.

This method gave satisfactory indication of probable clinkering characteristics, but proved tedious and somewhat lacking in definiteness, as the judgment of the operator entered too largely into the results.

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[a] Hubley, F. C., Bituminous coals; predetermination of their clinkering action by laboratory tests: Proc. Eng. Club of Philadelphia, vol. 32, January, 1915, pp. 35–85; abstracted in Power, vol. 40, 1914, p. 796; Jour. Am. Soc. Mech. Eng., vol. 37, 1915, pp. 208-211; Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 815–824.

To remedy these objections, Hubley devised an apparatus, which he called a "fusiometer," for indicating mechanically the progressive softening of a pellet of ash. This apparatus consists of an iron frame carrying a small gas-fired furnace, into which project two carbon rods, one-half inch in diameter on a common vertical axis. Between the carbon rods is placed the ash sample, compressed without binder, under a pressure of 30,000 pounds per square inch, into the form of a cylindrical pellet five-eighths of an inch in diameter and one-half inch high. The upper carbon rod, free to move vertically between suitable guides, is weighted to exert a pressure on the ash pellet of 1.5 pounds per square inch. A silk cord connecting the upper end of the rod and a counterweight runs over a pivoted pulley to which is attached a pointer magnifying twelve times on a scale any expansion or contraction of the pellet. The temperature of the furnace is measured with a thermocouple. Temperatures up to 2,700° F. may be attained with illuminating gas, and above 3,000° F. with Blau gas.

The method of making a test is described by Hubley as follows:

> The ash pellet is placed in position between the carbons and adjusted so that the pointer is midway between 0 and 1.00 on the scale. The furnace is then gradually heated at the rate of 50° to 100° F. per minute, simultaneous temperature and scale readings being made at ½-minute intervals. As the heat is increased, negative movement of the pointer will indicate carbon expansion up to a temperature where first softening of the ash pellet is indicated by a positive movement of the pointer. The experiment is continued till final collapse of the pellet is indicated on the scale. These results, if plotted, the temperatures as abscissæs, and the pointer movement as ordinates, produce a curve, an ordinate of which at any point is a measure of the relative rate of softening of the ash pellet at that temperature.
>
> The length of the softening range, and its position on the temperature scale in relation to the working temperature range of a boiler fire, as well as the increasing rate of softening, whether gradual throughout the range (indicating high viscosity), or very slight for most of the range, followed by a sudden collapse, are the factors in the fusiometer results which must be considered in predicting the probable clinkering action of a coal in a boiler fire. A long range ash with a slow but steady increase in softness, apparently indicated by the gently sloping curve, is productive of the close gummy clinker of high viscosity. This type, provided the softening range coincides approximately with the working temperature range of the boiler, produces the greatest losses from clinker formation in a boiler fire. This refers more particulaly to stokers in which the fires are cleaned at regular intervals by a dropping of the back grates.
>
> The other type of ash fusion is a short-range, spongy, porous formation of low viscosity, which, if it does not occur too low on the temperature scale, can be handled with ease in the boiler fires. From a comparison of boiler-house results with a large number of tests made, the fusiometer curve for this type of fusion appears to be distinguished by a most decided downward dip in the curve just prior to final softening.
>
> The term "fusing point" is misleading and indeterminate, while "fusing range" of a substance can be determined with exactitude. For purposes of

relative comparison, however, an arbitrary point in the fusing range may be selected and called the "fusing point" of the ash. For this point in the case of the fusiometer results, the author suggests the temperature at which the pellet has collapsed, owing to softening, to one-half of its original height.

Hubley shows a number of fusiometer curves illustrating the two types of fusion and discusses them together with the boiler-house reports in regard to the nature and extent of clinker formed under actual operating conditions. He states that usually he found these curves of much use in indicating the clinkering characteristics of the coals represented, and suggests the following specification:

> The pellet shall not have collapsed due to softening to more than one-half of the original height at a temperature, T, and in addition shall not show a collapse of more than one-eighth of the original height due to softening at a temperature of 300° F. under temperature T.
>
> The first part of this specification will limit the final fusion, while the second part is intended to control viscosity and avoid a sloping curve. The fixing of temperature T must vary with plant conditions. It is obvious that the buying field for steam coal will be restricted or broadened according as the temperature T is raised or lowered, so that the fixing of this temperature must be governed by the value which a plant management places on a loss of boiler capacity, and risk of steam failure against the saving made in purchasing the cheaper fuel.

The excellent agreement Hubley obtained on duplicate samples of the same ash does not prove that the method will give correspondingly close agreement in different laboratories, as variations in the slagging reactions of the ash will affect results obtained by this method as well as those obtained by the Seger-cone method. The principal factors would be the nature of the atmosphere—whether oxidizing or reducing—and the rate of heating. If a gas furnace is used, the nature of the atmosphere will vary with different adjustments of gas to air. The effect of different atmospheres on the softening of cones is described elsewhere. Undoubtedly their effect will be similar in the fusiometer furnace.

O. W. PALMENBERG, 1915.

Palmenberg[a] states that the Seger-cone method, modified as described below, is used in his laboratory, and has proved satisfactory in furnishing laboratory results that are comparable with the results obtained in practice.

> The coal ash which has been burned free of carbon and heated to insure oxidation of the iron is made into a round, thin cone 1½ to 2 inches high, weighing about 1 gram or less. This cone is placed upon a thin piece of Battersea fire clay in a vertical position and set in the middle of the muffle of a Meker

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[a] Palmenberg, O. W., Discussion of paper on the clinkering of coal: Jour. Am. Soc. Mech. Eng., vol. 37, 1915, pp. 211–212; Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 824–828.

> gas furnace. After heating the furnace, the blast is applied rapidly at first until the temperature is nearly that at which the cone begins to bend. Then the heat is regulated carefully and temperature readings with an optical pyrometer made every five minutes or oftener, depending upon the rate of heating; care must be taken to have the heating carried on slowly. As soon as the cone shows signs of bending, the temperature is noted and the operation continued until the cone has completely bent over and touched the plate, the temperature at which the cone touches the plate being called the fusing temperature.

Palmenberg prefers using long, thin cones, which in his experience are not subject to "separation of the more fusible constituents," an objection raised by Marks against the vertical-cone method. In regard to the effect of unburned carbon in the ash, Palmenberg states that this depends on the conditions of the tests and the nature of the ash; he found that as much as 10 per cent carbon had no effect, as seemingly it all burned off before the fusion took place.

E. B. RICKETTS, 1915.

Ricketts[a] describes the Seger-cone method as used in the laboratory of the New York Edison Co. as follows:

> About 2 pounds of coal is coked in a crucible in a gas furnace for about an hour, after which the lump of coke is broken up and a stream of compressed air is fed in near the bottom of the crucible for from two to three hours until the coke is all reduced to ash. When burning the coke down, care should be taken to keep the temperature below 1,500° F. (820° C.), so as to prevent premature fusion. The ash is then tested, a small amount at a time, with oxygen to make sure that all the carbon is consumed. It is of vital importance that no carbon be left in the ash as a small trace may cause results several hundred degrees too high. The carbon-free ash is moistened with a little water and molded in a paper cone 2½ inches long and ½-inch base. The cone is dried for five or six hours and then fused in a muffle furnace without removing the paper in which it was molded. The cone is mounted horizontally so that it overhangs the point of support about 1½ inches. The point of fusion is taken when the cone reaches a 45° position and is read on a Wanner optical pyrometer.
>
> The furnace consists of a gas melting furnace of 8 inches inside diameter. It is heated by horizontal blast burners which direct the flame around and over a covered Hessian crucible containing the ash cone. Observation of the interior of this crucible and of the cone is made through a suitably placed quartz tube which passes through the side of the melting furnace and the Hessian crucible.

Ricketts states further that with this method results of different tests of the same sample of coal corresponded within a few degrees and that the boiler tests with Babcock & Wilcox boilers fired with a Taylor stoker, showed a relation between efficiency of furnace and boiler and the fusion temperature of the ash. Thirteen coals were tested; the ash-fusion temperatures being between 2,225° F. (1,220°

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[a] Ricketts, E. B., Discussion of paper on the clinkering of coal: Jour. Am. Soc. Mech. Eng., vol. 37, 1915, pp. 213–214; Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 830–834.

C.) and 2,850° F. (1,570° C.). Those coals whose ash fused below 2,375° F. (1,300° C.) gave a distinctly lower efficiency than those whose ash fused above 2,450° F. (1,345° C.). The clinkering effect of coals whose ash-fusion temperatures were between the two temperatures mentioned was doubtful. The results of ash-fusion tests of coal from several hundred cargoes, from some 25 different sources, were said to confirm consistently the results of the earlier tests.

COMPOSITION OF ASH AND DISTRIBUTION OF ASH-FORMING CONSTITUENTS IN COAL.

Ash is the incombustible residue left from the complete combustion of the carbon in coal. It is derived from the inorganic matter in the coal, and is composed largely of compounds of silica, alumina, iron, lime, together with smaller quantities of magnesia, titanium, alkali, sulphur, and phosphorus compounds. The silica, alumina, and titanic oxide are derived from sand, clay, shale, and slate, the iron oxide mainly from iron pyrite, and the lime and magnesia from their corresponding sulphates and carbonates.

As shown in the ash analyses given in Table 2 (p. 27), the chemical composition varies so widely that no typical composition can be given. Probably the analyses of most samples of coal ash will come between the following limits:

Typical limits of ash analyses.

Constituent.	Per cent.
SiO_2	40 to 60
Al_2O_3	20 to 35
Fe_2O_3	5 to 25
CaO	1 to 15
MgO	0.5 to 4
Na_2O+K_2O	1 to 4

The ash-forming constituents consist of (1) "inherent" or "intrinsic" impurities that are present in an intimate mixture with the coal substance, and are derived either from the original vegetable material or from external sources by sedimentation, precipitation, or aeolic action during the process of accumulation of coal-forming vegetation; (2) impurities formed either during or after the laying down of the coal bed, which occur in the form of partings, veins, and nodules of clay, shale, slate, pyrite and calcite; and (3) impurities that become mechanically mixed with the coal in the process of mining, such as fragments of roof and floor.

INTRINSIC ASH.

The ash-forming constituents comprising the intimately mixed impurities may be obtained by placing coarsely crushed coal in a solution of calcium or zinc chloride having a specific gravity of 1.35. The coal will float, and the heavier extraneous impurities will sink. In some coals the percentage of intrinsic ash is so low that the ash-forming constituents must have come almost wholly from the coal-forming vegetation. A noteworthy example of coal exceptionally low in ash is that in the Sewell bed of West Virginia; several mine samples taken from this bed near Beckley, in Raleigh County, showed on analysis 1.5 to 2 per cent ash. Ordinarily the intrinsic ash is between 2 and 8 per cent, although in many coals it may reach a fairly high percentage, especially in bony coals, which yield 25 to 40 per cent ash.

In the minute joints or cleavage planes of certain coals are thin flakes of calcite, gypsum, silica, or clay,.and some coals, as certain Arkansas coals, have veinlets of pyrite. These impurities are not intimately mixed with the coal substance, but yet can not be separated from the coal by any commercial methods of preparing the coal for use. Microscopic crystals of pyrite are also found distributed to such an extent in some coals that washing produces no appreciable lowering of the sulphur content.

The "intrinsic" ash may be either more or less fusible than the total ash of the coal as mined, depending on the composition of the two kinds of ash. In general, the ash from a coal having a high intrinsic ash content, such as is found in bony coal, is less fusible than the ash from a coal having low intrinsic ash content, as the high ash content is due to an intimate admixture of siliceous or argillaceous silt, which increases the softening temperature. On the other hand, a coal of normally low intrinsic ash content, containing well distributed microscopic crystals of pyrite, or joints filled with flakes of calcite, gypsum, or pyrite, is likely to form troublesome clinkers when burned. Such a coal contains well mixed and finely divided essential constituents of an easily fusible slag.

EXTRANEOUS ASH.

The percentage of ash derived from extraneous impurities varies considerably, depending on the number and size of the partings in the seam, the possibility of separating these from the coal, and the care with which the coal is mined. Impurities of this character may be removed to some extent by suitable methods of washing, picking, or screening. The possibility of improving the quality of coal by washing may be determined by the laboratory float-and-sink test already described; also, by determining the fusibility of the ash of the raw coal, as compared with that of the float coal, the effect of better preparation on the fusibility of the ash may be determined.

Better preparation may either increase or decrease the ash fusibility, depending on the fusibility of the intrinsic ash and the nature of the extraneous impurities. Such extraneous impurities as slate, sandstone, or shale tend to raise the softening temperature; limestone and pyrite tend to lower the softening temperature, although limestone, if added in excess, will increase the softening temperature. Experience has shown that many of the best low-ash coals form very troublesome clinkers. A very fusible intrinsic ash would be expected to give excessive trouble in the fuel bed, as the various constituents are finely divided and intimately mixed in the coal, thus making a favorable condition for slag formation. It is doubtful whether large lumps of extraneous impurities have much influence on slagging in the furnace. Large pieces of slate are usually found unfused in the refuse.

NATURE OF FUSION OF ASH.

MELTING POINTS OF PRINCIPAL CONSTITUENTS OF ASH.

According to Day and Sosman,[a] "the melting point of a pure substance may be defined as the temperature at which the crystalline and liquid substance can remain side by side in equilibrium. Only the addition or withdrawal of a quantity of heat will cause the disappearance of one of the two forms in contact. Melting is therefore characterized by two concurrent changes, the appearance or disappearance of a particular crystal structure, and the appearance or disappearance of a quantity of heat. This definition applies to pure compounds. If the material is a mixture or a solid solution, it will have, not a melting point, but a melting interval, with (theoretically) definite temperature limits."

According to this definition, the principal constituents of coal ash, considered by themselves and in their crystalline form, have melting points well above the temperature at which fusion in the mixed ash is known to begin. Quartz (SiO_2) at temperatures above 1,600° C. changes rapidly into cristobalite, which melts at about 1,625° C.[b]

At this temperature the rate of melting is very slow; and the fused silica has such a high viscosity that it does not flow or change shape distinctly until a temperature of 1,750° C. is reached.[c]

Pure alumina (Al_2O_3) melts at 2,050° C.[d]

Kaolinite (Al_2O_3, $2SiO_2$, $2H_2O$), the crystalline form of the essential constituent of clay, is unstable at high temperatures and appears to decompose into aluminum silicate (Al_2O_3, SiO_2) and free silica.

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[a] Day, A. L., and Sosman, R. B., The melting points of minerals in the light of recent investigations on the gas thermometer: Am. Jour. Sci., vol. 31, 1911, p. 342.

[b] Fenner, C. N., The stability relations of the silica minerals: Am. Jour. Sci., vol. 36, 1913, pp. 331–384.

[c] Kanolt, C. W., Melting points of fire bricks: U. S. Bur. Standards Tech. Paper 10, 1912, p. 14.

[d] Kanolt, C. W., Melting points of some refractory oxides: U. S. Bur. Standards Bull., vol. 10, 1913, p. 15.

Sillimanite (Al_2O_3, SiO_2) melts at 1,816° C.[a] Kanolt[b] found the temperature of visible flowing of an English white kaolin to be 1,740° C. A sample of brownish-white German kaolin began flowing at 1,735° C. Lime (CaO) melts at 2,572° C.,[c] and magnesia (MgO) at 2,800° C.[d]

According to Sosman[e]—

> Ferric oxide (Fe_2O_3) begins to dissociate, as the temperature rises, into oxygen and a solid solution containing ferrous iron; this may be considered as a solid solution of Fe_3O_4 in Fe_2O_3. At a given temperature the initial dissociation is high, but it drops rapidly as the percentage of FeO in the solid increases, passing through a range in which the pressure falls rather slowly with change of composition and finally falling rapidly to the dissociation pressure of Fe_3O_4, which is very low (less than 0.04 mm. of mercury at 1,200° C.).
>
> Fe_3O_4, in turn, dissociates into oxygen and a mixture of oxides whose character has not yet been determined.
>
> The properties of FeO are still practically unknown. The most of the recorded methods for preparing "ferrous oxide" yield only a mixture of metallic iron (or iron carbide) with an oxide whose composition falls between FeO and Fe_3O_4.

The melting point of magnetic (Fe_3O_4) is 1,580° C.[f]

RELATION OF MELTING POINTS OF ASH CONSTITUENTS TO FUSION IN THE MIXTURE.

It is evident that coal ash, being a mixture of a number of substances both crystalline and amorphous, can have no single, definite melting point; neither is the gradual softening and fusion of the ash merely the successive melting of the various ash constituents, but it is a more complicated process in which reactions involving the formation of new and more fusible compounds take place. Furthermore, the melting temperature of these newly formed compounds is influenced by solution in one another to a greater or less degree, as in the familiar example of salt and ice. Pure water freezes at 0° C. If increasing quantities of salt are added to the water the freezing point is depressed more and more until a certain minimum freezing point is reached after which further addition of salt causes the freezing point to rise again. This minimum freezing point of a mixture is known as the eutectic point, and the composition of the mixture at this point is known as the eutectic composition.

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[a] Shepard, E. S., and Rankin, G. A., The binary systems of alumina, silica, lime, and magnesia: Am. Jour. Sci., vol. 28, 1909, pp. 293–333; Ztschr. anorg. Chem. Bd. 68, 1910, pp. 370–420.

[b] Kanolt, C. W., Melting points of fire bricks: U. S. Bur. Standards Tech. Paper 10, 1912, p. 14.

[c] Kanolt, C. W., Melting points of some refractory oxides: U. S. Bur. Standards Bull., vol. 10, 1913, p. 19.

[d] Kanolt, C. W., Place quoted.

[e] Sosman, R. B., The common refractory oxides: Jour. Ind. and Eng. Chem., vol. 8, 1916, p. 985.

[f] Sosman, R. B., The common refractory oxides; Jour. Ind. and Eng. Chem., vol. 8, 1916, p. 985.

FORMATION OF COMPOUNDS AND EUTECTICS AS ILLUSTRATED IN LIME-SILICA SYSTEM.

The compounds formed on heating and cooling various mixtures of two components, with their melting and transformation points, are enumerated in figure 1, which is a concentration-temperature diagram for the lime-silica system.[a]

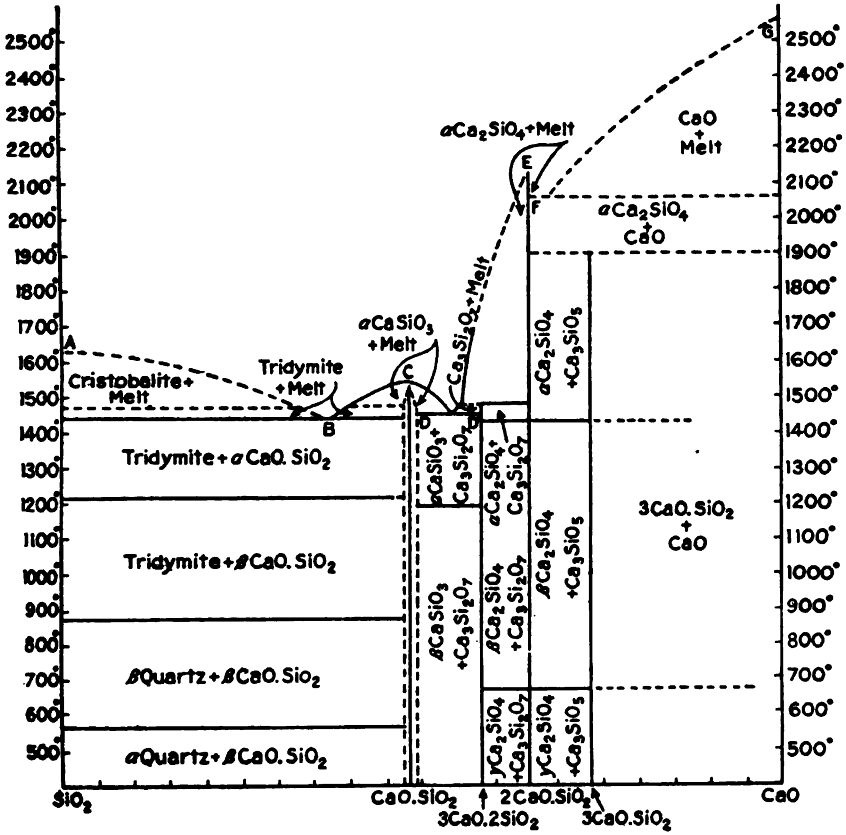

FIGURE 1.—Concentration-temperature diagram for the lime-silica system.

The melting point of pure silica in the form of cristobalite is given as 1,625° C. The melting point of pure calcium oxide (CaO) is given as 2,570° C. As increasing percentages of CaO are added to the SiO_2 the melting point is gradually depressed until at the point B a minimum or eutectic point is reached. The eutectic composition is 37 per cent CaO and 63 per cent SiO_2, and the melting point of the eutectic is 1,436° C. A further addition of lime increases the melting point until at C a maximum is reached at 1,540° C., which is the melting point of the compound pseudowollastonite ($\alpha CaSiO_3$), containing 48.2 per cent CaO and 51.8 per cent SiO_2. Any mixture of silica and lime between the points A and B and between the points B and C will show evidence of fusion when the temperature reaches 1,436° C., the melting point of the eutectic; as at this temperature that part of the mixture which is present in eutectic proportion (37CaO to $63SiO_2$) will melt, leaving an excess of one or the other component unmelted. The nearer the mixture as a whole approaches the eutectic composition, the greater the proportion that melts at the eutectic temperature.

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[a] Rankin, G. A., and Wright, F. E., The ternary system $CaO-Al_2O_3-SiO_2$: Am. Jour. Sci., vol. 39, 1915, p. 5.

The point E, 2,130° C., represents the melting point of the alpha form of calcium orthosiliciate (αCa_2SiO_4). Another eutectic occurs between the points E and C, and a third between E and G.

Figure 1 shows that in mixtures of CaO and SiO_2 there are possible two compounds and three binary eutectics having definite melting points. All other ratios of CaO and SiO_2 than these five have a melting interval rather than a melting point. The interval is between the melting point of the eutectic and the temperature represented by the upper curve, A B C, at which all the material has become liquid.

DETERMINATION OF THE MELTING INTERVAL.

It is possible to determine this melting interval by the "quenching method" of the geophysical laboratory.[a]

By this method a small charge of the mixture is held at a definite temperature until equilibrium is insured, when the melted mixture is dropped suddenly into a basin of mercury, in which it is instantly cooled before any crystallization can occur. The degree of melting for the given temperature is shown by the percentage of glass found when the melted mixture is examined under the microscope. By repeating the test several times at different temperatures the melting interval may be determined.

RELATION OF MELTING INTERVAL TO CLINKER FORMATION.

The quenching method, although undoubtedly the only scientific method of determining the degree of melting at any given temperature, does not give any information regarding the viscosity of the fusion, and therefore would not in itself serve as a practical technical test of probable clinkering characteristics. As regards clinker formation, information is desired as to the temperature at which the ash may form a slag of sufficient fluidity to flow or agglomerate in the fuel bed; this temperature is usually below the temperature of complete melting as shown by the quenching method, owing to the formation of low-melting eutectics, as already described.

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[a] Day, A. L., and Sosman, R. B., The melting points of minerals in the light of recent investigations on the gas thermometer: Am. Jour. Sci., vol. 31, 1911, p. 348.

The difference between the flowing or softening temperature and the temperature of complete melting varies greatly in different ashes, depending on the relative proportion of initially melting eutectic and the viscosity of this initial slag. If this slag is very fluid it may flow readily to form clinkers, and yet contain a considerable proportion of solid constituents in suspension. On the other hand, if the ash were composed almost wholly of alumina and silica, it could be completely melted and yet not flow readily, owing to its high viscosity. Hence, any laboratory method of determining the fusibility of ash, together with the degree of clinker formation, should indicate the viscosity as well as the melting temperatures. A theoretically correct method would be one in which the viscosity-temperature curve of the melt for blast-furnace slags is determined, as has been done by one of the authors.[a] This method is not available, as when it is used the iron oxides are reduced by the graphite parts of the apparatus. Probably the simplest method of roughly determining the temperature at which the melting ash forms a slag of standard viscosity is the usual softening-temperature test, which is employed by the metallurgist in determining the "formation temperature" of slags.[b]

DETERMINATION OF SOFTENING TEMPERATURE.

Determination of the softening temperature of an ash is usually made by molding the ash in the form of a small triangular pyramid similar to a Seger cone, and noting the temperature of deformation when the cone is heated at a definite rate. The temperature at which the cone has bent over until the apex touches the base, or at which it collapses to a lump, is taken as the softening point.

Day and Shepard[c] have shown that a cone made of silicate mixtures that are capable of forming eutectics begins to weaken as soon as the eutectic begins to melt; its further progress is then governed entirely by the relative quantity of eutectic present and its viscosity after melting. If the constituents of the ash are such as to form a relatively large proportion of thinly fluid eutectic, the deformation point of the cone will lie close to the melting point of the eutectic; on the other hand, if there is a large excess of some refractory component, as silica or aluminum silicate, or if the eutectic is viscous, the excess component will form a rigid skeleton, which is not pulled down by the eutectic, the deformation point approaching more nearly the melting point of the rigid component.

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[a] Feild, A. L., A method for measuring the viscosity of blast-furnace slags at high temperatures: Tech. Paper 157, Bureau of Mines, 1916, p. 21.

[b] Fulton, C. H., Principles of metallurgy, 1st ed., 1910, p. 274.

[c] Day, A. L., and Shepard, E. S., The lime-silica series of minerals: Am. Jour. Sci., vol. 22, 1906, p. 267. Also, for a full discussion of the theory of Seger cones, see Sosman, R. B., The physical chemistry of Seger cones: Trans. Am. Cer. Soc., vol. 15, 1913, pp. 482–498.

FACTORS INFLUENCING SOFTENING TEMPERATURE.

The whole phenomenon of softening and fusion of ash is a result of chemical reaction, melting, and solution, in which the time factor and the fineness of division of the constituents are of the greatest importance. Some reactions between ash constituents, and between certain oxides, with the atmosphere in which the ash is heated take place before any melting begins. Other reactions of various velocities make their appearance after incipient fusion. Also, certain constituents have a slow rate of melting, as, for example, silica and feldspar. These substances can be heated above their melting points and yet remain in a crystalline form for some time. It is obvious, then, that the softening temperature of an ash cone must be affected by (1) the size, shape, and inclination of the cone, (2) the fineness of the ash, (3) the rate of heating, and (4) the nature of the atmosphere in which the cone is heated.

INFLUENCE OF IRON OXIDES.

Owing to the invariable presence of iron oxide in coal ash, the nature of the surrounding gases—whether oxidizing or reducing—determines, in part, the rôle which the iron shall play in the slagging reactions. In the ordinary method of preparing a coal ash by burning off the carbon in air at a red heat the iron content is practically all converted to ferric oxide (Fe_2O_3). As this oxide is heated in air it begins to dissociate at about 1,100° C.,[a] forming a solid solution containing ferrous oxide (FeO). The degree of dissociation increases with the temperature, so that ultimately all the ferric oxide is converted to magnetite (Fe_3O_4) at approximately 1,350° C.[b] Therefore, in determining the softening temperature of an ash in air, with absolutely no reducing gases present, we are dealing with the formation of slags in which the iron component enters the reaction as ferric oxide or as a solid solution varying in composition from Fe_2O_3 to Fe_3O_4. Such slags are more refractory and more viscous than those formed by heating mixtures under conditions favoring the formation of ferrous iron. Peters[a] states that—

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[a] Sosman, R. B., and Hostetter, J. C., The oxides of iron. Solid solution in the system Fe_2O_3–Fe_3O_4: Jour. Am. Chem. Soc., vol. 38, 1916, pp. 807–833; Hostetter, J. C., and Sosman, R. B., The dissociation of ferric oxide in air: Jour. Am. Chem. Soc., vol. 38, 1916, pp. 1188–1198.

[b] Walden, P. T., On the dissociation pressure of ferric oxide: Jour. Am. Chem. Soc., vol. 30, 1908, p. 1350.

> Ferric oxide forms compounds with silica which require a high temperature for their fusion, and is consequently an unwelcome base for slags; although this substance is an almost invariable constituent of oxidized ores, it seldom makes any trouble in the blast furnace, for the reason that it is easily reduced by the fuel gases to ferrous oxide (FeO).
>
> In the more neutral atmospheres of the reverberating smelter, however, it is likely to cause delay by combining with silica, making it more difficult to melt the slag.

On the other hand, if the ash cone is heated in a reducing atmosphere of hydrogen and water vapor[b] or carbon monoxide and carbon dioxide, the ferric oxide is reduced largely to ferrous oxide, which is a much stronger fluxing agent than the higher oxides of iron. Steffe[c] found the following formation temperatures for various ferrous silicates and ferro-calcic silicates:

Formation temperatures of ferrous and ferro-calcic silicates, after Steffe.

Compound.	Formation temperature, ° C.
$4FeO,\ SiO_2$	1,158–1,174
$2FeO,\ SiO_2$	1,162–1,183
$4FeO,\ 3SiO_2$	1,162–1,181
$FeO,\ SiO_2$	1,158–1,171
$2FeO,\ 3SiO_2$	1,321–1,334
$2(FeO,\ CaO)SiO_2$	1,122–1,137

As shown by Greiner,[d] the viscosity of ferrous silicate slags is lower than that of slags in which the iron component enters the reaction in the ferric form. Hence, the conclusion seems logical that in general lower softening temperatures may be expected if, before fusion begins, the atmosphere surrounding the cone is able to reduce most of the iron to the ferrous state, but not to metallic iron, for, if the iron is reduced to the metallic state, one of the most active fluxing constituents is removed from the system. Most ashes have an excess of high-melting constituents like silica and alumina, so that any considerable reduction of iron oxide to metallic iron tends to increase the refractory effect of the silica and alumina. Hence, lower softening temperatures may be expected in tests made in gas furnaces, in which some reducing gases come in contact with the ash, whereas higher results should be found both in furnaces in which air only surrounds the cone, and in carbon or graphite electric furnaces in which strongly reducing atmospheres reduce the iron oxides to metallic iron.

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[a] Peters, E. D., Principles of copper smelting: New York, 1907, pp. 399–400.

[b] Hilpert, S., and Beyer, J., Ber. deut. chem. Gesel., Jahrg. 44, 1911, pp. 1608–1619.

[c] Steffe, Herman, Ueber die Bildungs temperaturen einiger Eisenoxydul-Kalkschlacken und einiger kalkfreien Eisenoxydul-Schlochen, deren Kenntniss für das Verschmelzen der Bleierze Bedeutung hat: Dissertation, Berlin, 1908.

[d] Greiner, E., Ueber die abhängigkeit der Viscosität in Silikat Schmelzen von ihrer Zusammensetzung: Inaugural dissertation, Jena, 1907, p. 55.

EXPERIMENTS WITH SEGER CONES.

DESCRIPTION OF FURNACES.

In studying the effect of various factors on the softening temperature of ash, the authors made tests with a selected series of coal ashes in different types of furnaces such as are in more or less common use for such tests. Each of these furnaces has its own particular atmosphere—oxidizing, reducing, or neutral—which is the principal cause

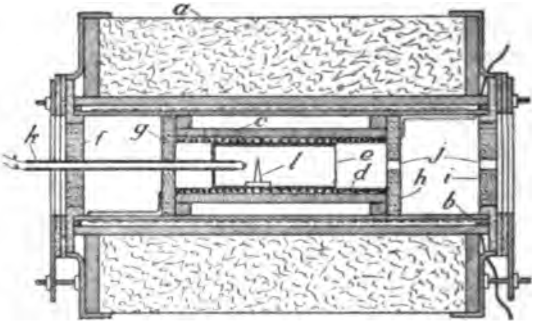

FIGURE 2.—Section through platinum-wire resistance furnace. *a*, Heraeus tube furnace, type B; *b*, porcelain tube, 2⅝ inches in inside diameter and 12 inches long, wound with a platinum-foil heating element; *c*, alundum tube, 1½ inches in inside diameter and 6 inches long, with $\frac{3}{16}$-inch wall; *d*, 17 feet of platinum wire 3/100 of an inch in diameter, wound on the inside of tube *c*, six turns to the inch in the middle and closer at the ends to compensate partly for the radiation at the ends; *e*, alundum extraction thimble No. 5811, 30 mm. in diameter and 80 mm. long; *f*, *g*, and *h*, *i*, alundum disks; *j*, observation hole; *k*, thermo element made of platinum and platinum-rhodium; *l*, ash cone.

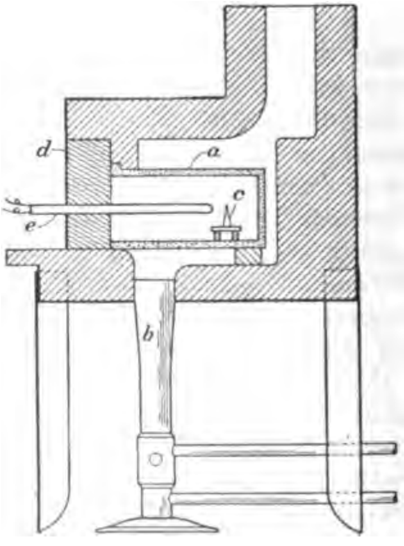

FIGURE 3.—Meker muffle furnace No. 29. *a*, Magnesite muffle; *b*, Meker blast burner; *c*, ash cone; *d*, removable fire-clay door with horizontal slot for thermocouple tube *e*.

of the large variations that were found on comparing the softening temperatures of the same sample in the different furnaces.

In the comparative tests six furnaces were used, as follows:

1. Platinum-wire resistance furnace (fig. 2), having an oxidizing atmosphere of air, with absolutely no reducing gases present.

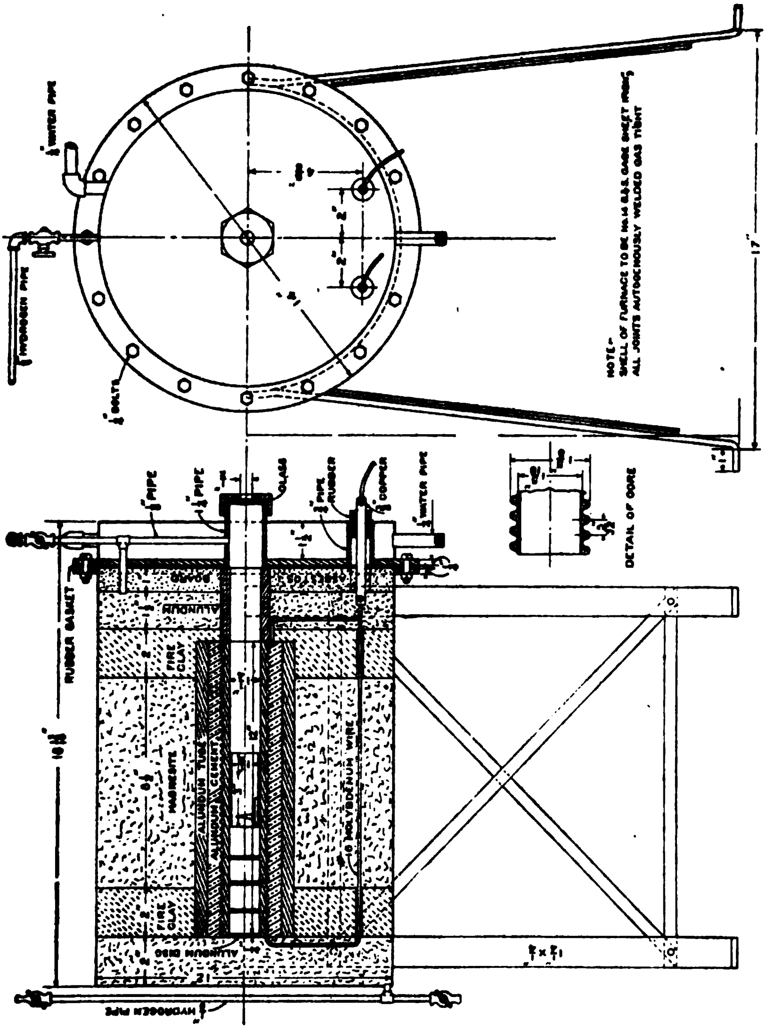

FIGURE 4.—Molybdenum furnace.

2. Meker muffle furnace No. 29 (fig. 3), having an atmosphere of air and combustion gases.

3. Muffle furnace No. 2, having an atmosphere of air and combustion gases.

4. Molybdenum-wire resistance furnace (fig. 4), having a reducing atmosphere of hydrogen,

5. Granular-carbon resistance furnace (fig. 5), having a reducing atmosphere of carbon monoxide.

6. Northrup graphite resistance furnace (fig. 6), having a reducing atmosphere of carbon monoxide.

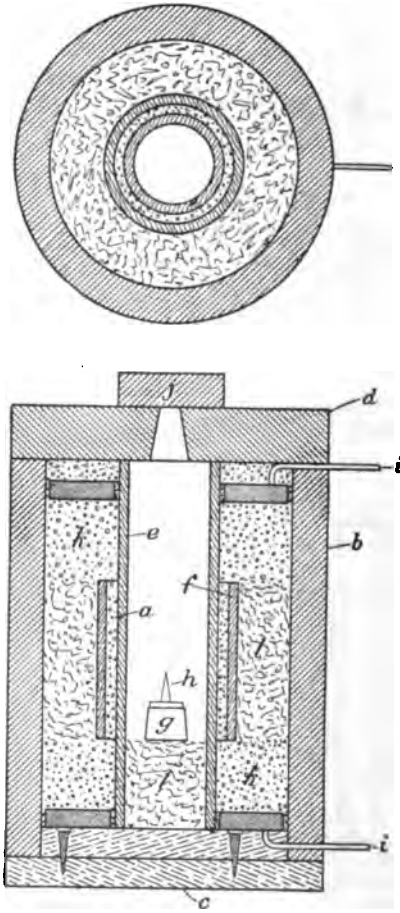

FIGURE 5.—Granular-carbon resistance furnace. *a*, Granular-carbon resister, consisting of electrode carbon crushed to pass a 6-mesh and remain on a 12-mesh screen; *b*, fire-clay cylinder, 8 inches in external diameter, 10 inches high, 1-inch wall; *c*, transit base, ¾ inch thick; *d*, fire-clay cover plate, 2 inches thick; *e*, alundum tube, 2-inch bore, 9 inches long, ¼-inch wall; *f*, alundum tube, 3-inch bore, 4 inches long, 5/16-inch wall; *g*, inverted alundum crucible; *h*, ash cone; *i*, *i*, wrought-iron electrodes; *j*, fire-clay plate covering peephole; *k*, granular carbon; *l*, magnesia insulating material.

Before studying the results obtained in each furnace, as presented in the following pages, the reader is invited to notice the general method of operating the furnaces as outlined in the titles of the figures showing them.

DESCRIPTION AND ANALYSES OF COAL SAMPLES USED IN TESTS.

The description and origin of the coal samples are given in Table 1 following, and the analyses of the ash and partial analyses of the coal in Table 2 following. As may be seen from an inspection of these tables, the series of fuels tested include anthracite, bituminous, semibituminous, and subbituminous coal, lignite, and peat; the composition of the ash covers a fairly wide range, the silica content varying from 12.3 to 76 per cent; alumina from 8.6 to 34.7; ferric oxide from 3.8 to 69.7; lime from 0.6 to 18.6; and magnesia from 0.2 to 10 per cent.

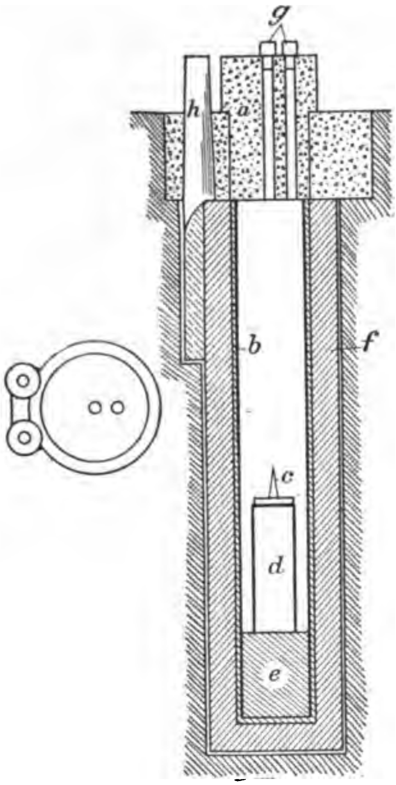

FIGURE 6.—Arrangement for softening temperature test in Northrup furnace. *a*, Refractory plug; *b*, graphite tube for protecting heater from oxidation, 1⅝ inches in internal diameter, 12⅝ inches long; *c*, ash cone; *d*, inverted alundum extraction capsule, 3 inches high; *e*, solid graphite cylinder, 2 inches high; *f*, graphite heater unit; *g*, observation holes, ¼ inch in diameter; *h*, terminal of graphite heater unit.

TABLE 1.—*Character and source of coal samples.*

Sample No.	Lab. No.	Kind of coal.	Coal bed.	Location of mine.		
				Nearest town.	County.	State.
1	15840	Bituminous	Coal Creek	Fraterville	Anderson	Tenn.
2	15841	do.	American	Parrish	Walker	Ala.
3	15842	do.	Sterling	Manring	Claiborne	Tenn.
4	15843	do.	Jellico	Jellico	Campbell	Do.
5	15844	do.	Mingo	Fork Ridge	Claiborne	Do.
6	15845	do.	Coal Creek	Oliver	Morgan	Do.
7	15846	do.	Mary Lee	Red Star	Walker	Ala.
8	15847	do.	Nickel Plate	Jefferson	Jefferson	Do.
9	15848	do.	Thompson	Marvel	Bibb	Do.
10	16018	do.	Red Stone	Lemely Junction	Barbour	W. Va.
11	16019	do.	Pittsburgh	Morgantown	Monongalia	Do.
12	16243	do.	No. 5	Booneville	Warrick	Ind.
13	16583	Semibituminous	Pocahontas No. 3	Simmons	Mercer	W. Va.
14	16584	do.	do.	Bramwell	do.	Do.
15	16585	do.	do.	Elk Ridge	McDowell	Do.
16	16586	do.	do.	Big Four	do.	Do.
17	16587	do.	do.	Jenkin Jones	do.	Do.
18	16589	do.	do.	Boissevain	Tazewell	Va.
19	17081	Anthracite (buckwheat).				Pa.
20	17189	do.		Shenandoah	Schuylkill	Do.
21	17534	do.			Luzerne	Do.
22	17590	do.		Shamokin	Northumberland	Do.
23	18248	do.				Do.
24	14762	do.				Do.
25	17563	Semibituminous	Lower Kittanning	Lonaconing	Alleghany	Md.
26	[a] 7244	do.[b]	Pocahontas No. 3	Pocahontas	Tazewell	Va.
27	[a] 7305	do.[b]	do.	do.	do.	Do.
28	7308	do.[b]	do.	do.	do.	Do.
29	[c] 7309	do.[b]	do.	do.	do.	Do.
30	18193	do.	do.	do.	do.	Do.
31	18198	do.	do.	do.	do.	Do.
32	18203	do.	do.	do.	do.	Do.
33	18349	do.	do.	do.	do.	Do.
34	18350	do.	do.	do.	do.	Do.
35	18208	do.	Lower Kittanning	Ebensburg	Cambria	Pa.
36	18296	do.		Bering River Field		Alaska.
37	18297	do.		do.		Do.
38	18298	do.		do.		Do.
39	18302	do.		do.		Do.
40	18308	do.		do.		Do.
41	18310	do.		do.		Do.
42	18312	do.		do.		Do.
43	18300	do.		do.		Do.
44	18303	do.		do.		Do.
45	18306	do.		do.		Do.
46	18348	do.		do.		Do.
47	18347	do.		do.		Do.
48	18350	do.		do.		Do.
49	12690	Bituminous	Pittsburgh	Greensburg	Westmoreland	Pa.
50	[c]12691	do.	do.	do.	do.	Do.
51	[a] 7228	do.	do.	Marianna	Washington	Do.
52	7230	do.	do.	do.	do.	Do.
53	[a] 7141	do.	do.	do.	do.	Do.
54	[c] 7458	do.	do.	Carnegie	Alleghany	Do.
55	13316	Subbituminous		Batan	Philippine Islands	
56	[c] 7548	Lignite		Rockdale	Milan	Tex.
57	[a] 7381	do.		do.	do.	Do.
58	[a] 7490	do.		Lytle	Medina	Do.
59	[a] 7494	do.		do.	do.	Do.
60	[c] 7522	do.		Calvert	Robertson	Do.
61	7534	do.		do.	do.	Do.
62	[a] 7536	do.		do.	do.	Do.
63	[a] 7159	Peat briquets		Elizabeth City	Pasquotauk	N. C.
64	17559	Bituminous	Widow Kennedy		Dickinson	Va.
65	17560	do.	do.		do.	Do.
66	18121	do.	do.	Dante	Russell	Do.
67	19514	Semibituminous	Lower Kittanning or B.	Colver	Cambria	Pa.
68	19817			Stoke-on-Trent	Birchenwood Collieries.	England.
69	19818	Semibituminous	Pocahontas			
70	19841					China.
71	19842					Do.
72	20021	Bituminous			Matanuska field	Alaska.
73	20022	do.			do.	Do.
74	20023	do.			do.	Do.
75	20153	do.	Anderson	West Fork Station	Coos	Oreg.

[a] Clinker from gas producer. [b] Bone coal. [c] Clinker from boiler furnace.

TABLE 1.—*Character and source of coal samples*—Continued.

Sample No.	Lab. No.	Kind of coal.	Coal bed.	Location of mine.		
				Nearest town.	County.	State.
76	20191	Semibituminous	Pocahontas			W. Va.
77	20204	do.	do.			Do.
78	20217	do.	do.			Do.
79	20851	do.	Sewell	Dunlap and Prudence.	Fayette	Do.
80	20852	do.	do.	do.	do.	Do.
81	20853	do.	do.	do.	do.	Do.
82	20913	Bituminous	Jellico		Campbell	Tenn.
83	20914	do.	Coal Creek		Anderson	Do.
84	21328	Semibituminous		Huntington	Sebastian	Ark.
85	21895	Bituminous	Upper and Lower Banner.	Dante	Russell	Va.
86	[a]22665	Semibituminous	Lower Kittanning or B.	Nanty Glo	Cambria	Pa.
87	22855	Bituminous	Lower Weir-Pittsburg.	Frontenac	Crawford	Kans.
88	22988	do.	Saginaw	Akron	Tuscola	Mich.
89	22991	do.	No. 6	Duquoin	Perry	Ill.
90	23014	do.	No. 2		Saginaw	Mich.
91	2 S. A.	Semianthracite	Spadra		Johnson	Ark.
92	3 Poca.	Semibituminous	Pocahontas No. 3		McDowell	W. Va.
93	4 S. B.	do.	Sewell		Fayette	Do.
94	6 Wash.	Bituminous			Pierce	Wash.
95	9 Ill.	do.	Herrin or No. 6		Macaupin	Ill.
96	10 Wyo.	Subbituminous	Carney		Sheridan	Wyo.
97	11 Lig.	Lignite			Williams	N. Dak.
98	13629	Bituminous	Freeport	New Kensington	Westmoreland	Pa.
99	23331	do.	Pittsburgh		Washington	Do.
100	23507	do.	Freeport	New Kensington	Westmoreland	Do.
101	19159	Semibituminous				W. Va.
102	10155	do.				Do.
103	19161	do.				Do.
104	22934	Bituminous	Lower Kittanning	Vintondale	Cambria	Pa.
105	23136	do.		Castle Gate	Carbon	Utah.
106	23137	do.		Sunnyside	do.	Do.
107	23138	do.		Clear Creek	do.	Do.
108	23581	do.	Pittsburgh	Blaine Diamond	Allegheny Washington	Pa.
109	25366	do.		Hale and Ladysmith.	Clearfield	Do.
110	25370	do.		Norfolk No. 3	McDowell	W. Va.

[a] Clinker from gas producer.

TABLE 2.—*Analyses of ash and coal.*

Lab. No.	Sample No.	Analyses of ash.									Analyses of dry coal.		
		SiO_2.	Al_2O_3.[a]	Fe_2O_3.	TiO_2.	CaO.	MgO.	Na_2O.	K_2O.	SO_3.	Sulphur.	Ash.	Fe_2O_3.
		P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P ct*	*P. ct.*	*P. ct.*	
15840	1	35.7	23.5	32.9	1.2	3.2	1.1	0.3	1.1	0.5	3.1	10.8	3.6
15841	2	47.3	34.6	9.8	1.8	1.3	.4	2.1	2.5	.1	1.7	17.4	1.7
15842	3	55.8	33.5	5.0	.9	1.5	.7	.5	2.2	.1	.8	15.6	.8
15843	4	43.3	31.4	13.6	1.2	4.2	1.4	.6	2.9	1.4	1.5	8.6	1.2
15844	5	42.2	30.6	19.0	1.2	1.3	1.0	1.3	2.9	.2	1.4	7.5	1.4
15845	6	12.3	12.2	69.7	.4	3.9	.7	.3	.6	.2	5.8	8.2	5.7
15846	7	54.1	34.7	4.5	1.5	1.2	.9	.6	2.5	.2	.7	17.7	.8
15847	8	46.8	28.7	18.0	1.4	.6	.9	1.6	2.4	.3	2.3	12.7	2.3
15848	9	54.8	27.0	7.0	1.3	4.3	1.7	.3	3.1	1.4	.6	17.5	1.2
16018	10	38.4	24.2	22.4	1.1	7.7	.9	.3	1.9	3.8	2.1	7.9	1.8
16019	11	50.4	24.0	20.4	1.4	1.7	.2	1.0	1.0	.3	2.7	10.9	2.2
16243	12	37.1	17.6	35.9	.7	3.2	.9	.4	1.8	2.3	5.8	11.5	4.1
16583	13	54.8	27.0	7.8	1.6	1.6	1.5	2.2	1.9	.5	.7	8.0	.6
16584	14	54.8	29.2	6.9	1.8	1.4	.6	1.9	2.1	1.0	.7	7.5	.5
16585	15	54.1	24.8	9.4	2.3	4.0	1.4	1.0	.8	2.8	.6	7.1	.7
16586	16	37.2	25.5	11.8	1.5	12.6	1.9	1.4	.4	5.6	.6	5.9	.7
16587	17	51.1	25.2	10.1	1.8	5.1	1.6	.8	.9	3.1	.6	6.8	.7
16589	18	51.8	25.0	9.0	2.0	4.0	1.5	1.3	.8	4.3	.7	5.6	.5
17081	19	56.1	31.4	5.0	1.9	1.0	1.0	.7	3.1	.5	.7	21.5	2.1
7244	26	68.9	21.4	4.5	2.6	2.0	.7	.6	.5		.6	17.2	.8

TABLE 2.—*Analyses of ash and coal*—Continued.

Lab. No.	Sample No.	Analyses of ash.									Analyses of dry coal.		
		SiO_2.	Al_2O_3.[a]	Fe_2O_3.	TiO_2.	CaO.	MgO.	Na_2O.	K_2O.	SO_3.	Sulphur.	Ash.	Fe_2O_3.
		P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	
7305	27	69.4	21.5	4.8	2.0	1.6	.8	.5	1.4	......	.7	14.8	.7
7308	28	64.7	23.2	4.7	2.6	1.2	.8	.3	.5	1.5	.7	14.8	.7
7309	29	69.6	19.7	5.2	2.2	2.0	.4	.4	.5	......	.7	14.1	.7
12690	49	54.7	32.9	8.9	......	1.6	.9	.6	.2	......	.9	11.1	1.0
7228	51	54.5	27.0	12.1	1.8	1.5	.4	.5	1.9	......	1.7	11.4	1.4
7230	52	53.2	26.0	15.8	1.4	1.0	.7	.3	1.6	.4	1.7	11.4	1.5
7141	53	54.6	25.7	12.4	1.4	4.3	.6	.6	1.3	......	1.4	12.5	1.6
7458	54	56.8	28.2	11.3	1.2	1.0	.8	.6	1.6	......	1.8	12.3	1.4
13316	55	15.2	8.6	13.3	......	18.1	10.0	5.3	1.8	26.9	......	8.2	1.1
7548	56	47.9	23.7	5.5	2.0	18.6	3.3	.4	.2	......	1.5	16.6	.9
7381	57	46.2	24.3	6.2	1.8	18.5	3.0	.6	.5	......	1.5	15.5	1.0
7490	58	43.2	16.9	7.1	1.6	12.2	1.7	.2	.3	16.0	2.2	19.1	1.4
7494	59	52.5	21.5	8.8	1.6	14.4	2.4	.5	.2	......	2.2	19.1	1.7
7522	60	47.5	29.1	5.3	2.0	14.0	2.2	.4	.3	......	1.4	18.6	1.0
7534	61	39.3	24.8	3.8	1.9	14.9	1.7	.2	.4	12.5	1.4	16.6	.6
7536	62	46.5	28.4	5.2	2.0	16.1	2.3	.7	.2	......	1.4	16.6	.9
7159	63	76.0	11.7	4.5	1.0	2.8	3.1	1.0	.9	......	1.4	32.3	1.5
19842	71	54.1	25.8	9.0	1.0	1.9	1.2	1.6	3.8	1.4	.6	12.4	1.1
20021	72	49.9	29.1	10.0	1.1	3.7	1.4	.9	.7	2.5	.5	22.7	2.3
20204	[b] 77	59.7	26.4	7.0	......	2.6	1.3	......	(c)	......	.6	5.7	.4
20217	[b] 78	56.3	25.9	9.5	......	3.1	1.5	......	(c)	......	.6	6.2	.6
20851	79	54.3	24.6	12.4	1.1	1.4	.4	2.8	1.9	1.0	.9	7.4	.9
20913	82	48.0	28.0	11.7	1.9	3.6	1.1	1.4	1.9	2.9	1.5	8.6	1.0
20914	83	37.9	27.3	18.4	3.0	5.6	1.4	1.9	1.1	3.8	3.1	10.8	2.0
21328	[b] 84	37.4	23.3	28.8	......	3.8	3.0	......	(b)	1.1	2.3	9.5	2.7
[a]21895	[b] 85	38.8	25.5	7.9	1.3	12.1	2.4	2.5	1.7	8.3	.6	8.6	.7
22665	[b] 86	31.0	22.7	41.9	......	2.1	.5	......	(b)	.6	3.1	8.2	3.4
22855	87	34.2	14.6	31.5	.7	7.6	.9	1.1	1.5	6.5	5.0	19.0	6.0
22991	89	52.0	25.9	13.4	1.1	2.5	.6	.8	1.6	1.8	1.8	11.2	.5
13629	98	38.2	23.4	23.5	......	5.6	1.1	5.3	1.4	2.9	2.9	12.5	2.9
23507	100	42.7	25.6	18.9	1.2	5.2	.4	.7	1.3	4.2	2.4	11.9	2.2
17559	64	50.5	29.2	9.3	2.1	1.8	2.1	.7	3.5	.5	.7	11.2	1.0
17560	65	47.8	27.0	13.6	2.1	2.4	1.7	.4	3.1	1.3	.7	8.4	1.2
18121	66	31.9	23.2	19.6	1.8	9.8	1.1	.7	2.6	8.8	1.8	7.6	1.5
19514	67	43.8	29.0	18.4	2.6	2.6	.2	.5	1.1	1.9	1.4	7.0	1.3
19817	68	48.9	24.6	10.5	2.3	4.2	2.1	1.1	2.7	3.0	.9	14.2	1.5
19818	69	49.8	25.5	9.6	1.7	3.8	2.5	1.3	3.3	2.6	.9	16.7	1.6
19841	70	......	......	......	......	......	......	......	......	......	.5	5.4	......
20022	73	51.3	25.9	8.3	2.3	5.1	3.0	1.1	1.1	2.4	.4	22.2	1.8
20023	74	49.1	28.7	7.9	1.4	5.9	1.8	.6	.8	3.3	.4	13.3	1.1
20153	75	55.9	24.9	6.5	3.5	3.5	.5	1.8	1.0	2.1	.9	21.7	1.4
20191	76	54.2	22.5	8.4	3.9	4.0	1.0	1.7	1.1	3.5	.7	5.9	.5
20852	80	46.2	27.9	18.4	2.0	2.0	.6	1.2	1.3	1.0	1.2	6.3	1.2
20853	81	47.4	27.3	15.2	2.1	1.7	1.3	1.2	2.7	.8	1.0	7.2	1.1
20913	82	......	......	......	......	......	......	......	......	......	1.0	5.5	......
20914	83	......	......	......	......	......	......	......	......	......	1.6	7.0	......
21895	85	......	......	......	......	......	......	......	......	......	.6	8.6	......
22988	88	17.8	15.6	59.8	.6	1.9	.5	1.1	.7	1.9	4.7	7.8	4.7
23014	90	40.2	26.6	14.9	1.2	7.2	.7	.8	2.4	5.8	2.4	15.2	2.3
2	91	11.0	27.5	34.6	1.4	10.7	3.1	.6	.3	9.7	1.7	9.6	3.3
3	92	59.3	21.9	5.9	6.2	1.9	.6	1.8	1.4	1.1	.6	7.4	.4
4	93	40.8	25.8	24.5	3.1	2.6	.7	.8	.7	.4	.8	3.5	.9
6	94	20.8	31.6	23.9	2.8	9.0	1.1	1.4	.8	8.6	.8	8.6	2.1
9	95	58.6	21.7	7.5	3.3	2.7	.7	2.0	2.2	1.6	4.5	11.5	.9
10	96	23.0	24.3	4.4	.7	20.6	3.0	3.8	.5	19.2	.5	5.2	.2
11	97	7.7	12.1	6.9	.3	22.9	4.2	18.2	.6	26.8	.8	8.8	.6
23331	99	46.3	22.7	10.8	1.3	8.7	.4	.7	1.1	7.5	1.5	12.3	1.3
22934	104	30.3	24.6	37.9	1.3	3.0	.2	.5	.6	2.0	2.7	7.4	2.7
23136	105	48.4	16.5	5.6	......	14.5	2.6	2.1	.7	8.8	.4	5.4	.3
23137	106	55.1	34.0	4.4	......	4.1	.1	.7	.2	1.3	.7	5.3	.2
23138	107	60.0	17.6	6.2	......	7.0	.7	1.5	.4	6.4	.6	4.1	.3
23581	108	48.0	24.7	11.6	1.8	5.8	.5	1.1	1.0	5.0	1.5	10.2	......
25366	109	42.5	31.8	19.6	1.8	2.1	.2	.8	.6	.9	1.5	8.8	......
25370	110	63.8	22.6	5.7	3.7	1.3	.7	1.2	.7	.7	.5	10.0	......

[a] P_2O_5 included with Al_2O_3. TiO_2 not determined separately but included with Al_2O_3 in all cases where no result for TiO_2 is given.

[b] Analysis is not that of coal ashed in laboratory but of ashes taken from steaming tests in which similar coal was used.

[c] Alkalies not determined.

PREPARATION OF ASH.

The coal samples were ground to 60 mesh with a crusher, rolls, and ball mill as described in Technical Paper 8.[a] The 60-mesh material was spread out on shallow, 6-inch, fire-clay roasting dishes, and reduced completely to ash with occasional stirring in a muffle furnace at a temperature not exceeding 750° C. All of the ash was finally put through a 100-mesh screen and thoroughly mixed.

PREPARATION OF CONES.

Sufficient ash to make the desired number of cones was transferred to an agate mortar, moistened with dextrin solution, and worked into a plastic mass with a spatula or pestle. After the brass mold (fig. 7) had been lubricated with vaseline the plastic material was firmly pressed into it with a knife spatula, and the surface struck off smooth to make a neat, solid triangular pyramid. The cone was then pushed out of the mold by applying a small knife blade at the base. With a little practice and proper lubrication of the mold the cone can be ejected immediately after molding without waiting for the cone to dry; a slight rounding of the bottom groove of the mold greatly facilitates removal of the cone.

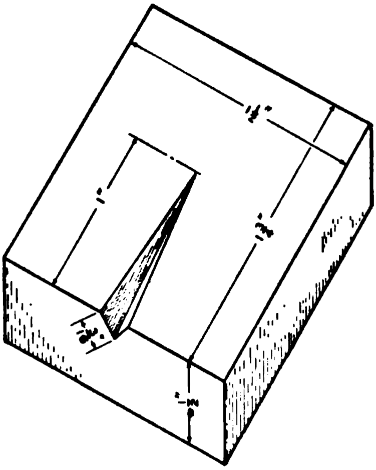

Figure 7.—Brass cone mold.

After the cones had dried, they were mounted in a refractory base composed of a mixture of equal parts of kaolin and alumina (Al_2O_3). This mixture was moistened with water to make it workable, and enough was taken for the base to be made and was spread on a sheet-iron plate. A small hole was made, into which the cone was set, and the base material was worked around the bottom of the cone so that it would be firmly set at the desired inclination in the base. The iron plate was then put on a hot plate, and the mounted cones were dried slowly until all water had been driven off. After the dextrin had been burned by igniting the mounted cones in a muffle, they were ready for use. In the earlier experiments the cones were made directly from the 100-mesh ash; later it was found that ash ground to an impalpable powder (or at least to pass a 200-mesh screen) made a more substantial and more easily molded cone than 100-mesh material. Four different sizes of cones were used, as follows:

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[a] Stanton, F. M., and Fieldner, A. C., Methods of analyzing coal and coke: Tech. Paper 8, Bureau of Mines, 1913, pp. 7–9.

Sizes of cones.

No. 1—Side of base, ¼ inch; height, 1 inch.
No. 2—Side of base, ¼ inch; height, 1½ inches.
No. 3—Side of base, 3/16 inch; height, 1 inch.
No. 4—Side of base, ½ inch; height, 2½ inches.

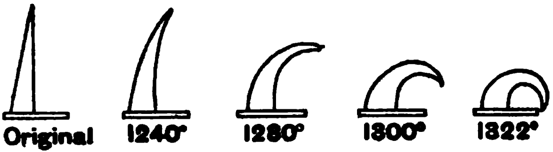
GENERAL METHOD OF HEATING.

The general procedure in making a softening-temperature determination was the same with all the furnaces used, though necessarily the details varied in the different furnaces. The test cone was put into the cold or nearly cold furnace and the furnace was heated at the rate of 10° to 15° C. per minute up to a temperature not less than 200° C. below the probable softening point, when the rate (usually 2° or 5° C. per minute) which had been adopted for that particular determination was begun. Temperature readings were taken every five minutes, and more frequently when the softening temperature was approached. Observations of the appearance of the cone were made at least as often as temperature readings were taken, special care being taken to note any deformation or warping due to shrinking before actual softening began. The point of initial softening or deformation was taken as the temperature at which the first noticeable bending, rounding at the top, or swelling of the cone took place. Warping of the cone due to shrinkage was not considered as the beginning of fusion.

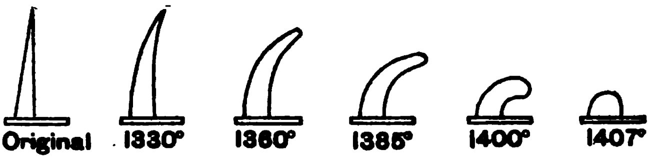
The softening point, deformation point, or "fusion point," so called, was taken as the temperature at which the apex of the cone had bent over to touch the base, or, failing to bend, had fused down to a lump or ball. Sketches (fig. 8) were made of the appearance of the cone at the initial and final deformation points and at several intermediate points of deformation, the corresponding temperatures being noted. Immediately after the temperature corresponding to complete deformation had been read, the current or other source of heat was reduced so that the appearance of the cone could be verified by examination after removal from the furnace.

TEMPERATURE MEASUREMENTS.

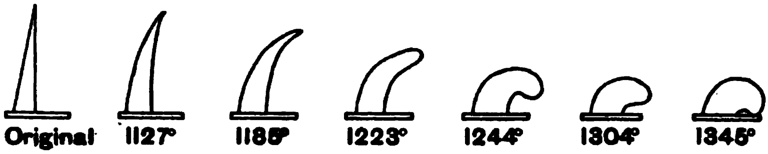
The temperature measurements were made in two ways, namely, by means of a Heraeus platinum and platinum-rhodium thermocouple with Siemens and Halske high-resistance millivoltmeter, in the platinum-resistance, Meker, and muffle No. 2 furnaces, and the down-draft ceramic kiln, and by means of a Wanner pyrometer in the molybdenum, carbon-resistance, and Northrup furnaces. The thermocouple and the millivoltmeter were standardized from time to time in the physical laboratory of the bureau under the direction of J. K. Clement, physicist, by comparison with a standard thermocouple. The cold junction was kept at the temperature of melting ice during standardization and during use in measuring temperatures. Further checks on the furnace and the pyrometer were obtained by placing crystals of pure diopside[a] in the positions occupied by the ash cones and noting the temperature of melting; usually the crystal melted between 1,381° and 1,395° C. (corrected temperature readings). The melting point of diopside is 1,391° C.[b]



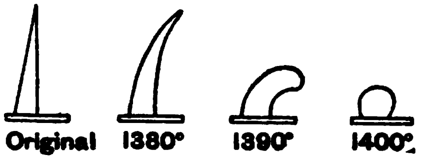
SAMPLE 16, SHOWING GOOD BEND.



SAMPLE 15, WHICH BENT AND MELTED TO A BALL.



SAMPLE 5, WHICH BENT AND PUFFED.



SAMPLE 11, WHICH BENT AND MELTED TO A BALL.

FIGURE 8.—Appearance of ash cones at various stages of softening.

The Wanner pyrometer which was originally standardized by the Reichsanstalt, was rechecked by the Bureau of Standards after most of the work described in this paper had been completed, and was found to conform to the original standardization. It was checked daily against the amyl-acetate flame. As this pyrometer was used only with those furnaces that had strongly reducing atmospheres it was possible to check the accuracy of the pyrometer readings by placing thin strips of Kahlbaum's pure copper or Kahlbaum's pure nickel in place of the cones and noting the apparent melting temperature. This was done at least once a week, and at the beginning and the end of each series of tests.

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[a] Obtained through the kindness of Dr. A. L. Day, director of the Geophysical Laboratory.

[b] Day, A. L., and Sosman, R. B., The melting points of minerals in the light of recent investigations on the gas thermometer: Am. Jour. Sci., vol. 31, 1911, p. 346.

The results obtained in these checks were usually within 10° C. of the true melting point at the copper point and within 20° C. at the nickel point.

SUMMARY OF RESULTS OBTAINED IN THE DIFFERENT FURNACES.[a]

VARIATIONS DUE TO SIZE AND SHAPE OF CONE.

Three different sizes of cones were compared in the Meker, the carbon-resistance and the molybdenum furnace, namely, cone 1, ¼ by 1 inch; cone 2, ¼-inch by 1½ inches; and cone 3, $\frac{3}{16}$-inch by 1 inch. The results, which are given in Table 3 following, indicate that cones with a wider base as compared to height, as cone 1, give higher softening temperatures than the more slender cones like cones 2 and 3. In general, the slender type of cone represented by size 3, $\frac{3}{16}$ by 1 inch, gave shorter softening intervals and less trouble from intumescence due to evolution of gases from the melting ash.

TABLE 3.—*Results of tests showing influence of size of cone on softening temperature.*

[Rate of heating, 2° C. per minute; 100-mesh ash; cones inclined 35° from vertical.]

Sample No.	Meker furnace.			Carbon furnace.			Molybdenum furnace.		
	Cone 1.	Cone 2.	Difference.	Cone 1.	Cone 2.	Difference.	Cone 1.	Cone 3.	Difference.
	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*
13	1,406	1,273	+133	1,390	1,322	+68	1,217	1,217	0
15	1,307	1,186	+121	1,311	1,250	+61	1,167	1,161	+ 6
16	1,225	1,212	+ 13	1,394	1,341	+53	1,241	1,241	0
17	1,242	1,182	+ 60	1,251	1,220	+31	1,192	1,179	+13
18	1,289	1,204	+ 85	1,316	1,250	+66	1,562	1,558	+ 4
4	1,265	1,305	− 40	1,357	1,306	+51	1,227	1,234	− 7
5	1,388	1,376	+ 12	1,401	1,356	+45	1,234	1,231	+ 3
8	1,357	1,343	+ 14	1,192	1,166	+26	1,318	1,263	+55
9	1,335	1,318	+ 17	1,327	1,249	+78	1,213	1,199	+14
1				1,100	1,090	+10	1,123	1,123	0
2				1,631	1,593	+38	1,430	1,417	+13
3				1,558	1,562	− 4	1,457	1,396	+61
6				1,207	1,179	+28	1,350	1,336	+14
7				1,663	1,638	+25	1,590	1,580	+10
10				1,187	1,167	+20	1,185	1,192	− 7
11				1,190	1,139	+51	1,220	1,220	0
12				1,088	1,088	0	1,068	1,060	+ 8
14				1,520	1,424	+96	1,326	1,316	+10
Average			+ 46			+41			+11

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[a] Complete experimental data and details of furnaces used are given in a previous paper, The fusibility of coal ash in various atmospheres, by A. C. Fieldner and A. E. Hall, Jour. Ind. and Eng. Chem., vol. 7, 1915, pp. 399–406, 474–481.

INFLUENCE OF FINENESS OF ASH.

Ash ground to an impalable powder tended to soften at a slightly lower temperature than 100-mesh ash, as is shown in Table 4, following. The difference averaged 3° and in no case exceeded 40° C. Ash pulverized to at least 200 mesh could be molded into more substantial cones than 100-mesh material.

TABLE 4.—*Influence of fineness of ash on softening temperature.*

[Tests made in molybdenum furnace No. 2; ¼ by 1 inch cone placed in a vertical position; rate of heating, 2° C. per minute.]

Ash sample No.	Softening point.		Difference.	Ash sample No.	Softening point.		Difference.
	100-mesh.	Impalpable powder.			100-mesh.	Impalpable powder.	
	° *C.*	° *C.*	° *C.*		° *C.*	° *C.*	° *C.*
39	1,291	1,283	+ 8	67	1,223	1,185	+38
9	1,248	1,304	−56	68	1,216	1,205	+11
42	1,340	1,313	+27	69	1,251	1,237	+14
48	1,380	1,390	−10	70	1,259	1,263	− 4
8	1,435	1,481	−46	71	1,327	1,300	+27
19	1,519	1,510	+ 9	72	1,368	1,350	+18
2	1,492	1,520	−28	73	1,380	1,370	+10
64	1,108	1,101	+ 7				
65	1,155	1,127	+28	Average			+ 3
66	1,174	1,173	+ 1				

INFLUENCE OF INCLINATION OF CONES.

Mounting the cones with a considerable inclination—25° or 45° from the vertical—led to premature deformation points in some samples, which were caused by a further bending over due to shrinkage of the cone in its base, rather than deformation due to softening and flowing of the ash. As shown in Table 5, following, differences of 64° and 65° C. were obtained with two samples. Vertical or nearly vertical cones were free from this source of error and gave the most concordant results.

TABLE 5.—*Influence of inclination of cones on softening temperature.*

[Tests made in molybdenum furnace No. 2; size of cones, ¼ by 1 inch; ash ground to an impalpable powder; rate of heating 2° C. per minute.]

Ash sample No.	Softening point.			Difference.	
	Cone inclined 45° from vertical.	Cone inclined 25° from vertical.	Cone vertical.	45° and vertical.	25° and vertical.
	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*
39	1,287	1,283	1,283	+ 4	0
42	1,295	1,313	1,304	− 9	+9
9	1,248	1,304	1,313	−65	−9
33	1,396	1,390	1,390	+ 6	0
8	1,414	1,481	1,478	−64	+3
19	1,494	1,510	1,502	− 8	+8
2	1,520	1,520	1,520	0	0
Average				−19	+2

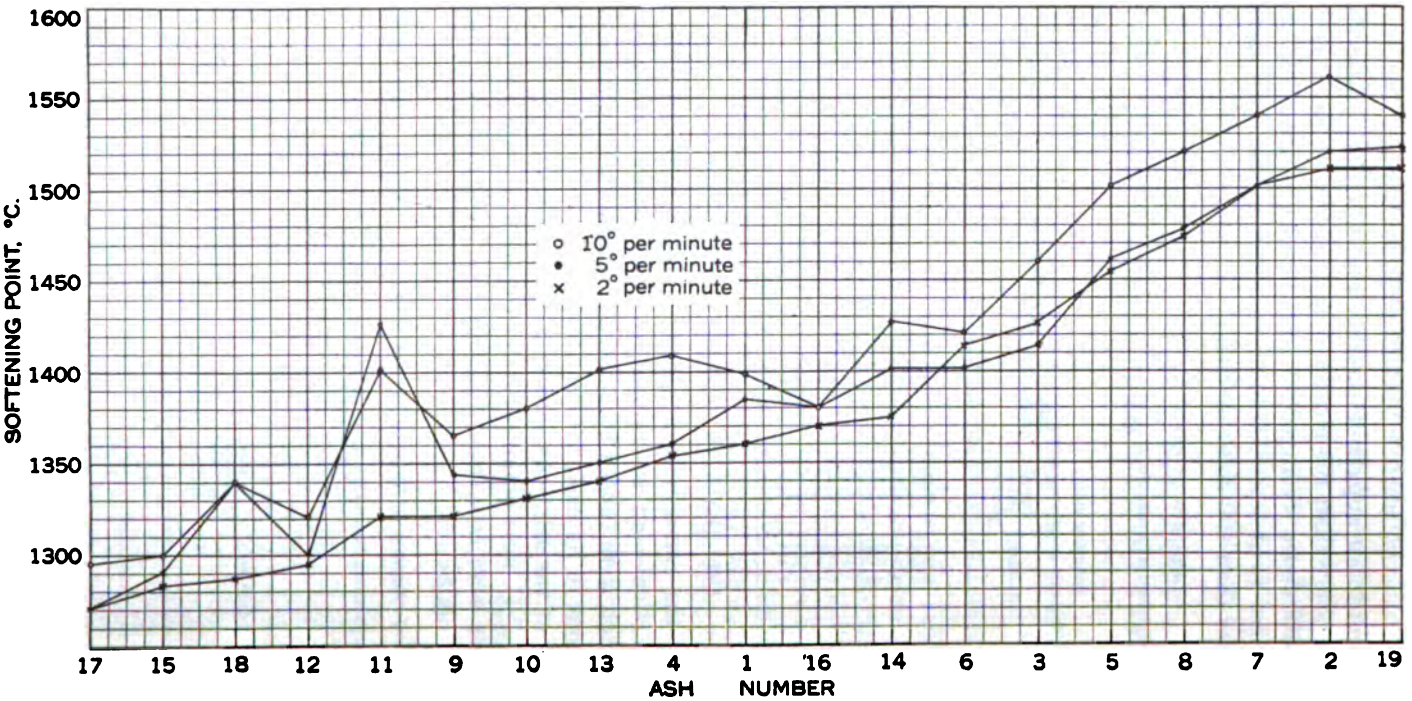

FIGURE 9.—Curves showing results of tests to determine influence of rate of heating on the softening temperature of ash cones in an atmosphere of hydrogen.

INFLUENCE OF RATE OF HEATING.

As stated previously, the rate of heating has an important bearing on the softening points of silicate mixtures made in the form of Seger cones. The fusion reactions of such mixtures require some time to approach a state of equilibrium. As it was considered probable that these reaction velocities in oxidizing atmospheres would differ from those in reducing atmospheres, tests were made at rates of heating of 10°, 5°, and 2° C. per minute in an oxidizing atmosphere of air in the platinum furnace, and in a reducing atmosphere of hydrogen in the molybdenum furnace. The softening points obtained are given in figures 9 and 10. These results show that varying the rate of heating from 2° to 10° C. per minute causes much less increase of softening temperature in oxidizing atmospheres than in reducing atmospheres of hydrogen. This would be expected because of the fact that the melting reactions in hydrogen are further complicated by reducing reactions, all of which are related to the time factor.

It is, of course, desirable to economize time by heating the test specimen as rapidly as is feasible. A safe maximum rate of heating in oxidizing atmospheres seems to be 10° C. per minute. However, in reducing atmospheres of hydrogen, several samples when subjected to a temperature rise of even 5° C. per minute gave results

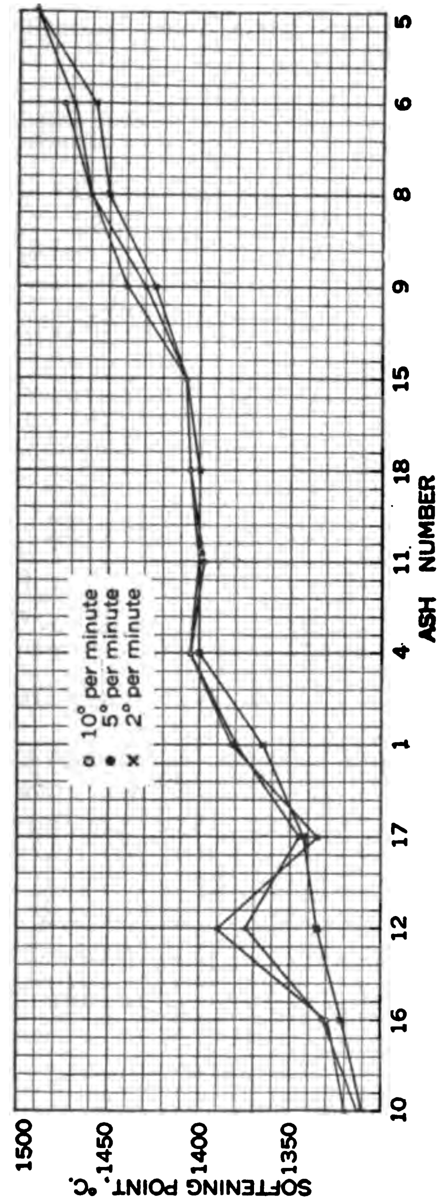

FIGURE 10.—Curves showing results of tests to determine influence of rate of heating on the softening temperature of ash cones in an atmosphere of air.

that were 50° C. higher than when subjected to a rise of 2° C. per minute. Further tests were therefore made in hydrogen, a combined 5° and 2° rate being used. The furnace temperature was increased at the rate of 5° C. per minute until deformation began; the rate was then reduced to 2° C. per minute until the cone was down. As shown by Table 6, the results approximated those obtained with the usual rate of 2° C.

TABLE 6.—*Results of tests showing comparative softening temperatures in molybdenum furnace No. 2 at two different rates of heating.*

[Ash ground to an impalpable powder; cone, $\frac{1}{16}$ by 1 inch, placed in a vertical position.]

Ash sample No.	Softening point. Temperature rise of 2° C. per minute.[a]	Softening point. Temperature rise of 5° and 2° C. per minute.[b]	Difference.
	° C.	*° C.*	*° C.*
58	1,199	1,220	−21
50	1,220	1,217	+ 3
49	1,520	1,514	+ 6
55	1,380	1,370	+10
28	1,390	1,394	− 4
52	1,448	1,478	−30
61	1,365	1,370	− 5
Average			− 6

[a] Furnace temperature was increased at the rate of 2° C. per minute until cone was down.

[b] Furnace temperature was increased at the rate of 5° C. per minute until deformation began; the rate was then reduced to 2° C. per minute until the cone was down.

INFLUENCE OF DEXTRIN BINDER.

The results of a few experiments made to show the effect of not burning out the dextrin binder before placing the ash cone in the molybdenum furnace are given in Table 7 following. The maximum apparent effect was a lowering of 11° C. in the softening point. Mixing 10 per cent of dry dextrin with the ash caused a reduction of only 14° C. However, the conclusion that dextrin causes no material effect would apply only under the conditions of determining the softening point in hydrogen and water-vapor mixtures as was done in the experiments mentioned.

TABLE 7.—*Results of experiments to determine influence of dextrin binder on softening temperature of ash cones.*

[Test made in molybdenum furnace No. 2; cones $\frac{3}{16}$ in. by 1 inch, inclined 35° from vertical; 100-mesh ash; rate of increase of heating 2° C. per minute.]

Ash sample No.	Softening point.		Difference.
	Dextrin binder not ignited.	Dextrin binder ignited.	
	°*C.*	°*C.*	°*C.*
2	1,492	1,502	−10
8	1,435	1,434	+ 1
12	1,260	1,271	−11
8	1,430	1,434	− 4
Average			− 6

INFLUENCE OF DIFFERENT CONDITIONS ON SOFTENING POINT OF ASH SAMPLE 8.

Conditions.	Softening point, °C.
Cone made in usual manner, then ignited in air at 850° C.	1,434
1 per cent dry dextrin mixed with the ash; dextrin not ignited	1,427
10 per cent dry dextrin mixed with the ash; dextrin not ignited	1,420

INFLUENCE OF VARIOUS ATMOSPHERES.

None of the various factors affecting softening temperatures that have thus far been discussed seems capable of causing the exceedingly large differences in softening temperature that have been reported from different laboratories with the same samples of ash. For instance, Marks[a] has called attention to differences as great as 390° C. that were obtained with the same sample of ash by two different commercial laboratories. He himself obtained differences of 200° C. with some duplicate samples that were tested in a Meker muffle furnace and a Hoskins graphite electric furnace. As stated by Marks, these large differences were undoubtedly due to the oxidizing or reducing action of the respective furnace atmospheres. It was realized at the outset of the investigation described in this report that the reaction of the furnace gases with the iron constituent of the ash would largely determine the softening temperature. Therefore, comparative tests were made in the six different furnaces previously described in order to show the effect of various oxidizing and reducing atmospheres.

The first series of oxidizing-atmosphere tests was made in at atmosphere of air in a platinum resistance furnace (fig. 2, p. 22), in which there could be no trace of reducing gases. The second series of tests was made in a gas-heated Meker muffle furnace (fig. 3, p. 22), and the third series in a fire-clay gas muffle-furnace. As shown in figure 11, the platinum-furnace results were higher than those obtained in the gas furnaces. There was evidence of reduction by products of incomplete combustion from the fuel gases surrounding the muffles. The reduction was seemingly greater in the muffle furnace than in the gas furnaces. There was evidence of reduction by products of incomplete combustion from the fuel gases surrounding the muffles. The reduction was seemingly greater in the muffle furnace than in the Meker furnace, probably because the muffle furnace contained a rather porous fire-clay muffle, whereas the Meker furnace contained a special refractory muffle of dense texture.

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[a] Marks, L. S., The clinkering of coal: Jour. Am. Soc. Mech. Eng., vol. 37, 1915, pp. 205–214.

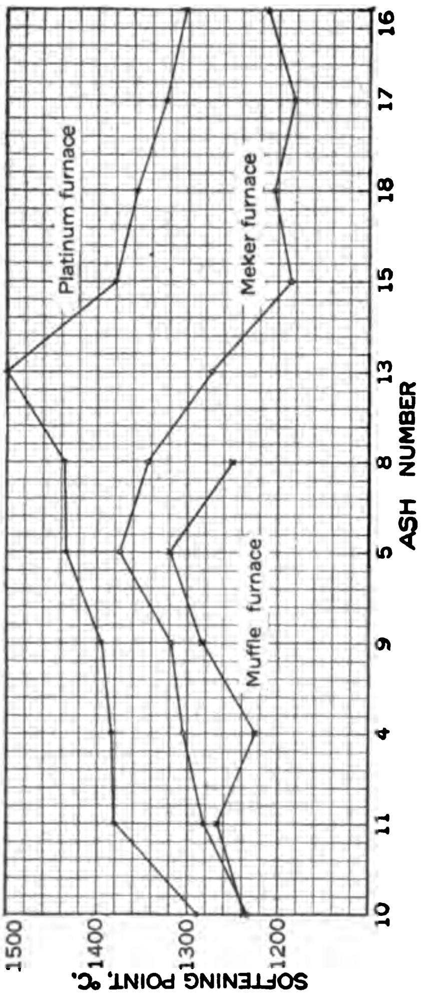


FIGURE 11.—Curves showing results of tests to determine softening temperatures in Meker furnace, muffle furnace No. 2, and platinum furnace.

From these results it is evident that softening-temperature tests made in gas furnaces must be made with the furnace operated under definite oxidizing or reducing conditions if duplicate results are to be obtained.

In the tests made in the gas furnaces there was evidence of some reduction of ferric iron to the ferrous form, but not to metallic iron. To show the effect of strongly reducing atmospheres such as would cause the formation of metallic iron, tests were made in a granular-carbon resistance furnace (fig. 5, p. 24) and a Northrup graphite furnace (fig. 6, p. 25), both having atmospheres of carbon monoxide (CO), and in a molybdenum resistance furnace (fig. 4, p. 23) having an atmosphere of hydrogen. The results obtained in the carbon, the Northrup, and the platinum furnaces are given in figure 12. As would be expected, the strongly reducing atmosphere of carbon monoxide completely changes the softening characteristics of the ash, so that results widely different from those found in the platinum furnace in air were obtained.

As both the Northrup and the granular-carbon resistance furnace produced atmospheres of carbon monoxide, similar softening temperatures were expected in the two series; that such did not prove to be the result is shown in figure 12. In 16 of 18 samples tested, the softening points in the Northrup furnace were higher than in the carbon furnace; the maximum was 396° C. and the average difference for the series was 134° C. The checks made on materials of known melting points, at various times during the course of these experiments, effectually rule out the possibility of attributing these large differences of temperature measurement. The only explanation offered by the authors is that in the carbon furnace the reduction of ferric oxide to metallic iron did not proceed as rapidly or as completely as in the Northrup furnace; in the latter the ash cone is heated in a closed tube of graphite, practically all air being excluded, so that only nitrogen, carbon monoxide (CO), and a fractional percentage of carbon dioxide (CO_2) surrounds the ash at temperatures above 1,100° C. Such an atmosphere would reduce the ferric oxide to metallic iron and thus prevent the formation of a low-melting ferrous silicate eutectic. On the other hand, the carbon furnace had an alundum tube between the carbon resistor and the cone, and it also permitted air to leak into the heating space as shown by the appearance of carbon monoxide flames at the peephole and around the cover plate. Sufficient carbon dioxide may therefore have been present to retard the reduction to metallic iron, thus leaving enough of the strongly fluxing ferrous oxide in the ash to form a low-melting fluid eutectic which caused the cone to collapse at a comparatively low temperature. Examination of polished sections of fused cones from both furnaces disclosed more metallic iron in those from the Northrup furnace.

Figure 13 shows the results obtained in the series of tests in the hydrogen atmosphere of the molybdenum furnace as compared with the results of the tests in the platinum and the Northrup furnaces. It is quite evident that the softening temperature in hydrogen is different from that in air or carbon monoxide.

The softening temperatures obtained in the six different furnaces are given in Table 8 following. The maximum differences range from 134° to 396° C. As regards 14 samples that softened below 1,500° C. in the platinum furnace in air, 9 gave their highest results in the strongly reducing carbon monoxide atmosphere of the Northrup furnace (see fig. 12); and 5 gave their highest results in the oxidizing atmosphere of the platinum furnace. The lowest softening points in 18 samples were distributed among the different furnaces as fol-

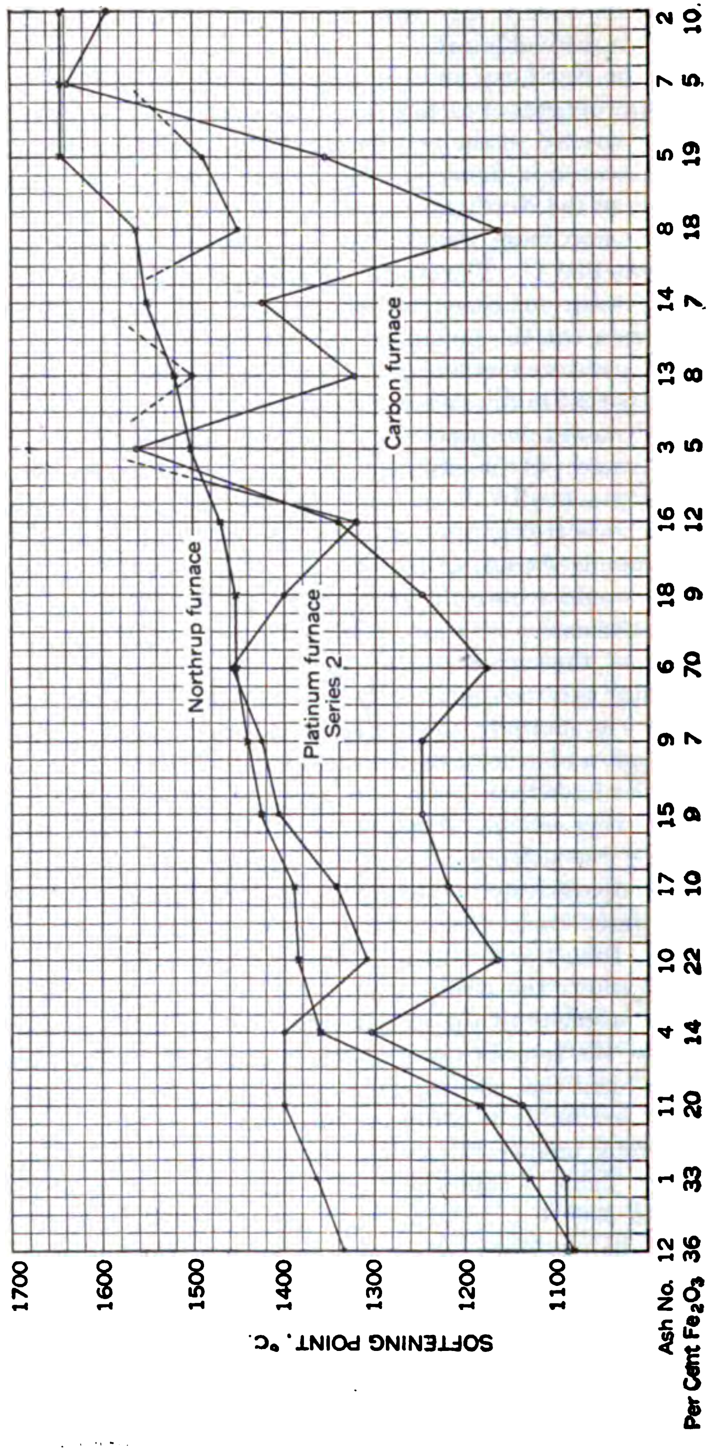


FIGURE 12.—Curves showing results of tests to determine softening temperatures in Northrup, carbon-resistance, and platinum furnaces.

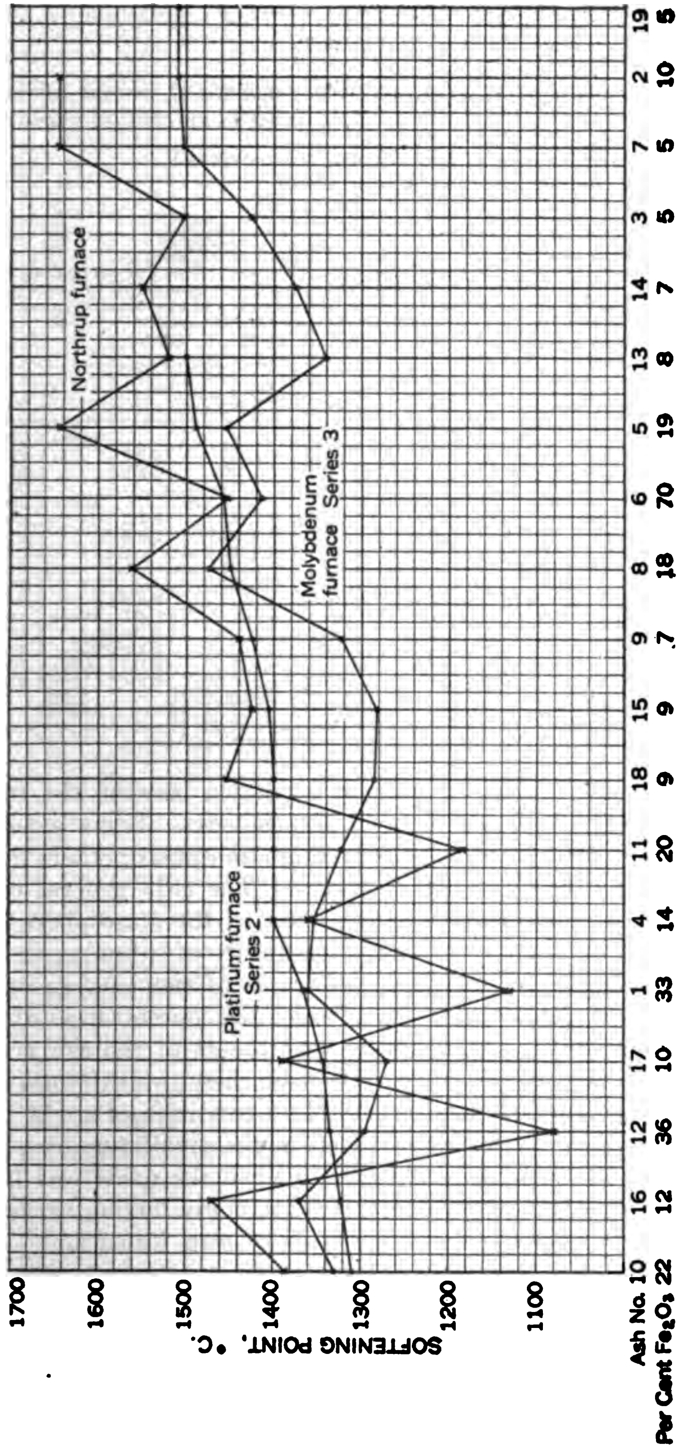


FIGURE 13.—Curves showing results of tests to determine softening temperatures in Northrup, platinum, and molybdenum furnaces.

lows: Carbon furnace, 6; Meker furnace, 5; molybdenum furnace, 4; muffle furnace, 2; Northrup furnace, 1.

TABLE 8.—*Results of tests to determine influence of various atmospheres on softening temperatures of ash cones in different furnaces.*

[Rate of heating, 2° C. per minute.]

Ash sample No.	Softening point.						Maximum difference.
	Northrup furnace,[a] in atmosphere of CO+N.	Carbon furnace,[b] in atmosphere of CO+CO_2+N.	Molybdenum furnace,[b] in atmosphere of H.	Muffle furnace,[b] in atmosphere of air+combustion gases.	Meker furnace,[b] in atmosphere of air+combustion gases.	Platinum furnace,[a] in atmosphere of air.	
	°*C.*	°*C.*	°*C.*	°*C.*	°*C.*	°*C.*	°*C.*
1	1,131	1,090	1,360	1,192	....	1,365	275
2	1,645	1,593	1,511	....	....	Above 1,500	134
3	1,502	1,562	1,427	....	....	Above 1,500	135
4	1,360	1,306	1,355	1,226	1,305	1,400	174
5	1,645	1,356	1,455	1,320	1,376	1,490	325
6	1,455	1,179	1,414	1,318	....	1,458	279
7	1,645	1,638	1,502	....	....	Above 1,500	143
8	1,562	1,166	1,474	1,251	1,343	1,450	396
9	1,440	1,249	1,822	1,284	1,318	1,425	191
10	1,385	1,167	1,331	1,199	....	1,310	218
11	1,185	1,139	1,322	1,214	....	1,400	261
12	1,080	1,088	1,295	1,182	....	1,335	255
13	1,520	1,322	1,340	....	1,273	1,500	247
14	1,551	1,424	1,375	....	1,417	Above 1,500	176
15	1,427	1,250	1,283	....	1,186	1,407	241
16	1,470	1,341	1,370	....	1,212	1,322	258
17	1,390	1,220	1,271	....	1,182	1,343	208
18	1,455	1,250	1,287	....	1,204	1,400	251

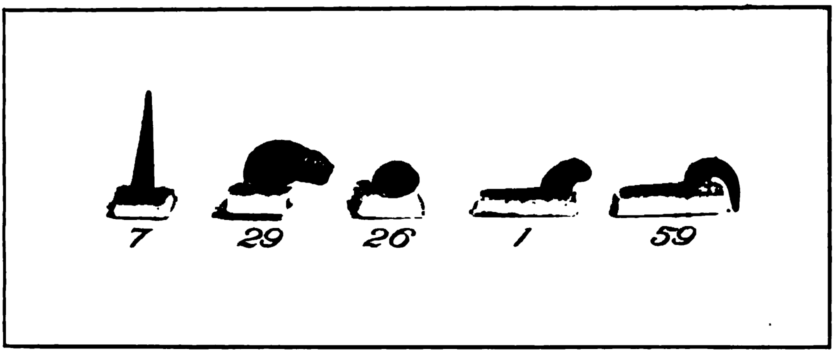
[a] Ash ground to an impalpable powder; cones ⅛ by 1 inch, placed in vertical position.
[b] Ash 100 mesh; cones ½ inch by 1½ inches, inclined 35° from vertical.

There seems to be no consistent relation between the results obtained in the different furnace atmospheres. Some ashes have a higher softening point in an oxidizing atmosphere; others in a reducing atmosphere of carbon monoxide. The only general conclusion that can be made regarding the effect of the atmosphere is that a given ash will have its highest temperature of softening in either an atmosphere of air, free from all traces of reducing gases, or in a strongly reducing atmosphere in which the concentration of reducing gases is so great that the iron constituents of the ash are largely reduced to the metallic form. Between these two extremes of oxidation and reduction, there seems to be a condition of slight reduction by mixed gases such as was found in the muffle furnaces, in which reduction of ferric oxide proceeded until the oxide was largely in the ferrous state. On account of its strongly fluxing action ferrous oxide greatly lowers the temperature of slag formation.

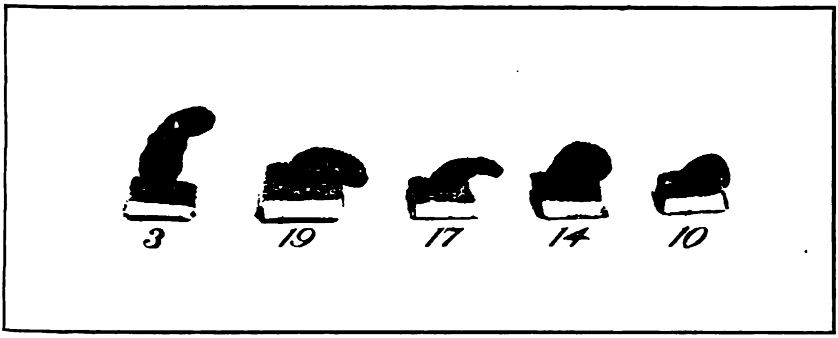
APPEARANCE OF CONES SOFTENED IN VARIOUS ATMOSPHERES.

PLATINUM FURNACE, ATMOSPHERE OF AIR.

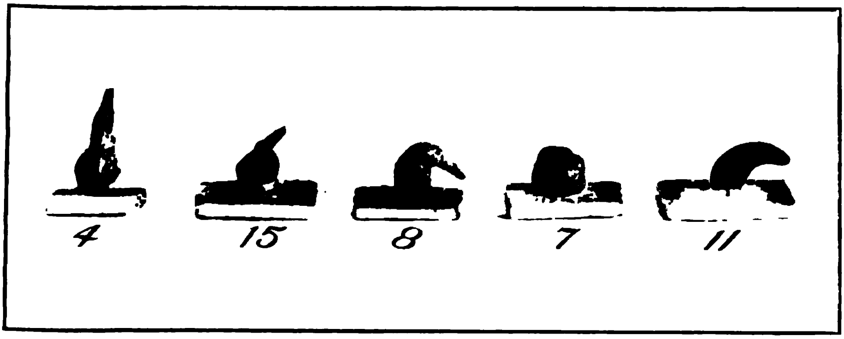
In Plate I, *A*, are shown the various shapes into which cones of different ashes deform when heated in air in the platinum furnace. The manner of deformation is somewhat characteristic for each indi-



A. TYPICAL FORMS OF CONES FUSED IN AIR IN THE PLATINUM FURNACE.



B. TYPICAL FORMS OF CONES FUSED IN HYDROGEN IN THE MOLYBDENUM FURNACE.



C. TYPICAL FORMS OF CONES FUSED IN CARBON MONOXIDE IN THE NORTHRUP FURNACE.

vidual ash. No. 59 illustrates the typical form of bending of Seger cones; No. 26 illustrates the type that fuses to a lump; No. 29 is representative of the intumescing type; No. 7 is an ash which did not soften at 1,500°. The color of the fused cones varied from a buff, or reddish brown, to a glossy black. The black cones were magnetic.

MOLYBDENUM FURNACE, ATMOSPHERE OF HYDROGEN.

Plate I, *B*, illustrates typical forms from the molybdenum furnace (hydrogen atmosphere). Owing to the reducing atmosphere, these cones almost always developed a pronounced vesicular structure. The swelling due to the evolution of gas in some cones, as in ash No. 3, prevented the cones from bending in the normal manner. This is one of the reasons for the somewhat larger variations between duplicate determinations in reducing atmospheres. The fused cones were usually light to dark gray or grayish black; they had a vitreous luster and invariably showed iron globules when examined under the microscope.

NORTHRUP FURNACE, ATMOSPHERE OF CARBON MONOXIDE AND NITROGEN.

The cones fused in the Northrup furnace (Pl. I, *C*) (CO atmosphere) varied from a dark-gray to a brownish-black or metallic-black color. The surface lacked the vitreous luster of the cones fused in air or hydrogen. Carbon was deposited on the cones and the bases under the cones in increasing quantity as the temperature exceeded 1,400° C., as shown by the blackened base of ash No. 8 (Pl. I, *C*), which was heated to 1,550° C.

Another peculiarity in the cones tested in the Northrup furnace was the softening and swelling of the thick bottom parts of some of the cones before the apexes began to soften. Ash No. 4 began swelling at the base at 1,200° C.; heating was continued at a temperature increase of 2° C. per minute until a temperature of 1,375° C. was reached, when the current was interrupted and the furnace allowed to cool. As shown in Plate I, *C*, fusible constituents appeared to separate and run down the sides of each cone, leaving a refractory skeleton standing. However, when the cone was broken, the irregular contour appeared to be due to the evolution and expansion of gases in the interior of the softening slag. The expanded bottom was a thin-shelled bubble of slag.

The first perceptible swelling of the lower part of ash No. 8 took place at 392° C. below the temperature at which it finally bent over in the position shown in Plate I, *C*.

The earlier fusion of the thick part of the cone may be due to the inability of the carbon monoxide to penetrate and reduce to metallic iron the iron oxide in the interior as rapidly as in the thin upper part. This phenomenon of irregular softening was noted only in the Northrup and the carbon-resistance furnaces.

STANDARD METHODS FOR DETERMINING SOFTENING TEMPERATURE OF ASH.

A standard method for determining the softening temperature of ash may now be considered. Obviously, the statement that an ash has a "fusing" or softening temperature of 1,500° C. means nothing unless the exact conditions of making the test are defined. The method must therefore be carefully standardized in all its details before directly comparable results may be obtained by different workers.

The most important consideration is the atmosphere in which the test shall be made. In order to obtain results that can be duplicated in different laboratories, the softening temperature in air, as was determined in the platinum furnace, is undoubtedly the most satisfactory method, but, unfortunately, ash forming in a fuel bed is not free from the effect of reducing gases. Consequently, clinkering may take place under conditions of partial reduction similar to those that prevailed when the low softening points were obtained in the experiments described. Therefore the danger point as regards clinker formation is the lowest possible softening temperature of an ash under such conditions of reduction as are probable in a fuel bed.

The problem resolves itself into an investigation of the effects of various mixtures of a reducing gas and its product of oxidation on the softening temperature with a view to finding that mixture which will give the lowest softening temperature. The two principal reducing gases to be considered are hydrogen and carbon monoxide, the respective oxidation products being water and carbon dioxide.

FUSIBILITY OF COAL ASH IN MIXTURES OF HYDROGEN AND WATER VAPOR.

THEORETICAL CONSIDERATIONS.

The equilibrium between hydrogen, water vapor, Fe, and Fe_3O_4 according to the reaction,

$$3Fe+4H_2O \rightleftarrows Fe_3O_4+4H_2 \qquad (1)$$

was first studied by Deville[a] at various temperatures between 200° and 1,000° C., and later with greater precision by Preuner[b] at three temperatures. Preuner found the following mean values for the equilibrium constant:

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[a] Deville, H. Sainte-Claire, Action de l'eau sur le fer et le hydrogène sur l'oxyde de fer: Compt. rend., t. 70, 1870, pp. 1105–1111 and 1201–1205.

[b] Preuner, G., Das Gleichgewicht zwischen Eisen, Eisenoxyduloxyd, Wasserstoff, und Wasserdampf: Ztschr. phys. Chem., Bd. 47, 1904, pp. 385–417.

Equilibrium data of Preuner.

Temperature of solid phase, °C.	p_{H_2O}.
900	0.69
1025–1050	.78
1150	.86

Chaudron[a] has since reported two series of equilibria of iron and its oxides in hydrogen and water vapor corresponding to the following reactions:

$$Fe+H_2O \rightleftarrows FeO+H_2 \qquad (2)$$
$$3FeO+H_2O \rightleftarrows Fe_3O_4+H_2 \qquad (3)$$

His values for the equilibrium constants are as follows:

Equilibrium data of Chaudron.

Reaction 2.		Reaction 3.	
Temperature of solid phase.	$\frac{p_{H_2O}}{p_{H_2}}$	Temperature of solid phase.	$\frac{p_{H_2O}}{p_{H_2}}$
°*C.*		°*C*	
360	0.12	440	0.24
440	.18	480	.32
500	.23	500	.33
550	.32	615	.54
600	.31	700	1.00
630	.36	710	1.00
660	.39	800	1.35
760	.55	860	2.00
875	.74		

No similar data are available on the behavior of ferric oxide (Fe_2O_3); hence these data for magnetite include the only available information as to the probable or possible phases in which the iron oxide of coal ash may exist in various mixtures of hydrogen and water vapor at the sintering temperature of ash.[b]

In figure 14 are plotted the equilibrium data of Preuner and of Chaudron. Assuming that Fe_2O_3 reacts as regards reduction in a similar manner as does Fe_3O_4, it is evident that at temperatures up to 1,150° C. no reduction to metallic iron can take place in a mixture of equal parts by volume of hydrogen and water vapor. At a temperature of 850° C., according to Chaudron's figures, FeO would be the stable solid phase in mixtures containing not less than 43 or more than 63 per cent of water vapor, it being assumed that a gas mixture of constant composition is passed over the heated oxide continuously, sweeping the gaseous products of the reaction out of the apparatus.

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[a] Chaudron, G., Réactions réversibles de l'eau sur le fer et sur l'oxyde ferreux: Compt. rend., t. 159, 1914, pp. 237–239.

[b] Recent investigations by Sosman and his coworkers at the geophysical laboratory have shown that the equilibrium relationships of the iron oxides are further complicated by solid solution. In the system Fe_2O_3–Fe_3O_4 there is a continuous solid solution series from Fe_2O_3 to Fe_3O_4.

It seems possible, therefore, that a minimum softening temperature of coal ash may be found in a mixture approximating equal parts by volume of hydrogen and water vapor, provided the heating is sufficiently slow to allow the reaction to approach equilibrium at the various temperatures.

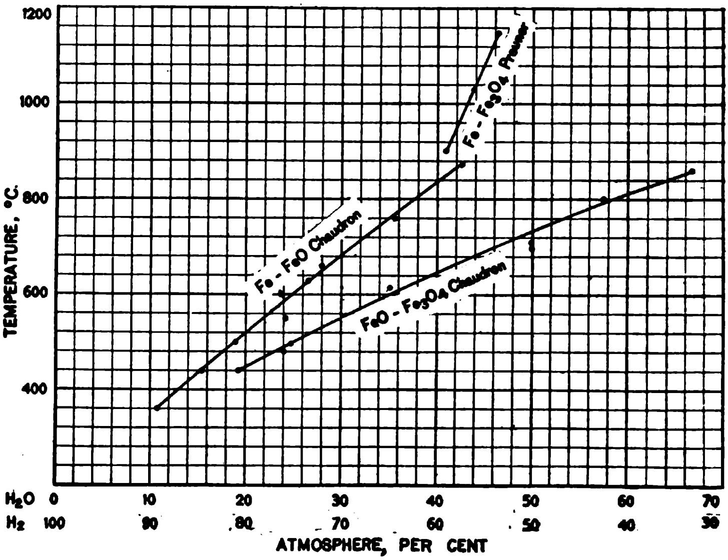


FIGURE 14.—Equilibrium diagram of the system H_2, H_2O, Fe, FeO, and Fe_3O_4.

OUTLINE OF EXPERIMENTS.

DESCRIPTION OF FURNACE.

The arrangement of the furnace and of the saturator used in the authors' tests is shown in Plate II, and figures 15 and 16. A Heræus

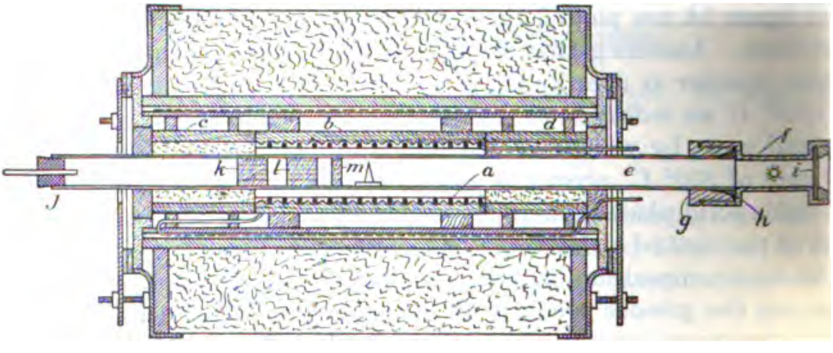
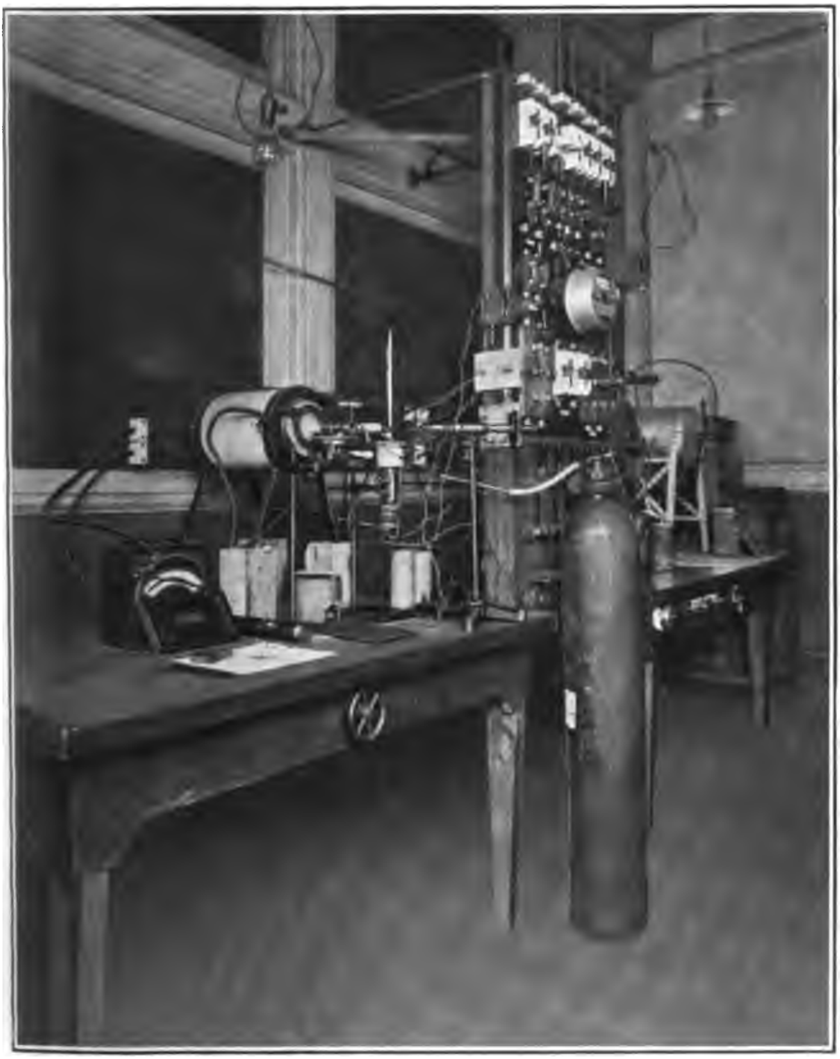


FIGURE 15.—Vertical longitudinal section through silica-tube furnace.

platinum-foil resistance furnace which happened to be available was equipped with an extra heater of $\frac{8}{100}$-inch platinum wire, *a* (fig. 15),



ARRANGEMENT OF SILICA-TUBE FURNACE AND ACCESSORIES.

wound on the inside of the alundum tube *b*, and connected in series with the platinum-foil[a] heating unit of the Heræus furnace.

This alundum heater tube was supported on two alundum rings, leaving a ½-inch air space between the heater tube and the Heræus porcelain tube. Two additional alundum tubes, *c* and *d*, were placed similarly, one at each end of the platinum-wound tube. Through the furnace, modified in this way, was passed a tube *e*, of fused silica, glazed on both sides, five-eighths inch in inside diameter and 17 inches long.

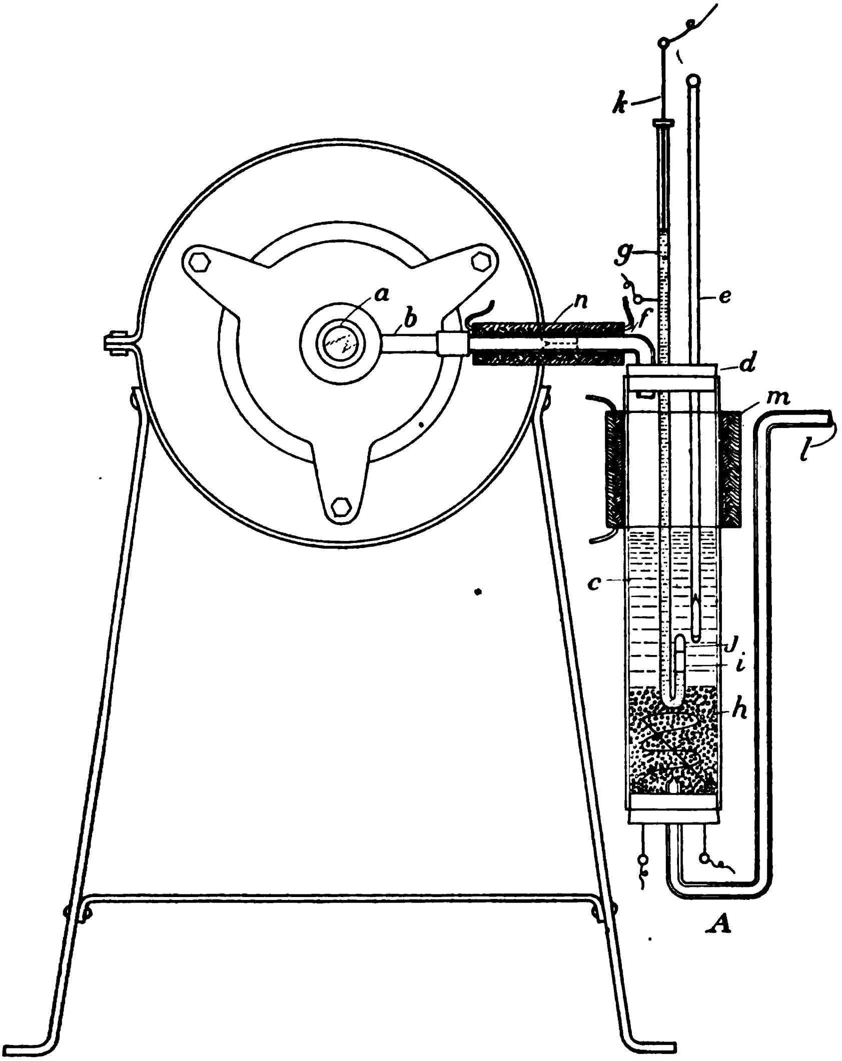


FIGURE 16.—Front elevation of silica-tube furnace and vertical section through saturator.

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[a] A nichrome heating element could as well be used for the outer heating unit.

The gas mixture entered the silica tube through the brass end piece *f*, which was specially designed to make a gas-tight connection by means of the brass bushing *g* and the asbestos packing *h*. A removable glass window, *i*, permitted removal and insertion of cones in the furnace, and observation of deformation and of temperature with the optical pyrometer. The glass was held in place between asbestos washers by a threaded brass ring.

An air bath was placed around the end piece *f* to maintain its temperature above 100° C., and thus avoid any condensation of water vapor; the exit end *g* was made sufficiently short so that its temperature should be above 100° C. The gas mixture passed out through the glass capillary tube inserted in the rubber stopper at *j*.

Alundum stops, *k*, *l*, and *m*, were placed at intervals back of the middle of the inner heater to provide a suitable background for taking readings with the optical pyrometer. These stops were made of thin-walled alundum tubes closed at one end, fitting snugly in the silica tube. To avoid rapid devitrification of the fused-silica tube, temperatures were not carried above 1,400° C.

DESCRIPTION OF "SATURATOR."

The apparatus by which various mixtures of hydrogen and water vapor were formed depended on the saturation of pure hydrogen gas with water vapor at temperatures ranging from zero to within a degree or two of the boiling point. It is shown in figure 16 with furnace connections as used, except the hot-air bath surrounding the brass end piece *a*, and the side tube *b*.

The saturator consists of a glass tube *c*, 1¾ inches in diameter and 9 inches long, closed at the ends with rubber stoppers. Through the upper stopper *d* passes the thermometer *e*, the outlet tube *f* communicating with the furnace through the brass tube *b*, and a compact adjustable vapor-pressure thermoregulator *g*, with electrical connections controlling through a relay the current in the heating coil *h*. The thermoregulator is essentially that described by Feild,[a] without the refinement of being independent of atmospheric pressure. It consists of a bent-glass tube, in the short arm of which is a small volume of liquid *i*—ether, alcohol, or water, depending on the temperature desired—resting on a mercury column. Above the liquid *i*, is a small volume of its saturated vapor *j*, the pressure of which depends only on the temperature and determines the height of the mercury column. This mercury column makes and breaks contact with the platinum wire *k*. The temperature control, with a continuous stream of hydrogen bubbling around the glass beads *h* and through the water *c* was within 0.1° to 0.2° C., which was sufficiently accurate for the purpose of the experiments. Hydrogen stored in a high-pressure cylinder was introduced at *l*, and its flow so adjusted as to give the desired current of hydrogen and water vapor through the fused-silica furnace tube. The water vapor present in the mixture of hydrogen and water vapor was prevented from condensing to liquid water on its passage to the furnace by two heating coils *m* and *n*, constructed of nichrome wire wound on alundum cores and heated, similarly to the immersed coil *h*, by a current from a low-voltage transformer operating at about 11 volts. An air-bath chamber of asbestos paper, heated by a small electric hot plate, was placed around the brass end piece *a* and the side tube *b*.

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[a] Feild, A. L., An electrical-contact vapor-pressure thermoregulator: Jour. Am. Chem. Soc., vol. 36, 1914, pp. 72–76.

Analyses were made of the mixtures of gas and water vapor issuing from the end of the silica tube. With the saturator adjusted for a mixture containing 50 per cent of each, 52 per cent hydrogen and 48 per cent water vapor were found; adjusting for an atmosphere of 89 per cent water vapor, 91 per cent was found. This degree of accuracy was considered satisfactory.

PREPARATION OF TEST PIECE.

The ash ground to 200 mesh was molded with the aid of dextrin solution into a slender triangular pyramid or Seger cone and mounted, with one side vertical, in a refractory base, according to the method previously described. On account of the small diameter of the fused-silica tube, it was possible to use only the upper half of the standard three-sixteenth by 1 inch cone adopted in the former experiments. Before the cone was placed in the furnace the dextrin binder was burned off by ignition at a red heat in a muffle.

MEASUREMENT OF TEMPERATURE.

The temperatures were read with a Wanner optical pyrometer, which was sighted on the thin alundum disk just back of the cone. A correction for absorption by the glass window was made.

The pyrometer was compared daily with the standard amyl-acetate flame, and once each week a thin piece of Kahlbaum's pure copper foil was substituted for the ash cone and its melting point read in an atmosphere of hydrogen. The results were between 1,075° and 1,090° C. (melting point, 1,083° C.).

RESULTS OF EXPERIMENTS.

Softening-temperature determinations were made with five different samples of coal ash (see Tables 1 and 2, pp. 26 and 27) containing

7 to 69 per cent of ferric oxide (Fe_2O_3), in various atmospheres ranging from pure dry hydrogen to pure water vapor. The temperature was raised at the rate of 10° to 15° C. per minute to approximately 900° C.; it was then raised 5° C. per minute until deformation began, and 2° C. per minute thereafter until the cone was down. The dry hydrogen was obtained by bubbling the hydrogen through a wash bottle of concentrated sulphuric acid. The 100 per cent water-vapor atmosphere was obtained by cutting off the hydrogen from the saturator and heating the water to boiling.

All the results obtained are given in Table 9 following, in which the softening temperature is given as a temperature interval, the lower figure being the temperature at which deformation (rounding of the apex or bending of the cone) began, and the higher figure being the temperature at which the cone bent so that the apex touched the base, or, failing to bend, softened to a lump. This latter final deformation point is meant when reference is made to the "fusion," "softening," or "deformation" point of an ash.

TABLE 9.—*Results of experiments to determine softening temperatures of ash in various mixtures of hydrogen and water vapor.*

Laboratory No.......	15848	16586	15844	16243	15845
Percentage of Fe_2O_3..	7.0	11.8	19.0	35.9	69.7
Percentage of H_2O in atmosphere.	Softening temperature.				
	°*C.*	°*C.*	°*C.*	°*C.*	°*C.*
0.0........	1,322-1,360	1,390-1,402	Above 1,400	1,304-1,370	Above 1,400
0.7........	....	....	....	1,331-1,355	....
5.7........	1,279-1,290	1,313-1,360	Above 4,000	....	Above 1,400
6.3........	....	....	....	1,248-1,263	....
7.6........	....	....	....	1,220-1,270	....
10.7........	....	....	[a] 1,340	....	....
12.4........	....	1,279-1,313	1,155-1,271	....	....
15.8........	....	....	....	....	1,304-1,330
18.1........	....	....	....	1,104-1,198	....
19.9........	....	....	1,173-1,271	....	....
29.6........	....	....	....	1,056-1,064	....
31.2........	1,179-1,271	1,227-1,251	1,173-1,263	....	1,227-1,322
41.7........	....	....	....	1,048-1,062	....
51.2........	....	....	....	1,072-1,078	....
57.5........	1,220-1,292	1,220-1,258	1,173-1,255	....	1,104-1,322
77.2........	....	....	....	[a] 1,095	....
83.2........	....	....	....	[a] 1,095	....
83.9........	1,173-1,295	1,241-1,249	1,227-1,263	....	1,206-1,350
90.3........	....	1,227-1,251	1,206-1,295	[a] 1,095	....
94.8........	....	....	....	1,048-1,080	....
100.0........	1,350-1,370	1,313-1,322	Above 1,400	1,304-1,313	Above 1,402

[a] Temperature at end of deformation of cone; temperature at which deformation began not observed.

The final deformation points are plotted in figure 17, in which the abscissas represent the percentages of water vapor in the mixture, and the ordinates the temperatures of final deformation. The initial deformation points are plotted in a similar manner in figure 18.

It will be noted that both sets of curves have well-defined minima when the water vapor formed 40 to 70 per cent of the atmosphere.

The final deformation points (fig. 17), which were more accuately determined, show a fairly uniform minimum in mixtures ranging from 30 to 70 per cent water vapor. The establishment of this relation enables the formulation of a practical method of determining the lowest softening temperature of an ash in a mixture of approximately equal parts of hydrogen and water vapor. Seemingly, the gas mixture need only be controlled between limits of 30 and 70 per cent water vapor to insure the lowest softening temperature of the ash, within the unavoidable experimental deviation in determininations of this character.

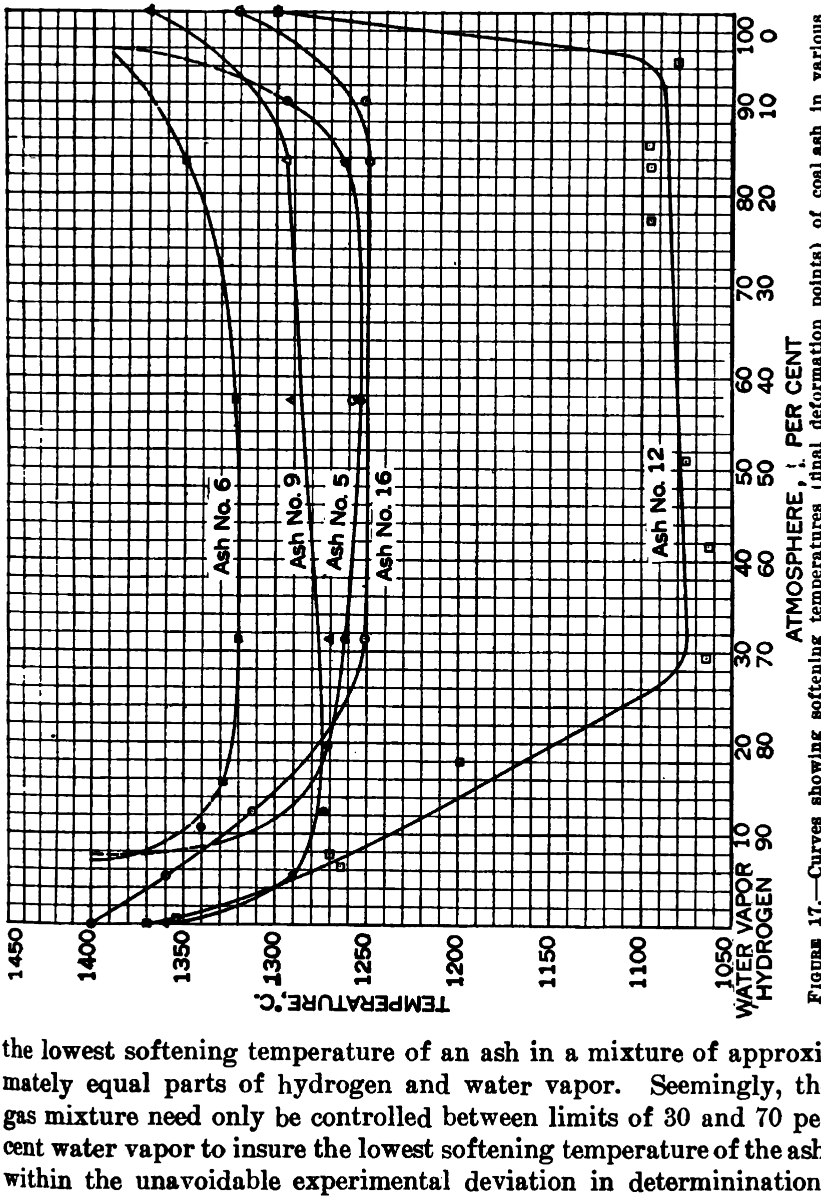

FIGURE 17.—Curves showing softening temperatures (final deformation points) of coal ash in various mixtures of hydrogen and water vapor.

The pronounced effect of viscosity on the rate of deformation of the ash cones is shown in the relative positions of the curves of initial deformation in figure 18 and the curves of final deformation in figure 17. Sample 16586 had the highest initial-deformation temperature and next to the lowest final-deformation temperature in an atmosphere containing equal parts of hydrogen and water vapor. This was due to the low viscosity of the melting eutectic formed from this ash, which contained 12.6 per cent of calcium oxide, a larger percentage of this basic oxide than was contained in any of the other samples.

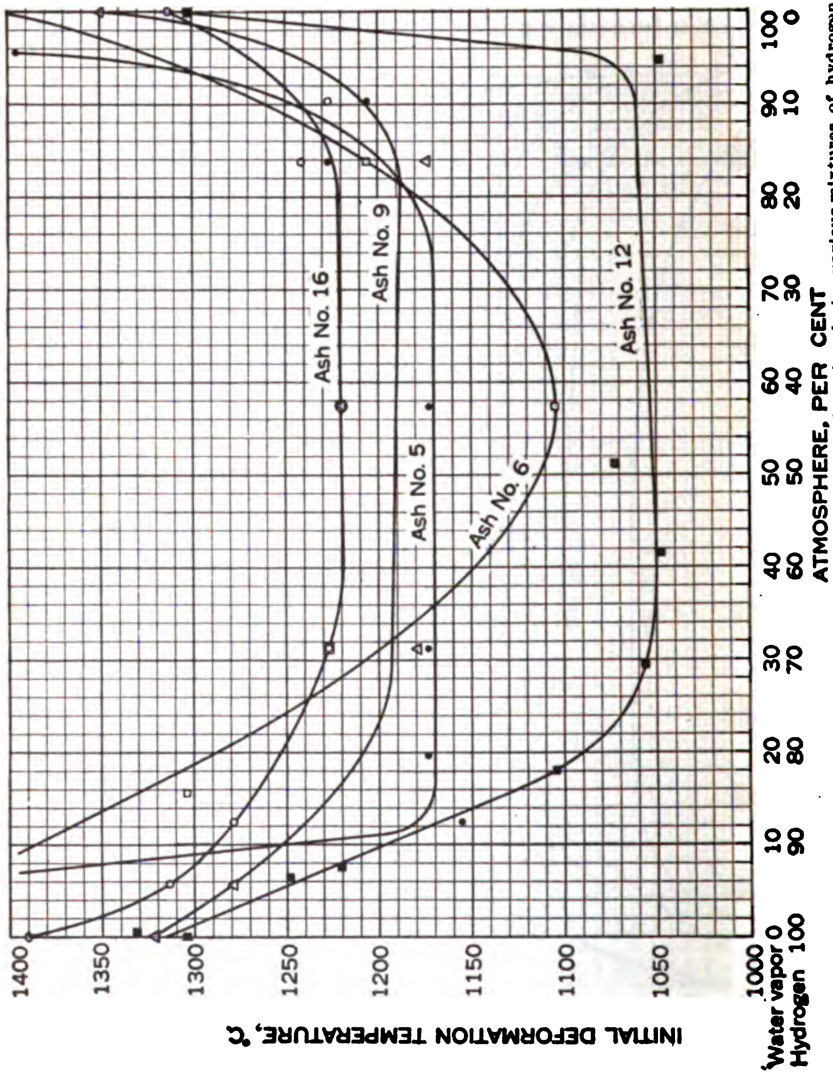

FIGURE 18.—Curves showing initial deformation temperatures of coal ash in various mixtures of hydrogen and water vapor.

FUSIBILITY OF COAL ASH IN MIXTURES OF CARBON MONOXIDE AND CARBON DIOXIDE.

The investigations of Kreisinger, Ovitz, and Augustine[a] have shown that, with ordinary rates of combustion, oxygen is not found in appreciable quantities in a fuel bed at distances greater than 3 or 4 inches above the grate. Samples of gas taken at a point 3 inches above the grate contained approximately 12 per cent combustible gas (principally CO) and 14 per cent carbon dioxide. In view of this predominating proportion of CO and CO_2 it is even more important to know the effect of mixtures of these two gases on the fusibility of coal ash than to know the effect of hydrogen and water vapor.

THEORETICAL CONSIDERATIONS.

Baur and Glaessner[b] found the following two series of equilibria in the system CO, CO_2, Fe, FeO, and Fe_3O_4:

$$Fe+CO_2 \rightleftarrows FeO+CO \qquad (1)$$

$$3FeO+CO_2 \rightleftarrows Fe_3O_4+CO \qquad (2)$$

Their average values for the equilibrium constants are as follows:

Equilibrium data of Baur and Glaessner.

Reaction 1.		Reaction 2.	
Temperature of solid phase.	$\frac{pCO}{pCO_2}$	Temperature of solid phase.	$\frac{pCO}{pCO_2}$
° C.		° C.	
552	1.16	450	0.812
596	1.25	490	0.887
651	1.38	550	0.786
662	1.40	650	0.587
680	1.44	850	0.351
750	1.56	950	0.299
850	2.12		
900	2.51		

The equilibrium constants shown above are plotted in figure 19. Assuming that Fe_2O_3 reacts like Fe_3O_4 as regards reduction, it is evident that at temperatures up to 1,000° C. no reduction to metallic iron nor oxidation to Fe_3O_4 can take place in a mixture of equal parts by volume of CO and CO_2; and unless the direction of the two curves changes rather suddenly, FeO should continue to be the stable phase at higher temperatures, probably throughout the usual softening range of coal ashes. Although these conditions of equilibrium do not necessarily apply in the presence of other components as in a coal-ash fusion, they at least point out the possibility of a minimum softening point due to ferrous iron in mixtures of approximately equal parts by volume of CO and CO_2, provided this mixture is passed over the heated ash continuously, sweeping the gaseous products of the reaction out of the apparatus.

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[a] Kreisinger, Henry, Ovitz, F. K., and Augustine, C. E., Combustion in the fuel bed of hand-fired furnaces: Tech. Paper 137, Bureau of Mines, 1916, p. 39.

[b] Kreman, R. (translation by Potts, H. E.), The application of physico-chemical theory to technical processes and manufacturing methods: London, 1913, pp. 117–119.

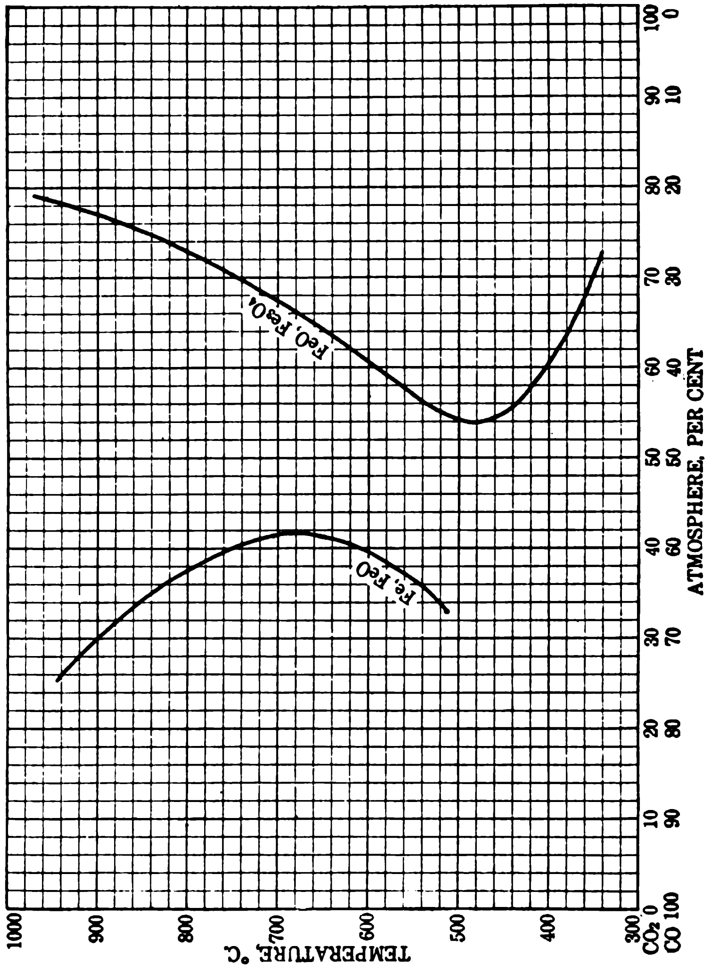

FIGURE 19.—Equilibrium diagram of the system CO, CO_2, Fe, FeO, and Fe_3O_4.

METHOD OF CONDUCTING EXPERIMENTS.

Softening-temperature determinations were made in various mixtures of CO and CO_2 with ash from the same samples that were used in the series in which atmospheres of hydrogen and water vapor were used, except sample 15,845, which had been entirely used. The same type of furnace and of silica-tube arrangement was used, the hydrogen tank and saturator being replaced with a large gasometer which contained the desired mixture of CO and CO_2. A gasometer similar to one described by Dennis[a] was found especially advantageous for this work in that a large volume of gas could be made and stored without much change in composition taking place. Instead of the usual arrangement of having the gas confined over a considerable surface of liquid, the bell was arranged to move freely up and down between two concentric cylinders; the inner one was closed at the top, thus limiting the liquid seal to the 1-inch annular space between. By using a mixture of approximately 60 per cent glycerin and 40 per cent water the solubility of CO_2 was further reduced to a minimum.

PREPARATION OF GAS MIXTURES.

The mixtures containing equal parts of CO and CO_2 were made by heating 250 grams of oxalic acid and 600 c. c. of concentrated sulphuric acid in a 1,000-c.c. side-tube distilling flask. From the flask the gas passed through a wash bottle containing concentrated sulphuric acid and then directly into the gasometer, in which a slightly diminished pressure was maintained. Air in the gasometer and connections was first removed by partly filling and then emptying the tank before the gas was stored. Mixtures containing more than 50 per cent CO_2 were made by adding more CO_2 from a tank of liquid carbon dioxide. The carbon dioxide from this latter source showed the following analysis: Carbon dioxide, 99.9 per cent; air, 0.1 per cent.

Mixtures containing less than 50 per cent CO_2 were made by interposing between the oxalic acid generator and the gasometer an absorbing train of sodium hydroxide solution for removing CO_2. CO_2 from the tank containing liquid CO_2 was then added to the CO in the gasometer. When the gasometer had been filled with the desired gas mixture the mixture was allowed to stand overnight to come to equilibrium. Samples were then taken over mercury and analyzed for carbon monoxide, carbon dioxide, and oxygen according to the Hempel method, as described in Technical Paper 63 of the Bureau of Mines.[b] Tests were made which assured the absence of any appreciable quantities of other combustible gases. In order to check any changes in the composition of the gas stored in the gasometer a sample was taken each day until the supply became exhausted. The change in composition from day to day never exceeded a few tenths of a per cent. Any small fractional percentage of oxygen found in the gas was calculated to CO_2, on the assumption that in the furnace it combined with CO, an equivalent deduction being made in the CO percentage.

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[a] Dennis, L. M., Gas analysis, 1913, pp. 25–27.

[b] Clement, J. K., Frazer, J. C. W., and Augustine, C. E., Factors governing the combustion of coal in boiler furnaces: Tech. Paper 63, Bureau of Mines, 1914, pp. 13–17, figs. 6 and 7.

RESULTS OF EXPERIMENTS.

The general procedure followed in making the first series of softening-temperature tests in CO and CO_2 was identical with that used in the tests with hydrogen and water vapor, as described on pages 57 to 60. Heating with a temperature rise of 5° C. per minute until initial deformation, and then with a temperature rise of 2° C. per minute until the cone was down gave concordant results in the hydrogen atmosphere. The same rate of heating was therefore used in the first series in which carbon monoxide was used. However, as the work proceeded it soon became apparent that wholly concordant results were not being realized; as all previous investigations on the effect of rate of heating had been made in reference to atmospheres of air or of hydrogen and water vapor, it seemed possible that a slower rate of heating might be necessary to attain uniform results in $CO-CO_2$ atmospheres. The softening temperature of ash sample 13629 was therefore determined at three different rates of heating in a mixture of approximately equal parts of CO and CO_2. As shown in the following table, when the rate of heat increase was 2° C. per minute, the ash softened at a temperature 72° C. lower than when the heat increase was 5° C. per minute.

Results of experiments to determine effect of rate of heating on softening temperature of ash.

Rate of temperature increase.	Softening temperature.	
	Initial.	Final.
	° C.	° C.
5° C. per minute	1,072	1,176
5° C. to initial deformation, then 2° C. to final deformation	1,048	1,152
2° C. per minute	1,022	1,104

In view of this seemingly greater lag of the reducing and softening reactions in carbon monoxide a second series of tests was made on the same ashes at a much slower rate of heating. In this series the ash was heated rapidly to a temperature of 950° C. and held at that temperature for a period of one hour; heating was then continued at a 2° C. rate until the cone was down.

The results obtained in the two series are given in Table 10 following, and in the curves of figures 20 to 23; the previously obtained curves from experiments with mixtures of hydrogen and water vapor are also plotted to the same scale to permit comparison. The results of analyses of the ashes follow:

Results of analyses of ashes represented in figures 20 to 23 and in Table 10.

Ash sample No.	SiO_2.	Al_2O_3.	Fe_2O_3.	TiO_2.	CaO.	MgO.	Na_2O.	K_2O.	SO_3.
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
16	37.2	25.5	11.8	1.5	12.6	1.9	1.4	0.3	5.6
5	42.2	30.6	19.0	1.2	1.3	1.0	1.3	2.9	.3
12	37.1	17.6	25.9	.7	3.2	.9	.4	1.8	2.3
9	54.8	27.0	7.0	1.3	4.3	1.7	.3	3.1	1.4

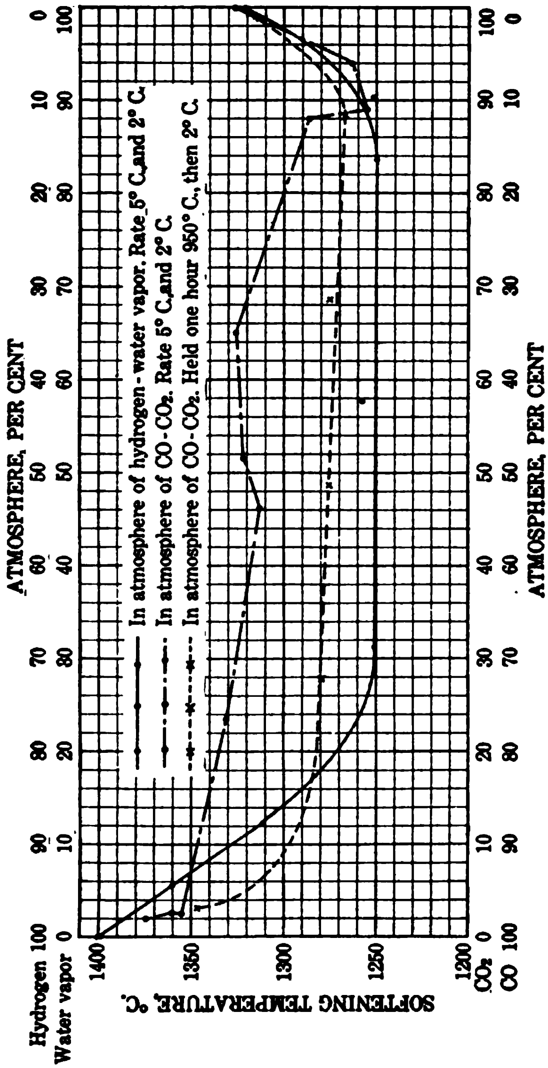

FIGURE 20.—Curves showing softening temperatures of ash sample 16 in mixtures of CO and CO_2 and of H_2 and H_2O.

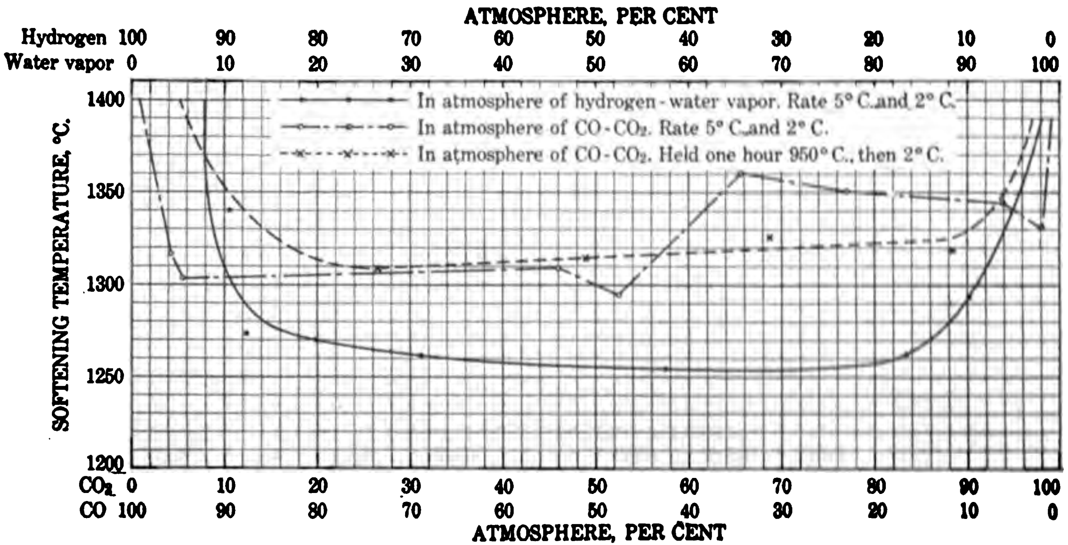

FIGURE 21.—Curves showing softening temperatures of ash sample 5 in mixtures of CO and CO_2 and of H_2 and H_2O.

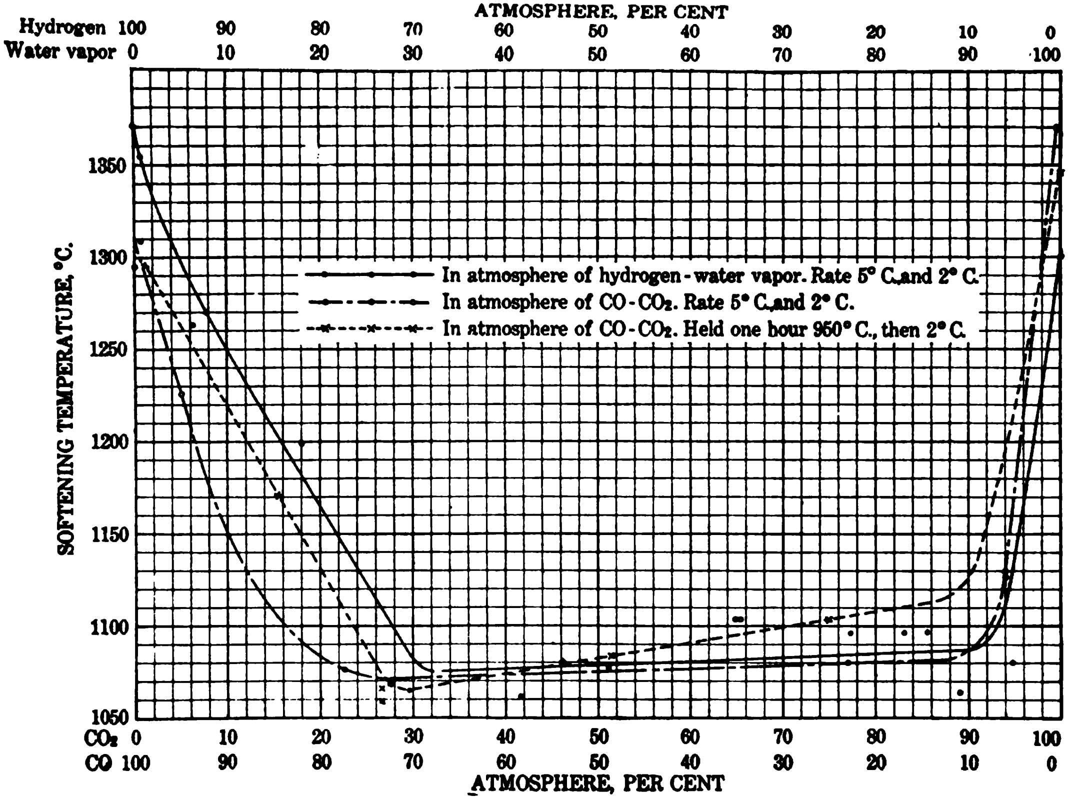

FIGURE 22.—Curves showing softening temperatures of ash sample 12 in mixtures of CO and CO_2 and of H_2 and H_2O.

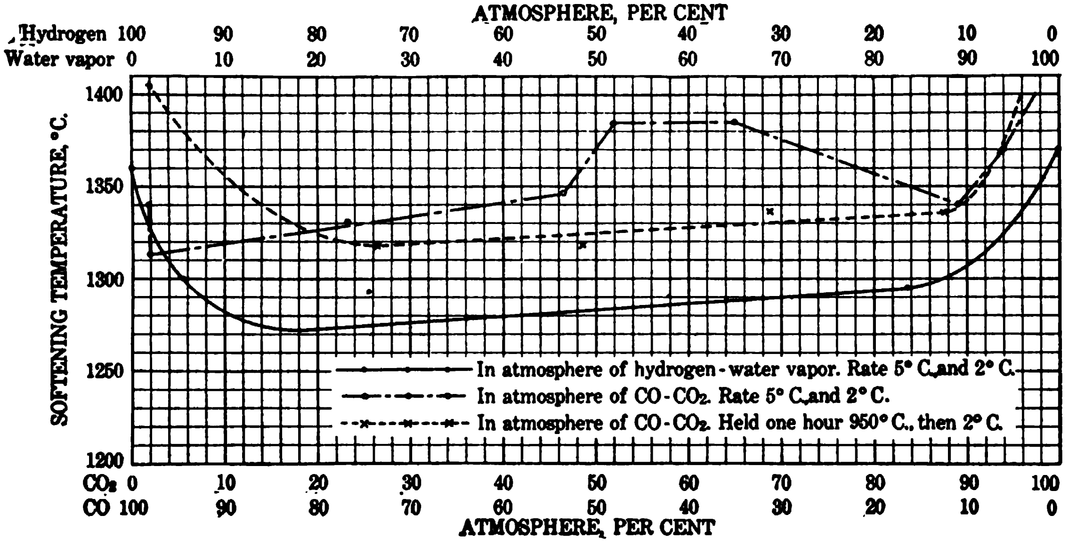

FIGURE 23.—Curves showing softening temperatures of ash sample 9 in mixtures of CO and CO_2 and of H_2 and H_2O.

TABLE 10.—*Results of tests to determine softening temperatures of certain ash samples in an atmosphere containing equal parts of CO and CO_2.*

ASH SAMPLE 9.

Composition of gas.		Softening temperature at heating rate of 5° and 2° C.[a]	Softening temperature with temperature of 900° to 950° C. for 1 hour, followed by rise of 2° C. per minute.	Magnetic properties and amount of metallic iron in fused cone.
CO.	CO_2.			
P. ct.	*P. ct.*	°C.	*° C.*	
98.2	1.8	1,227–1,331 1,276–1,340		Fairly magnetic; many small globules metallic Fe. Faintly magnetic; many small globules metallic Fe.
98.1	1.9	1,227–1,313		Faintly magnetic; a few globules of metallic Fe.
98.0	2.0		1,271–1,405	Fairly magnetic; many small globules metallic Fe.
76.7	23.3	1,227–1,331		Faintly magnetic; a few globules metallic Fe.
73.6	26.4		1,136–1,318	Faintly magnetic; no metallic Fe.
53.5	46.5	1,318–1,347		Do.
51.7	48.3		[a]1,237–1,318	Nonmagnetic; no metallic Fe.
48.1	51.9	1,313–1,385		Do.
35.2	64.8	1,350–1,385		Do.
31.3	68.7		1,175–1,337	Do.
12.7	87.3		1,275–1,337	Do.
11.1	88.9	1,271–1,340		Do.
6.0	94.0	1,304–1,370		Do.
.9	99.1		Above 1,400	Faintly magnetic; no metallic Fe.
.3	99.7	[b]Above 1,400		Fairly magnetic; no metallic Fe.

[a] A cone heated for 2 hours at 900° to 950° C. and then subjected to a temperature rise of 2° C. per minute thereafter started to bend at 1,174° C. and was down at 1,314° C.

[b] Just started.

ASH SAMPLE 16.

98.1	1.9	1,327–1,375		Strongly magnetic; many globules metallic Fe.
97.5	2.5	1,313–1,355 1,322–1,360		Fairly magnetic; many globules metallic Fe. Do.
97.0	3.0		1,318–1,347	Strongly magnetic; many globules metallic Fe.
76.7	23.3	1,252–1,331		Fairly magnetic; many globules metallic Fe.
72.3	27.7		1,167–1,279	Fairly magnetic; a few globules metallic Fe.
53.9	46.1	1,275–1,313		Faintly magnetic; a few globules metallic Fe.
51.7	48.3		[c]1,149–1,275	Faintly magnetic; very few globules metallic Fe.
48.4	51.6	1,300–1,322		Faintly magnetic; no metallic Fe.
35.2	64.8	1,300–1,326		Do.
31.5	68.5		1,133–1,275	Do.
12.0	88.0	1,174–1,287		Lost.
11.7	88.3		1,237–1,267	Nonmagnetic; no metallic Fe.
11.1	88.9	1,248–1,255		Do.
6.0	94.0	1,227–1,263		Fairly magnetic; no metallic Fe.
.9	99.1		1,233–1,314	Strongly magnetic; no metallic Fe.
.0	100.0	1,275–1,327		Do.

[c] A cone heated 2 hours at 900° to 950° C. and then subjected to a temperature rise of 2° C. per minute thereafter started to bend at 1,184° C. and was down at 1,275° C.

ASH SAMPLE 5.

98.2	1.8		1,251–[d]1,400	Fairly magnetic; many globules metallic Fe.
98.1	1.9	(e)		Strongly magnetic; many globules metallic Fe.
95.8	4.2	1,199–1,317		Do.
94.8	5.2	1,176–1,304		Fairly magnetic; many globules metallic Fe.
73.6	26.4		1,220–1,309	Faintly magnetic; no metallic Fe.
54.0	46.0	1,300–1,309		Nonmagnetic; no metallic Fe.
51.1	48.9		1,080–1,314	Do.
47.5	52.5	1,248–1,295		Do.
35.2	64.8	1,160–1,360		Faintly magnetic; no metallic Fe.
31.3	68.7		1,267–1,327	Do.
23.1	76.9	1,322–1,350		Nonmagnetic; no metallic Fe.
11.7	88.3		1,122–1,318	Fairly magnetic; no metallic Fe.
6.1	93.9	1,127–1,345		Faintly magnetic; no metallic Fe.
2.1	97.9	1,142–1,331		Lost.
.9	99.1		Above 1,400	Strongly magnetic; no metallic Fe.
.0	100.0	Above 1,400		Do.

[d] Above.

[e] Cone started softening at 1,243° C.; one-half melted down at 1,400° C.

TABLE 10.—*Results of tests to determine softening temperatures of certain ash samples in an atmosphere containing equal parts of CO and CO_2*—Continued.

ASH SAMPLE 12.

Composition of gas.		Softening temperature at heating rate of 5° and 2° C.[a]	Softening temperature with temperature of 900° to 950° C. for 1 hour, followed by rise of 2° C. per minute.	Magnetic properties and amount of metallic iron in fused cone.
CO.	CO_2.			
P. ct.	*P. ct.*	° C.	*° C.*	
100.0	0.0	1,270–1,295	...	Strongly magnetic; much metallic Fe.
99.2	.8	1,241–1,309	...	Do.
99.0	1.0	...	1,251–1,296	Strongly magnetic; many globules metallic Fe.
95.0	5.0	1,170–1,226	...	Strongly magnetic; much metallic Fe.
84.7	15.3	...	1,101–1,171	Fairly magnetic; many globules metallic Fe.
77.3	22.7	[f] 1,076	...	Faintly magnetic; no metallic Fe.
73.4	26.6	...	[a] 1,058	Nonmagnetic; No metallic Fe.
73.4	26.6	...	1,055–1,066	Do.
53.9	46.1	1,050–1,080	...	Nonmagnetic; no metallic Fe.
48.5	51.5	...	1,073–1,083	Do.
47.7	52.3	1,027–1,048	...	Faintly magnetic; no metallic Fe.
35.2	64.8	1,072–1,104	...	Nonmagnetic; no metallic Fe.
25.1	74.9	...	1,073–1,104	Do.
23.1	76.9	[f] 1,104	...	Lost.
		[f] 1,080	...	Nonmagnetic; no metallic Fe.
11.1	88.9	1,048–1,064	...	Fairly magnetic; no metallic Fe.
6.0	94.0	1,060–1,130	...	Do.
		1,048–1,127	...	Do.
.3	99.7	1,304–1,370	...	Strongly magnetic; no metallic Fe.
.0	100.0	...	1,300–1,347	Do.

[a] Starting point not observed.

DISCUSSION OF RESULTS.

The softening-point curves show that, except ash sample 16243, smooth curves were not obtained for the softening points in atmospheres of CO and CO_2 when the rate of increase of heating was 5° and 2° C. per minute; however, on allowing more time for reduction and softening of the ash by holding the cone at a temperature of 900° to 950° for a period of one hour and then increasing the heat 2° C. per minute until softening was complete, the seeming irregularities disappeared and curves similar in shape to those obtained in atmospheres of hydrogen and water vapor were obtained. These lower and more uniform curves show for each of the ashes tested a well-defined region of minimum softening temperature, which ranged between approximate limits of 75 and 10 per cent carbon monoxide. For most samples the minimum softening temperature in an atmosphere of carbon monoxide and carbon dioxide was somewhat higher than the softening temperature in an atmosphere of hydrogen and water vapor, the greatest difference being 65° C. for sample 15844.

The general similarity in shape of the H_2–H_2O and the CO–CO_2 curves would indicate that the variation in the fusibility of a given ash is determined principally by the state of oxidation of its iron content. The stable phase of iron is in turn determined by the temperature, by the relative concentrations of oxidizing and reducing gases, and, in part, by the other components of the ash. In a general way the effect as regards reduction would be similar for carbon monoxide and hydrogen; likewise the oxidizing effects of air, of carbon dioxide, and of water vapor on the iron component are similar enough to give maximum softening temperatures of the same order for a given ash, as shown in Table 11 following:

TABLE 11.—*Results of tests to determine comparative effects on softening temperatures of ash of atmospheres of air, of water vapor, of carbon dioxide, of carbon monoxide, of equal parts of hydrogen and water vapor, and of equal parts of carbon monoxide and carbon dioxide.*

Ash sample No.	Per cent of Fe_2O_3.	Softening temperature, ° C.						
		Oxidizing atmosphere.			Reducing atmosphere.		Atmosphere of equal parts of H_2 and H_2O.	Atmosphere of equal parts of CO and CO_2.
		Air.	Water vapor.	CO_2.	H_2.	CO.		
16243	35.9	1,335	1,313	1,347	1,370	1,295	1,080	1,085
15844	19.0	1,490	(a)	(a)	(a)	(a)	1,255	1,315
16586	11.8	1,322	1,322	1,327	1,402	1,375	1,250	1,275
15848	7.0	1,425	1,370	(a)	1,360	1,331	1,283	1,325

a Above 1,400.

The sharp rise of the softening-point curves at both the oxidizing and reducing end confirms the reasons previously advanced for the large variations in results obtained in different furnaces. It is evident that no concordant results can be obtained in oxidizing atmospheres containing small proportions of reducing gas or similarly in essentially reducing atmospheres with small admixtures of oxidizing gases, as in these regions the curves have a sharp inclination.

RELATION OF SOFTENING TEMPERATURES IN ATMOSPHERES OF HYDROGEN AND WATER VAPOR, AND OF CARBON MONOXIDE AND CARBON DIOXIDE TO CLINKER FORMATION IN FURNACES.

The general form of the softening-point curves suggests a choice of three types of atmosphere in which uniform softening points may be determined, namely, (1) a strongly reducing atmosphere of 100 per cent hydrogen or carbon monoxide; (2) a weakly reducing atmosphere of approximately 50 per cent hydrogen and 50 per cent water vapor, or of 50 per cent carbon monoxide and 50 per cent carbon dioxide; (3) an oxidizing atmosphere of 100 per cent water vapor, carbon dioxide, or air.

The atmosphere to be selected for a standard softening-temperature test should be one that produces a slag with the iron in the same state of oxidation as is usually found in fuel-bed clinkers. Determinations were therefore made of the relative percentages of ferrous, ferric, and metallic iron in the glassy parts of furnace clinkers and in the ash cones fused in the atmospheres mentioned above.

FERROUS, FERRIC, AND METALLIC IRON IN FUSED ASH CONES AND CLINKER SLAGS.

Fourteen to twenty ash cones, whose weight aggregated 2 or 3 grams, were placed in a platinum boat and heated to softening temperatures in the silica tube of the furnace at the different rates of heating used in the previous tests. In the tests with an atmosphere of hydrogen and water vapor the boat and its contents were held at this temperature for 30 minutes to permit the softening reactions to approach equilibrium. In the tests with an atmosphere of CO and CO_2 heating was stopped as soon as the softening point was reached. As in the softening-temperature determinations the cones were heated to redness previous to the tests to oxidize and remove the organic binders. At the end of the heating period the electric current was cut off and the furnace allowed to cool rapidly, without interrupting the gas stream. To avoid any possibility of change during cooling it would have been better to have quenched the fusion suddenly. However, the viscosity of the ash melted was so great that little change was probable at the rate of cooling used. The fused material appeared glassy.

As the particular ash samples used in the softening-temperature tests and in the fusions with hydrogen and water vapor became exhausted before the fusions in atmospheres of CO and CO_2 were made, it was necessary to use for the latter fusion tests other samples of ash from coal from the same mines. The new samples showed a slightly different total iron content and a softening temperature that deviated approximately 40° C. from that of the original samples. However, they served as well for use in determining the relative reducing effect of the different atmospheres on the ferric oxide.

After the furnace had cooled, the fused cones were removed from the boat and were pulverized, first by impact in a diamond mortar of hardened steel and then in an agate mortar, to 100 mesh and analyzed for ferrous, ferric, and metallic iron by the method of Mathesius.[a] The results, as given in Table 12 following, show well-defined differences in the state of oxidation of the iron content of the fused ash samples in the various atmospheres. In air, water vapor, and carbon dioxide 67 to 88 per cent of the ferric oxide remained as ferric iron; in hydrogen and in carbon monoxide, 49 to 78 per cent of the iron was reduced to the metallic form, the larger percentage being found in the cones fused in hydrogen atmosphere; in the cones fused in an atmosphere of equal parts of hydrogen and water and in the cones fused in an atmosphere of equal parts of CO and CO_2 little or no metallic iron was found, the dominant form being ferrous iron, the proportion of which ranged from 78 to 94 per cent.

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[a] Mathesius, Ludwig, Untersuchungen über die Reduzierbarkeit von Eisenerzen in strömenden Gasen: Stahl und Eisen, Bd. 34, 1914, pp. 866–867.

TABLE 12.—*Results of analyses of ash cones fused in various atmospheres and of furnace clinkers.*

Sample No.	Description of sample.	Atmosphere.	Rate of temperature rise per minute.	Maximum temperature of fusion, °C.	Total Fe.	Percentage of total Fe occurring as—			Character of slag.
						Metallic iron.	Ferrous iron.	Ferric iron.	
GROUP I. FERRIC IRON PREDOMINATING.									
20155.	Ash	CO_2	5° and 2° C.[a]	1,405	*P. ct.* 6.7	None.	12	88	Glassy, black; slightly magnetic.
20155.	do	do	(b)	1,405	6.8	None.	24	76	Glassy, black; nonmagnetic.
20242.	do	do	5° and 2° C.[a]	1,368	21.4	None.	28	72	Black, glassy; highly magnetic.
20242.	do	do	(b)	1,368	21.3	None.	33	67	Do.
16243.	do	H_2O vapor.	5° and 2° C.[a]	1,300	27.9	Trace.	28	72	Glassy, black, highly magnetic.
13629.	do	Air	do	1,400	15.5	Trace.	30	70	Dark red to almost black, glassy; highly magnetic.
GROUP II. METALLIC IRON PREDOMINATING.									
20155.	Ash	$96CO;4CO_2$	5° and 2° C.[a]	1,337	6.6	49	41	10	Dark gray; magnetic.
20155.	do	do	(b)	1,405	6.2	55	32	13	Do.
20242	do	$99CO;1CO_2$	5° and 2° C.[a]	1,309	20.7	60	36	4	Dark gray; highly magnetic.
20242	do	$97CO;3CO_2$	(b)	1,309	21.9	62	38	Trace	Do.
16243.	do	H_2	5° and 2° C.[a]	1,370	28.6	78	21	1	Black; metallic luster.
GROUP III. FERROUS IRON PREDOMINATING.									
20155	Ash	$48CO;52CO_2$	5° and 2° C.[a]	1,368	6.6	None.	81	19	Dark gray; slightly magnetic.
20155	do	do	(b)	1,405	5.7	None.	78	22	Do.
20242.	do	$47\ CO;\ 53\ CO_2$.	5° and 2° C.[a]	1,080	20.9	4	88	8	Dark, glassy; slightly magnetic.
20242.	do	do	(b)	1,080	20.9	6	79	15	Do.
20242.	do	$48\ CO;\ 52\ CO_2$.	5° and 2° C.[a]	1,368	20.4	7	87	6	Do.
20242.	do	do	(b)	1,370	19.8	8	84	8	Do.
16243.	do	$50\ H_2;\ 50\ H_2O$.	5° and 2° C.[a]	1,080	27.4	Trace.	82	18	Black, glassy; nonmagnetic.
15848.	do	do	do	1,280	4.7	6	94	Trace	Gray, glassy; nonmagnetic.
20137.	Clinker from a hand-fired furnace.				14.5	12	79	9	Dark, glassy; contained some magnetic pieces.
20145.	Clinker from boiler furnace.				12.7	6	75	19	Brown and black, glassy; contained some magnetic pieces.
20452.	do				8.9	9	75	16	Do.

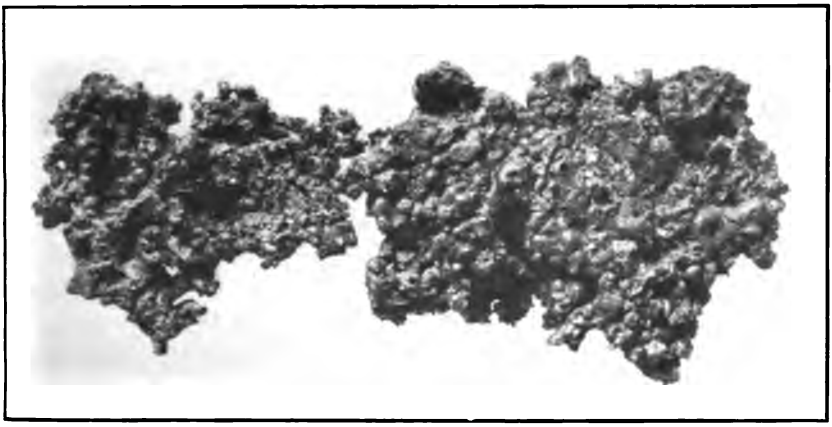
[a] Cone heated with a temperature rise of 5° C. per minute until initial deformation when the rate of increase was 2° C. per minute until final deformation.

[b] Cone heated for 1 hour at 950° C., when it was subjected to a temperature rise of 2° C. per hour.

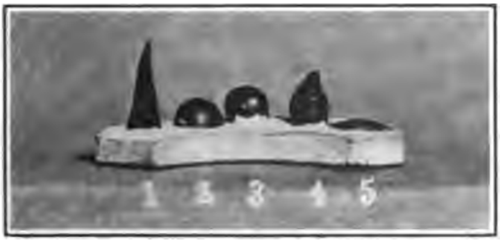
Together with the results of analyses of fused ash cones are given the results of analyses of the fused and glassy parts of several clinkers produced under working conditions in the fuel beds of different boiler furnaces operating with Pittsburgh coal. The coal, however, came from different mines. Clinker 20137 was obtained from the grate of a small hand-fired experimental furnace used for combustion investigations. Clinkers 20145 and 20452 were kindly furnished by Mr. A. A. Straub, steam engineer of the Duquesne Light Co., of Pittsburgh, Pa. These clinkers were dumped from the fuel bed while hot and were quenched with water. The clinkers were of the particularly troublesome taffylike variety that flow over the grate bars and shut off the air supply for combustion. Plate III, *A*, represents a photograph of clinker 20452. The edge at the top of the plate is the side that adhered to the corrugated grate of the new model type D Roney stoker. The matrix of this clinker was a black glass which could readily be chipped out in quantity for analysis without including any coke or unfused material. At the contact surface of coke and slag, particles of metallic iron could be identified. A microscopic examination, made by Mr. A. A. Klein, of the Bureau of Standards, of the powdered slag from this clinker and from the others showed the material to be essentially a glass with small quantities of sillimanite (Al_2O_3,SiO_2) and of an opaque iron mineral, probably magnetite, as the only crystalline phases present.

The results of analyses of the fused ash cones, and of the three fuel-bed clinkers given in Table 12 show that of the different atmospheres employed in the laboratory furnace, the atmosphere containing equal parts of hydrogen and water vapor and that containing equal parts of carbon monoxide and carbon dioxide produced slags that corresponded most closely in the state of oxidation of their iron content to the glassy parts of the furnace clinkers. In both the fused cones and the clinkers approximately 80 per cent of the iron appeared as ferrous iron, the form in which it imparts the maximum fluxing action on the silicate mixture.

In view of the results presented above, the conclusion is warranted that the minimum softening temperature of a coal ash as determined in either the $CO-CO_2$ or the H_2-H_2O atmosphere is more representative of the temperature of clinker formation under furnace conditions than the somewhat higher results obtained in strongly reducing atmospheres of hydrogen or carbon monoxide, on the one hand, and oxidizing atmospheres of air, water vapor, or carbon dioxide, on the other.



A. CLINKER NO. 20452 FROM GRATE OF RONEY STOKER.



B. TYPICAL FORMS OF CONES FUSED IN THE NO. 3 MELTER'S FURNACE.

RELATIVE MERITS OF CO–CO_2 AND OF H_2–H_2O ATMOSPHERES AS STANDARD ATMOSPHERES FOR FUSION TESTS.

The results of the experiments with the four ashes that have been investigated indicate that the effects of hydrogen and of carbon monoxide as a reducing agent are closely similar. The principal effect of either gas mixture is to cause the iron component to appear in the slag in the ferrous state. There are, however, certain minor reactions involving the formation of sulphides and carbides that proceed differently in the two atmospheres. For example, all the ash cones fused in an atmosphere containing equal parts of CO and CO_2 and in an atmosphere of 100 per cent CO, contained ferrous sulphide and some of the fused cones contained carbides; the sulphides were formed by reduction of sulphates in the ash. In the fusions made in an H_2–H_2O atmosphere special evidence of sulphide formations was not noted. Obviously carbides could not be formed in the latter atmosphere. These minor reactions and perhaps a slower reaction velocity of reduction may be the cause of the somewhat higher softening temperatures obtained in the CO–CO_2 mixtures.

In order to obtain further data on the relation between the minimum softening temperatures of various ashes, as determined in an atmosphere containing equal parts of H_2 and H_2O, in an atmosphere containing equal parts of CO and CO_2, and in air, many samples of coal ashes having various contents of iron oxide were tested in these atmospheres.

The results are given in Table 13, following, and are plotted in the order of increasing iron oxide content in figures 24 and 25. Table 13 and figure 24 also present a series of results obtained in gas furnace No. 3. These values are discussed in a subsequent part of this report.

TABLE 13.—*Results of tests to determine softening temperatures of coal ashes containing various percentages of ferric oxide in various atmospheres.*

[Each cone ⅜ by ¼ inch, mounted in a vertical position in quartz tube in furnace shown in figure 17, except certain tests in a gas furnace; rate of increase of heating for tests in atmosphere of equal parts of H_2 and H_2O, 5° C. per minute until initial deformation, then 2° C. per minute; for tests in atmosphere of equal parts of CO and CO_2 rate of increase of heating was 2° C. per minute.]

Ash sample No.	Fe_2O_3 in ash.	Softening temperature in atmosphere of:				Differences.				Softening interval.				Magnetism of cones fused in atmosphere of equal parts of CO and CO_2.
		Equal parts of H_2 and H_2O.	Equal parts of CO and CO_2.	Gas furnace.	Air.	Atmosphere of H_2 and H_2O and atmosphere of CO and CO_2.	Atmosphere of H_2 and H_2O and air.	Atmosphere of CO and CO_2.	Atmosphere of gas furnace and atmosphere of CO and CO_2.	H_2–H_2O atmosphere.	CO–CO_2 atmosphere.	Atmosphere of gas furnace.	Air.	
	P. ct.	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	
96	5.0	1,263	1,254	1,266	1,278	− 9	15	24	− 12	36	4	5	5	Not magnetic.
92	5.9	1,345	1,405	1,361	1,463	60	118	58	+ 44	125	134	21	8	Do.
97	6.5	1,313	1,292	1,236	1,345	− 21	32	53	+ 56	42	70	36	15	Slightly magnetic.
75	6.8	[a]1,327	1,405	1,469	1,520	78	193	115	− 64	36	122	38	64	Not magnetic.
9	7.0	1,271	1,330	1,369	1,425	59	154	95	− 39	92	177	54	55	Do.
77	7.0	1,300	1,360	1,401	1,470	60	170	110	− 41	77	89	113	60	Do.
95	7.4	1,227	1,288	1,325	1,370	61	143	82	− 37	54	107	148	20	Do.
13	7.8	1,209	1,345	1,262	1,500	136	291	155	+ 83	51	181	158	60	Do.
85	8.0	[a]1,228	1,233	1,240	1,320	5	92	87	− 7	37	83	84	20	Do.
73	8.2	1,255	1,405	1,358	1,450	150	195	45	+ 47	49	214	176	120	Do.
76	8.4	1,205	1,261	1,250	1,400	56	195	139	+ 11	56	91	21	120	Do.
71	9.0	1,179	1,320	1,298	1,465	141	286	145	+ 22	75	193	90	65	Do.
18	9.0	1,187	1,335	1,277	1,400	148	213	65	+ 58	52	186	121	60	Do.
74	9.2	1,227	1,370	1,327	1,405	143	178	35	+ 43	35	230	122	75	Do.
15	9.4	1,216	1,325	1,310	1,407	109	191	82	+ 15	31	134	138	67	Do.
78	9.5	1,212	1,271	1,282	1,405	59	193	134	− 11	38	149	48	55	Do.
69	9.5	1,161	1,197	1,166	1,350	36	189	153	+ 31	6	85	36	98	Do.
72	9.8	1,173	1,370	1,304	1,426	197	253	56	+ 66	69	243	75	96	Do.
17	10.1	[a]1,198	1,201	1,218	1,343	3	145	142	− 17	41	27	62	63	Do.
64	10.3	[a]1,291	1,310	1,240	1,474	19	183	164	+ 70	100	120	32	65	Do.
68	10.5	1,173	1,201	1,229	1,350	28	177	149	− 28	25	51	73	97	Do.
99	11.3	[b]1,201	1,215	1,197	1,299	14	98	84	+ 18	10	98	41	10	Do.
16	11.8	[a]1,251	1,261	1,261	1,322	10	71	61	0	24	49	53	82	Do.
82	11.9	1,181	1,310	1,325	1,456	129	275	146	− 15	63	177	64	66	Do.
79	12.4	1,149	1,310	1,336	1,414	161	265	104	− 26	48	170	21	46	Do.
89	13.3	[a]1,155	1,292	1,325	1,369	137	214	77	− 33	75	222	143	12	Do.
65	13.6	[a]1,165	1,190	1,166	1,409	25	244	219	+ 24	24	50	86	27	Do.
81	15.0	1,143	1,271	1,304	1,418	128	275	147	− 33	42	131	96	38	Do.
90	15.4	[a]1,170	1,212	1,177	1,324	42	154	112	+ 35	16	42	47	21	Do.
70	16.6	1,148	1,335	1,176	1,410	187	262	75	+159	44	182	31	28	Do.

8..............	18.0	1,205	1,365	1,320	1,450	160	245	85	+ 45	41	273	5	10	Slightly magnetic.
89..............	18.1	1,167	1,320	1,226	1,430	153	263	110	+ 94	66	170	32	30	Do.
67..............	18.3	[a]1,185	1,360	1,316	1,450	175	265	90	+ 44	50	258	69	10	Not magnetic.
100..............	18.9	[b]1,196	1,212	1,172	1,377	16	181	165	+ 40	31	142	26	20	Do.
5..............	19.0	1,255	1,314	1,340	1,490	59	235	176	− 26	82	181	54	50	Do.
66..............	19.5	[a]1,188	1,212	1,218	1,340	24	152	128	− 6	28	22	62	10	Do.
83..............	20.0	[a]1,116	1,190	1,145	1,368	74	252	178	+ 45	36	140	41	81	Slightly magnetic.
11..............	20.4	[a]1,132	1,143	1,156	1,400	11	268	257	− 13	47	62	26	20	Not magnetic.
10..............	22.4	[a]1,137	1,153	1,187	1,310	16	173	157	− 34	10	45	42	40	Do.
98..............	23.5	1,105	1,104	1,166	1,322	− 1	217	218	− 62	55	82	87	13	Do.
93..............	24.9	1,127	1,170	1,198	1,370	43	243	200	− 28	23	78	84	20	Do.
84..............	28.8	1,108	1,092	1,120	1,276	− 16	168	184	− 28	28	7	41	26	Do.
94..............	29.7	1,185	1,268	1,272	1,345	83	160	77	− 4	4	87	64	70	Slightly magnetic.
87..............	31.5	[a]1,119	1,092	1,116	1,305	− 27	186	213	− 24	33	38	12	26	Not magnetic.
91..............	34.6	1,263	1,292	1,341	1,497	29	234	205	− 49	90	133	211	97	Slightly magnetic.
12..............	35.9	[a]1,060	1,080	1,064	1,335	20	275	255	+ 16	12	80	10	65	Do.
86..............	41.9	1,071	1,077	1,076	1,358	6	287	281	+ 1	6	51	27	37	Not magnetic.
88..............	61.7	[a]1,113	1,130	1,125	1,452	17	339	322	+ 5	0	18	15	152	Slightly magnetic.
Average...	15.9	1,193	1,259	1,249	1,393	66	200	134	10	44	117	63	49	

[a] Cone $\frac{1}{16}$ by 1 inch, mounted vertically in molybdenum furnace shown in figure 29.
[b] Atmosphere of 60 per cent H_2 and 40 per cent H_2O used instead of 50 per cent of each.

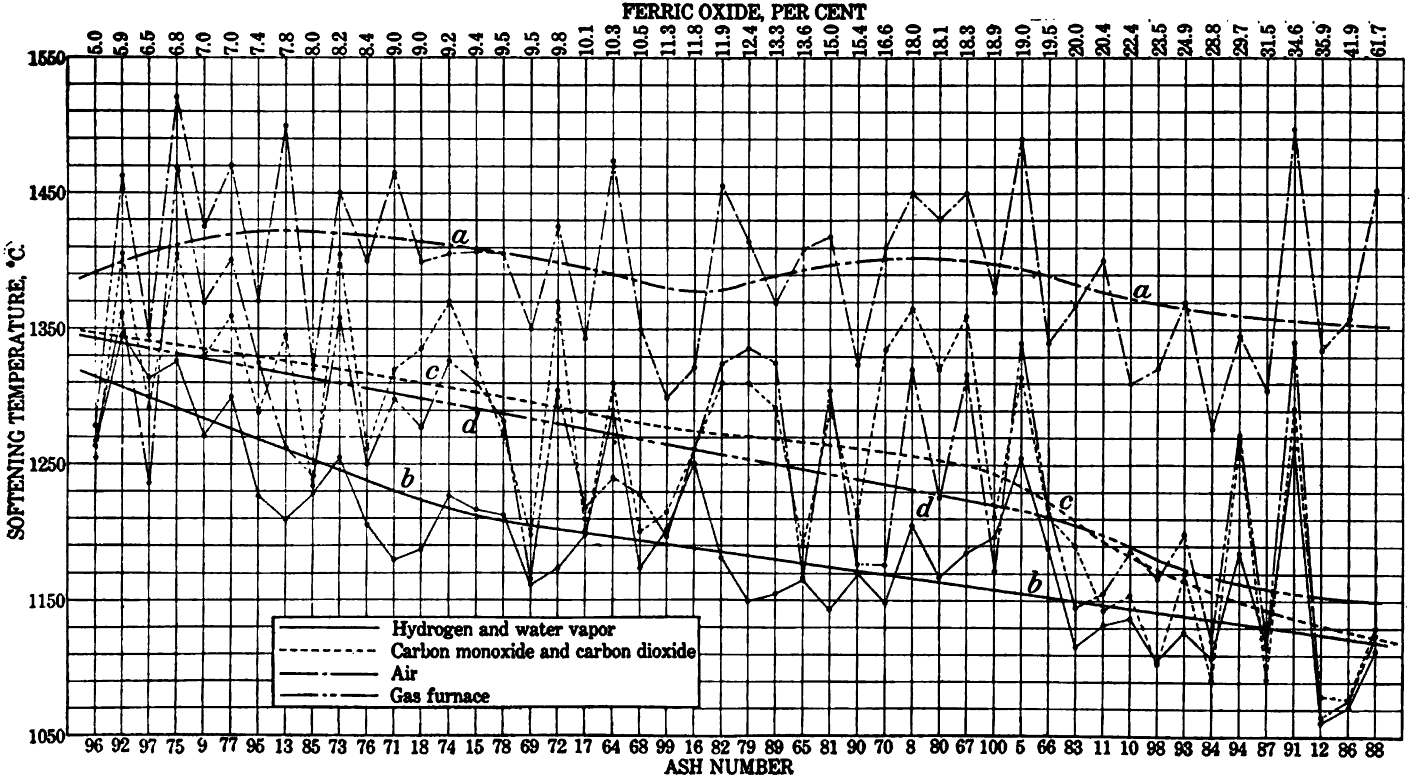


FIGURE 24.—Curves showing softening temperatures of coal ash in atmospheres of equal parts of H_2 and H_2O, equal parts of CO and CO_2, gas furnace No. 8, and air, with especial reference to the percentage of Fe_2O_3 in the ash. *a*, Average curve showing relation between softening temperature in air and percentage of Fe_2O_3 in ash; *b*, average curve showing relation between softening temperature in atmosphere of equal parts of H_2 and H_2O and percentage of Fe_2O_3 in ash; *c*, average curve showing relation between softening temperature in atmosphere of equal parts of CO and CO_2 and percentage of Fe_2O_3 in ash; *d*, average curve showing relation between softening temperature in gas furnace No. 8 and percentage of Fe_2O_3 in ash.

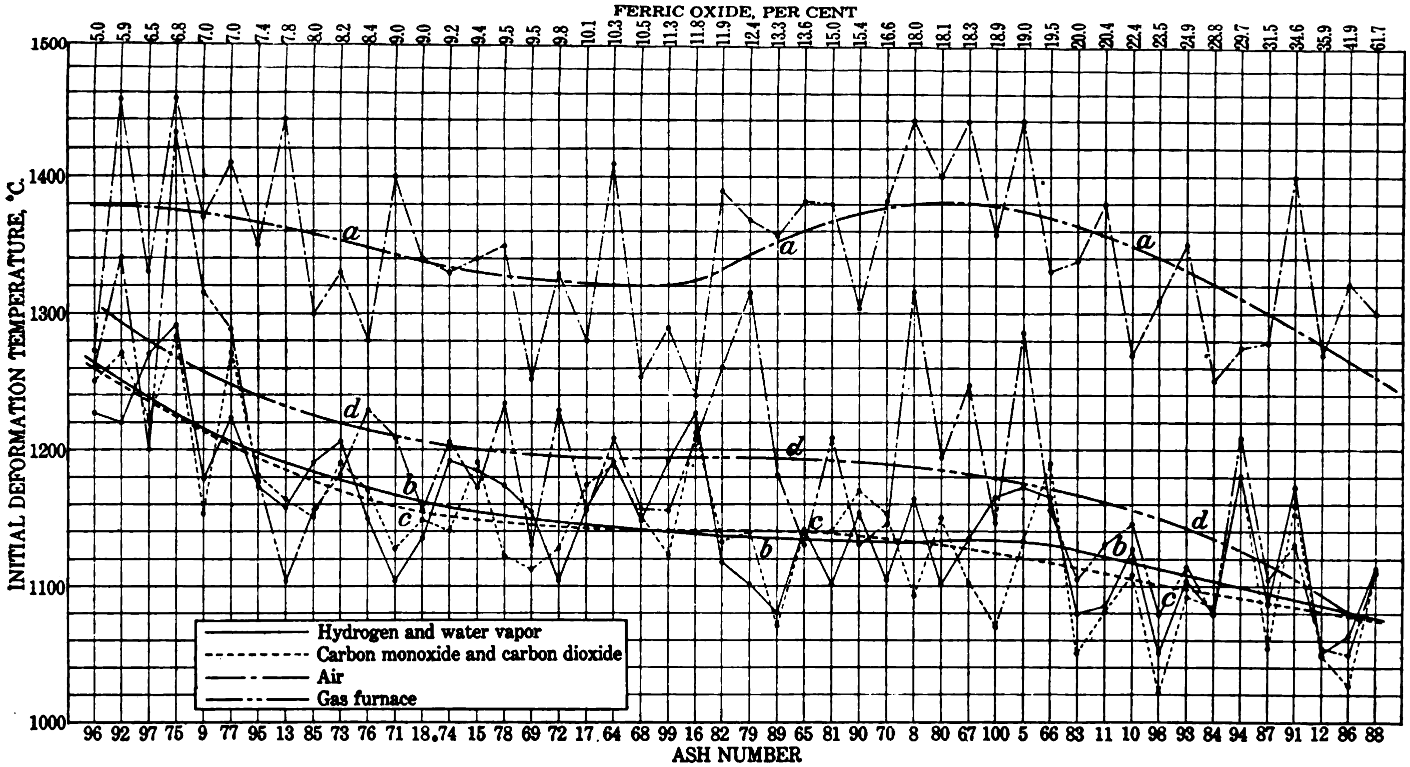


FIGURE 25.—Curves showing initial deformation points of coal ash in atmospheres of equal parts of H_2 and H_2O, equal parts of CO and CO_2, gas furnace No. 3, and air, with especial reference to the percentage of Fe_2O_3 in the ash. *a*, Average curve showing relation between initial deformation point in air and percentage of Fe_2O_3 in ash; *b*, average curve showing relation between initial deformation point in atmosphere of equal parts of H_2 and H_2O and percentage of Fe_2O_3 in ash; *c*, average curve showing relation between initial deformation point in atmosphere of equal parts of CO and CO_2 and percentage of Fe_2O_3 in ash; *d*, average curve showing relation between initial deformation point in gas furnace No. 3 and percentage of Fe_2O_3 in ash.

It will be seen from figure 25 that softening of the ash began in reducing atmospheres of H_2 and H_2O and of CO and CO_2 at practically the same temperatures, whereas in oxidizing atmospheres of air softening did not take place until a considerably higher temperature had been reached, the average difference varying from 100° to 250° C.

Many ashes, however, showed distinct differences in the temperature at which complete deformation of the cones took place in the atmospheres of H_2 and H_2O and of CO and CO_2, as shown in figure 24.

The curves representing average softening temperatures show that the differences gradually increased with an increase of Fe_2O_3—from 30° C. for an ash with 5 per cent Fe_2O_3 to 90° C. for an ash with 9 per cent Fe_2O_3. The average difference remained constant until 19 per cent Fe_2O_3 was exceeded when it rapidly decreased to 20° C. and less for ashes having 25 per cent or more of ferric oxide.

Individual samples showed differences both above and below the average curves. The largest observed difference was 197° C. The mean difference for the 48 samples tested was 66° C.

Seemingly, these differences in softening intervals of ash samples in atmospheres of H_2 and H_2O and of CO and CO_2 have some relation to the percentage of ferric oxide in the ash, becoming greatest in ashes with an iron oxide content between 9 and 19 per cent.

Silicate mixtures free from iron oxide did not show any material difference, as may be seen in the following results of tests made with standard pyrometric cones:

TABLE 14.—*Results of tests to determine softening temperatures of standard pyrometric cones*[a] *in H_2–H_2O and CO–CO_2 atmospheres, and in air.*

Cone No.	Per cent of Fe_2O_3 in Seger-cone formula.	Softening temperature in—			Difference.	
		Atmosphere of equal parts of H_2 and H_2O.	Atmosphere of equal parts of CO and CO_2.	Air.	H_2–H_2O and CO–CO_2 atmospheres.	H_2–H_2O atmosphere.
		°C.	°C.	°C.	°C.	°C.
06	8.52	1,018	1,026	1,040	+ 8	+22
04	8.54	[b]1,089	1,062	1,085	−27	− 4
02	8.57	1,119	1,119	1,178	+ 0	+59
1	8.58	1,146	1,164	1,182	−18	+36
2	4.38	1,165	1,170	1,181	+ 5	+16
4	None.	1,221	1,240	1,250	+19	+29
6	None.	1,281	1,292	1,286	+11	+ 5
8	None.	[b]1,332	1,330	1,366	− 2	+34
10	None.	[b]1,358	1,340	1,334	−18	−24
12	None.	1,358	1,370	1,367	+12	+ 9

[a] Manufactured by Edward Orton, jr., Columbus, Ohio.

[b] Average of closely checking duplicate determinations.

The average softening-point curves in figure 24 show clearly the effect of increasing percentages of iron oxide in lowering the softening temperatures of coal ashes in reducing atmospheres.

This relation was most clearly indicated in tests with an atmosphere of hydrogen and water vapor, the average softening-point curve dropping from 1,310° to 1,120° C. with an increase of Fe_2O_3 from 5 to 60 per cent. In an oxidizing atmosphere of air the average softening points for the same ashes dropped only from 1,400° to 1,340° C. Hence iron oxide in the ferric form or as magnetite is not to be regarded as having any marked effect in lowering the fusibility of a coal ash. It is only when the conditions are such as to reduce the iron component of a slag to the ferrous form that its effect becomes marked in lowering the temperature of fusion.

All the cones fused in air were strongly magnetic. Those fused in the $CO-CO_2$ atmosphere were mostly nonmagnetic, although a considerable proportion gave evidence of weakly magnetic properties when suspended in the field of a strong electromagnet. In no fused cone was any metallic iron detected under the microscope by the copper sulphate test.

The cones fused in the H_2-H_2O atmospheres were all nonmagnetic except some of those containing more than 40 per cent ferric oxide, which showed traces of metallic iron.

At first thought it would appear that a $CO-CO_2$ atmosphere should unquestionably be accepted as the proper laboratory medium for making fusion tests, as it comprises the principal constituent gases that control slag formation in the fuel bed. There is, however, some difficulty in developing a satisfactory laboratory furnace in which a definite standard concentration of CO and CO_2 can be maintained at temperatures covering the entire range of fusion temperatures of coal ash. For instance, the furnace used in the tests was limited to a maximum temperature of 1,400° C. At higher temperatures the silica tube rapidly crystallized and became permeable to gas. As refractory materials are generally permeable to gases at high temperatures, rapid gas currents would be necessary to maintain any definite atmosphere surrounding the cone.

A further seeming objection to the $CO-CO_2$ atmosphere is the slowness of the reducing and slagging reactions as shown in the experiments made in this atmosphere; approximate equilibrium seemed to be attained more rapidly in the H_2-H_2O atmosphere. It was therefore decided to extend the tests with an atmosphere of hydrogen and water vapor to temperatures above 1,400° C., a modified type of molybdenum furnace being designed for this purpose.

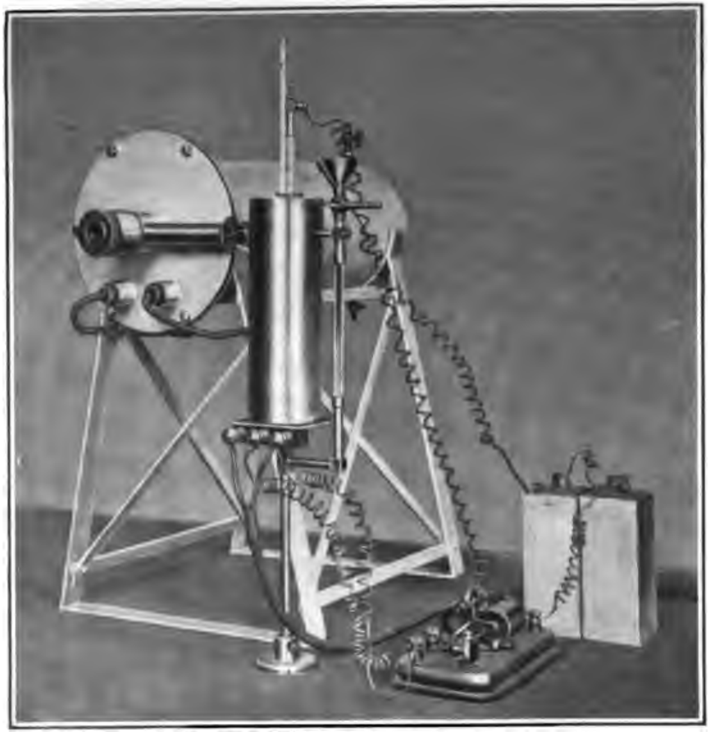
MODIFIED MOLYBDENUM FURNACE AND SATURATOR FOR SOFTENING-POINT DETERMINATIONS IN H_2–H_2O ATMOSPHERE.

In the experiments previously described the ash cone was heated in a glazed vitreosil tube, through which a current of hydrogen and water vapor was passed continuously. The tube was heated in a horizontal platinum-wire resistance furnace. This apparatus, although satisfactory for the more fusible ashes such as were studied in the former experiments, is not adapted to routine work in which samples softening at temperatures above 1,400° C. may be encountered. A molybdenum-wire resistance furnace was therefore designed in which the entire atmosphere inside the gas-tight sheet-iron jacket could be controlled within the required limits of hydrogen and water vapor (30 to 70 per cent water vapor), thus obviating the necessity of a gas-tight refractory tube.

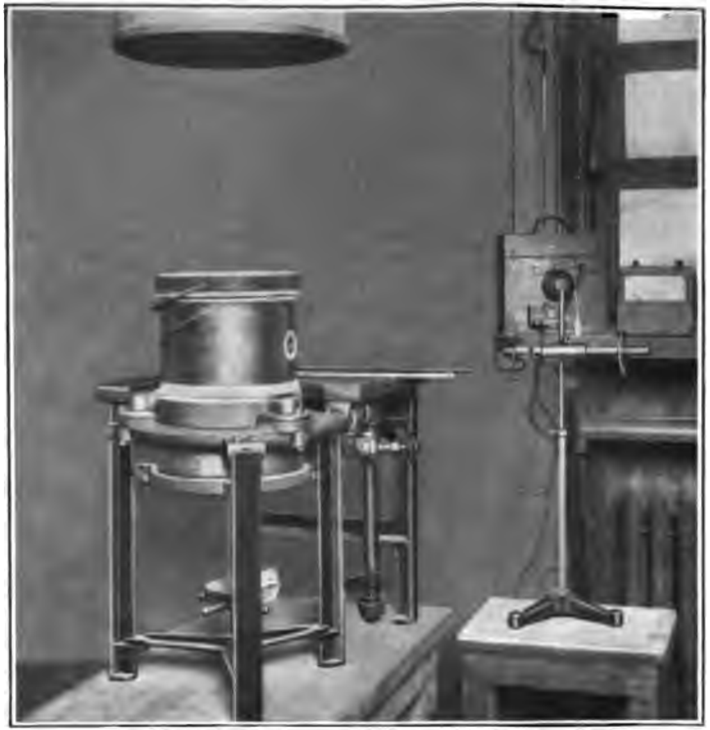
Preliminary tests, in which molybdenum wire was heated to 1,400° C. in various mixtures of hydrogen and water vapor, showed little oxidation of molybdenum in mixtures containing 60 per cent water vapor; at 70 per cent water vapor a slight superficial oxidation was noted, which was little greater at 90 per cent water vapor. Heating even in 100 per cent water vapor to 1,400° C. caused only a thin surface coating of light-colored oxide. On heating in air the metallic molybdenum was completely oxidized at 1,100° C. From these experiments it appeared that no material trouble from oxidation of the molybdenum heating element would be experienced in an atmosphere of equal parts of hydrogen and water vapor within the temperature limits required for softening most coal ashes, at least those containing over 7 per cent ferric oxide, which fuse at a temperature less than 1,500° C. For ashes that soften at a temperature higher than 1,500° C., it is permissible to run pure hydrogen through the furnace, as the iron oxide content is so low as to have little effect on the fusibility.

DESCRIPTION OF FURNACE AND SATURATOR.

The furnace and saturator are shown in Plate IV, *A*. The details are given in figures 26 and 27. The furnace consists in part of an outer shell of No. 14 B. & S. gage sheet iron, 7 inches in diameter and 11 inches long, autogenously welded, gas-tight at the seams, and having at the front a circular 1-inch flange perforated for six equally spaced bolts. Between this flange and the circular end plate, both of which are made of ¼-inch sheet iron, is placed an asbestos gasket, $\frac{1}{16}$ inch thick, painted with a thick mixture of powdered graphite and glycerin. This end plate and flange are drawn together sufficiently tight to produce a gas-tight connection. The mixture of hydrogen and water vapor enters the shell through a ⅛-inch horizontal



A. MOLYBDENUM FURNACE AND SATURATOR.



B. NO. 3 MELTER'S FURNACE ARRANGED FOR FUSING TESTS.

pipe, threaded into a 1¼-inch iron pipe serving as an observation tube, which in turn is threaded into the center of the front end plate. A vertical ⅛-inch iron pipe with a brass stopcock threaded into

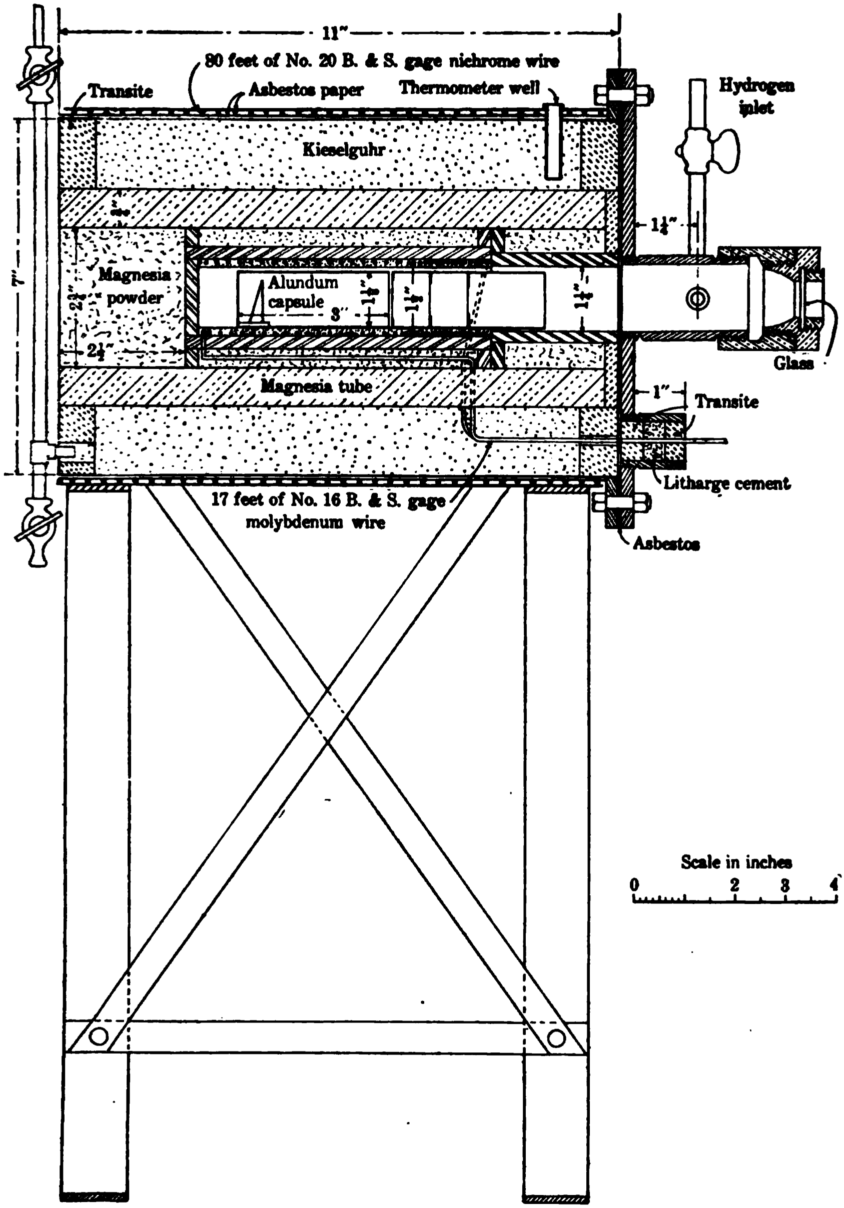


FIGURE 26.—Vertical longitudinal section through molybdenum furnace No. 3.

the top of the observation tube serves as a by-pass around the saturator for admitting pure hydrogen to the furnace. The free end of the observation tube carries a brass sleeve into which is threaded a brass cap bearing on an asbestos gasket, as shown in figure 26. The cap is fitted with a glass window, through which the temperature is read optically. The glass is held between two asbestos gaskets by a threaded brass ring. To insert a cone in the furnace, the entire cap is unscrewed from the brass sleeve. The furnace is supported in an

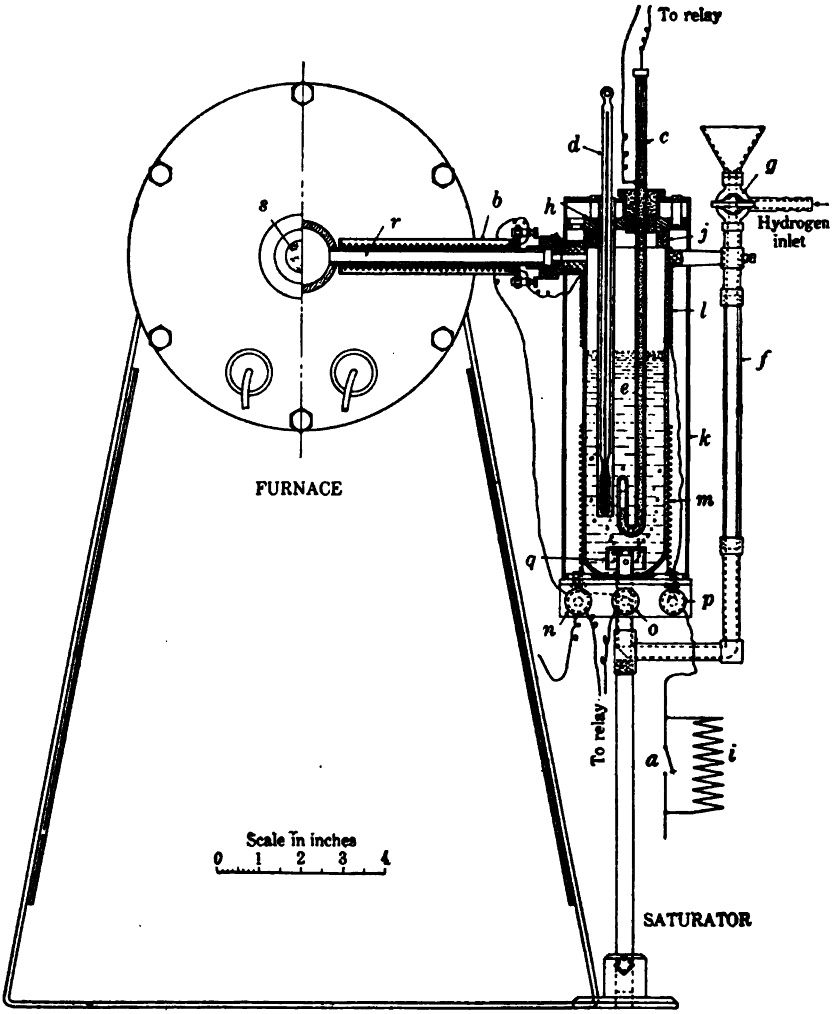


FIGURE 27.—Front elevation of molybdenum furnace No. 3 and vertical section through saturator. *a*, Switch for cutting out resistance *i*; *b*, metal jacket covering superheater; *c*, vapor-pressure thermoregulator; *d*, thermometer; *e*, water in saturator; *f*, glass gage; *g*, brass three-way cock; *h*, cover of saturator; *i*, resistance coil; *j*, body or saturator; *l*, upper heating coil of saturator for continuous duty, *m*, lower heating coil of saturator for intermittent duty; *n*, *o*, *p*, binding posts; *q*, perforated baffle for breaking up hydrogen into small bubbles; *r*, ⅛-inch iron pipe wound with asbestos and nichrome wire to prevent condensation of water vapor; *s*, glass window for observing cones.

iron frame so that its center is about 17 inches above the table, which is a convenient height for optical measurements of temperature.

The heating element consists of 17 feet of molybdenum wire, No. 16 B. & S. gage, wound upon the inside of an alundum tube, 1½ inches in internal diameter, 6 inches long, and closed at one end by an alundum disk cemented in with alundum cement. The wire is wound 6 turns to the inch except at the front, where compensation for radiation is obtained by winding 10 turns on the first inch. This tube, carrying the coil of molybdenum wire, when placed in position, lies in the axis of the cylindrical shell, 2½ inches from the rear end and 2½ inches from the front end, measured to the plane of the flange. The heated alundum tube is supported by being placed within a magnetic tube 2¾ inches in internal diameter and 10¾ inches long. The magnesite tube prevented deformation of the alundum tube at high temperatures. The narrow annular space between the tube carrying the molybdenum coil and the magnesite tube is filled with light calcined magnesia powder, free from carbonate.

The magnesite tube is provided with two slots at the front end for leading out the terminals of the molybdenum coil. At the front end the cylindrical heating chamber is continued along the axis by a short alundum tube, 1¼ inches in internal diameter. The 1¼-inch annular space between the magnesite tube and the outer iron shell is filled with previously ignited kieselguhr (infusorial earth). The magnesite tube is supported at the ends by rings of "transite," 1 inch thick. The rear terminal of the heating coil passes between the alundum tube and the magnesite tube to the front end of the heating coil, through a slot in the magnesite tube, and out of the furnace through a ¾-inch pipe, 1 inch long, threaded into the front plate. The front terminal passes through a similar slot in the magnesite tube and out of the furnace. The positions of these terminals can be better seen by reference to figure 27, which shows the front of the furnace.

The leads are centered in the pipes by an outer and an inner disk of "transite," between which is a layer of litharge cement. (See fig. 26.) This makes a gas-tight arrangement, which is not affected by the temperature to which the shell is subjected.

In order to obtain a background suitable for measuring the temperature of the test piece with accuracy, a thin alundum disk is cemented in at the rear end of the tube carrying the heating coil. The space back of this disk is provided with light calcined magnesia powder. The heating space, which is 1¼ inches in diameter and 6 inches long, affords a uniformly heated part approximately 3 inches long. The ash cone is placed in a No. 5811 alundum extraction thimble, 1⅛ inches by 3 inches. When this is to be inserted the brass cap on the observation tube is taken off and the thimble is pushed back about 1 inch beyond the center of the heating coil. Another alundum capsule having two or three diaphragms is then introduced to minimize radiation loss. Under normal conditions the cone is barely visible against the background.

In order to heat the outer shell of the furnace rapidly to a temperature at which no water can condense from the atmosphere of hydrogen and water vapor in the furnace, 110 feet of No. 20 B. & S. gage nichrome wire, about five turns per inch, is wrapped around the outside of the furnace. This length and size of wire will allow three amperes to flow when the wire is connected directly across a 220-volt circuit and will heat the outer shell to a temperature of 90° C. in 15 minutes, which will prevent the condensation of moisture from the H_2–H_2O atmosphere.

The wire is insulated from the furnace by first wrapping the iron shell with thin asbestos paper. The resistance wire is wound on this paper and fastened at each end of the furnace with asbestos cord. Another sheet of asbestos paper is then wrapped around the furnace to cover the resistance wire, as shown in figure 26.

The desired mixture of hydrogen and water vapor is obtained by bubbling hydrogen through water held at the proper temperature. The exact temperature depends on the prevailing barometric pressure and the proportions of water vapor and hydrogen desired. A temperature of 81° C. is required for a mixture containing equal parts of water vapor and hydrogen.

The saturator is constructed of brass and its temperature is controlled automatically by a vapor-pressure thermo regulator. The water in the saturator is heated by a coil of nichrome wire wound around the saturator and connected in series with a relay which is operated by the thermo regulator. Complete details of this saturator and an earlier form of the molybdenum furnace have been published in a previous report.[a] The method of operating the present form of molybdenum furnace in making a softening-temperature determination is essentially the same as described therein.

Similar results were obtained on repeating in this furnace softening-point determinations of a number of ashes that had been tested previously in the silica-tube furnace, shown in figure 17 (p. 51).

This furnace when operated as described above gave softening temperatures that were usually duplicated within the known errors of observation (about 25° C.). The fused cones from coal ash containing less than 50 per cent ferric oxide did not show any material degree of magnetism in the field of a strong electromagnet, indicating a reduction of the iron present in the ash to the ferrous state, but not to metallic iron. This condition corresponds to the minimum softening temperature, for the determination of which the furnace was designed. Most of the determinations in hydrogen and water vapor given in Table 13 were determined in this furnace, and it was given a thorough trial in routine work and in continuing the study of factors affecting the minimum softening temperature in atmospheres of hydrogen and water vapor.

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[a] Fieldner, A. C., and Feild, A. L., A new method and furnace for the determination of the softening temperature of coal ash under fuel-bed conditions: Jour. Ind. and Eng. Chem., vol. 7, 1915, p. 829.

EXPERIMENTS WITH VERTICAL AND HORIZONTAL PLACING OF CONES IN DETERMINATION OF SOFTENING TEMPERATURE.

Ricketts,[a] of the New York Edison Co., has modified the usual Seger-cone method of ash-fusing determinations by placing the cone in a horizontal position. Marks also prefers this method as being more sensitive and as giving indications which can be duplicated more accurately than the usual vertical method.[b] His objection to the vertical position was based on the seeming separation of a more fusible and very fluid constituent in certain ashes, which ran downward to the base of the cone and left the apex seemingly unchanged in a vertical or slightly inclined position until at a higher temperature the apex also melted.

On the other hand, Palmenberg[c] states that "this difficulty does not seem to present itself when the cones are made long and thin, at least not to the extent of interfering with the tests. If the ash is thoroughly ground and mixed, a complete fusion of the whole mass should take place."

A large number of determinations made with vertical cones in the Bureau of Mines laboratory did not show any separation of a fluid constituent when the determination was made in a mixture of equal parts of hydrogen and water vapor or a similar mixture of CO and CO_2, nor has there been any such separation of a low-melting slag in any coal ash containing less than 50 per cent ferric oxide when the determination has been made in the purely oxidizing atmosphere of a platinum-wire resistance furnace. However, in some coal ashes, as fused in several types of carbon and graphite resistance furnaces containing a strongly reducing atmosphere of carbon monoxide and nitrogen, certain phenomena observed were in general similar to the observations of Marks. These cones, as shown in Plate I, *C* (p. 42), did not deform in the usual manner. The upper part of the skeleton of a fused cone contained a considerable quantity of metallic iron and consisted simply of the unchanged original apex of the cone. The thickened lower part was a thin-shelled bubble of fused slag. Hence the authors consider that this phenomenon was not due to a separation of fusible constituents, but merely to an earlier fusion of the thick part of the cone. The thin part was rendered less fusible more rapidly than the thick part by reduction of iron oxide to the metallic iron.

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[a] Ricketts, E. B., Discussion of L. S. Marks's paper on "The clinkering of coal": Jour. Am. Soc. Mech. Eng., vol. 37, 1915, p. 213; Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 830–833.

[b] Marks, L. S., The clinkering of coal: Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 806–807; Jour. Am. Soc. Mech. Eng., vol. 37, 1915, p. 206.

[c] Palmenberg, O. W., Discussion of paper on "Clinkering of coal," by L. S. Marks: Trans. Am. Soc. Mech. Eng., vol. 36, 1914, p. 825; Jour. Am. Soc. Mech. Eng., vol. 37, 1915, pp. 211–212.

Although the objection cited by Marks to the use of the vertical position does not apply to softening-temperature determinations made in an oxidizing atmosphere nor in a mixture of equal parts of hydrogen and water vapor or of carbon monoxide and carbon dioxide, a number of fusibility determinations with cones placed horizontally have been made to determine whether horizontal placing in itself possessed any features that would make such placing preferable in routine testing to the usual vertical placing. The effect of inclination of the ash cone on the softening temperature has been shown previously for three different degrees of inclination, viz, 25°, 35°, and 45° from the vertical. Although the effect of the 25° inclination was negligible, the 35° and 45° inclinations gave average softening points about 20° lower than those obtained with cones in the vertical position. Undoubtedly the horizontally mounted cone should give still lower values. Marks[a] made a series of tests with cones placed horizontally and vertically in a Meker furnace having an oxidizing atmosphere. He found that the deformation temperature of a horizontal cone was always lower than of a vertical cone. The difference varied from 5° to 50° C. and was less for the more fluid melts. In making the tests with the cones in the horizontal position he took as the "down point" the temperature at which the tip of the cone pointed vertically downward.

METHOD OF CONDUCTING EXPERIMENTS.

Tests with cones in both the horizontal and the vertical position were made in the new type molybdenum furnace with an atmosphere containing equal parts of hydrogen and water vapor. The cones were three-sixteenths of an inch wide at the base by 1 inch high, and were heated at the usual rate of 5° and 2° C. per minute.

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[a] Marks, L. S., The clinkering of coal: Trans. Am. Soc. Mech. Eng., vol. 36, 1914, p. 810; Jour. Am. Soc. Mech. Eng., vol. 37, 1915, p. 206.

In tests with the cone in the horizontal position the cone was placed at right angles to the axis of the furnace, a position favoring easy observation of the softening process. The cone was supported by placing its base in a notch cut in the wall of an alundum capsule, which fitted snugly within the larger capsule used in the tests in which the cone was placed vertically. The small inner capsule was sliced longitudinally slightly above its axis, and the upper "half" was removed. Thus, the cone could be placed in position in the notch and secured by a fillet of the usual infusible mixture of kaolin and alumina. After the cone had dried, the capsule could be placed within the larger capsule without disturbing the mounted cone.

It was found just as difficult to define the down points of horizontal cones as to fix those of vertical cones. Some of the ash cones formed globules at the tips and shortened considerably in length before the axes bent appreciably; the tips of others bent vertically for only a short distance toward the bases; in a third type, the tip of the cone did not become vertical until the larger part of the cone axis was vertical or nearly vertical. In this work, therefore, the "down point" or softening point of cones placed horizontally is defined as the temperature at which a vertical line may be drawn through the tip of the cone or the center of the globule formed at the top, and a point on the axis of the cone midway between the tip and the base. Likewise a 45° deformation for horizontal cones is reached when a similar line makes an angle of 45° with the vertical.

The "softening point" or "down point" for vertical cones is the temperature at which the tip of the bent cone touches the base, or, if it fails to bend, the temperature at which the cone settles into a more or less spherical lump, which may or may not have swelled.

RESULTS OF EXPERIMENTS.

The results of softening-temperature determinations of seven synthetic mixtures and eight coal ashes with cones placed both vertically and horizontally are given in Table 15. The synthetic mixtures were made from pure North Carolina kaolin, ferric oxide, and calcium carbonate. The analyses of the mixtures and the ash samples are given in Table 16.

Table 15.—*Results of softening-temperature tests, showing relative effects of vertical and horizontal placing of cones.*

[Values in centigrade degrees.]

Sample No.	Cones placed vertically.			Cones placed horizontally.			Difference.		
	Temperature of initial deformation.	Down point	Softening interval.	Temperature of initial deformation.	Down point.	Softening interval.	Temperature of initial deformation.	Down point.	Softening interval.
M10	1,183	1,209	26	1,135	1,170	35	48	39	− 9
M23	1,103	1,105	2	1,072	1,113	41	31	−8	− 39
M33	1,063	1,086	23	1,053	1,063	10	10	23	13
M20	1,204	1,403	199	1,191	1,204	15	13	199	184
M19	1,401	1,408	7	1,216	[a]1,394	178	185	14	−171
M4	1,431	1,431	0	1,421	[a]1,428	7	10	3	− 7
M28	1,340	1,426	86	1,242	1,340	98	98	86	− 12
20913	1,164	1,202	38	1,113 1,089	1,177 1,191	64 102	63	18	− 45
19472	1,153	1,204	51	1,113 1,113	1,158 1,165	45 52	40	42	2
20914	1,080	1,116	36	1,051 1,058	1,113 1,096	62 38	25	11	− 14
16584	1,291	1,351	60	1,177	1,212	35	114	139	25
20915	1,057	1,099	42	1,044	1,063	19	13	36	23
15842	1,350	1,409	59	1,145	1,291	146	205	118	− 87
19160	1,504	1,551	47	1,482	[a]1,551	69	22	0	− 22
20233	1,320	1,378	58	1,127	[a]1,362	235	193	16	− 17
Average	....	....	49	....	....	70	71	49	− 22

[a] Cone did not bend vertically downward, but simply shortened up to a globule in the same manner as did cones mounted vertically.

Temperatures at Which Swelling of Cones Became Noticeable.

Sample No.	Cone placed vertically.	Cone placed horizontally.
	°C	°C
M33	1,063	1,058
20913	1,164	1,130
19472	1,153	1,165
16584	1,291	1,191
20915	1,057	1,055

Table 16.—*Results of analyses of coal ashes and synthetic mixtures tested with the cones used in the vertical and in the horizontal position.*

Sample No.	SiO_2.	Al_2O_3.	Fe_2O_3.	TiO_2.	CaO.	MgO.	Na_2O.	K_2O.	SO_3.	Total.
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
M10	32.6	27.9	32.7	....	6.9	0.04	....	....	....	100.1
M23	32.0	27.4	40.7	....	.03	.04	....	....	....	100.2
M33	38.1	21.4	35.5	1.3	.7	.7	0.5	2.1	0.3	100.6
M19	38.3	32.9	28.8	....	.04	.05	....	....	....	100.1
M4	45.6	39.1	5.9	....	9.6	.1	....	....	....	100.3
M28	53.3	29.9	9.7	1.8	.9	.9	.6	2.9	.4	100.4
20913	43.3	31.4	13.6	1.2	4.2	1.4	.6	2.9	1.4	100.0
19472	38.4	24.2	22.4	1.1	7.7	.9	.3	1.9	3.8	100.7
20914	35.7	23.5	32.9	1.2	3.2	1.1	.3	1.1	.5	99.5
16584	54.8	29.2	6.9	1.8	1.4	.6	1.9	2.1	1.0	99.7
20915	12.3	12.2	69.7	.4	3.9	.7	.3	.6	.2	100.3
15842	55.8	33.5	5.0	.9	1.5	.7	.5	2.2	.1	100.2
19160	54.2	33.0	7.4	....	1.1	.5	1.0	1.9	.9	100.0
20233	47.3	34.6	9.8	1.8	1.3	.4	2.1	2.5	.1	99.9

As would be expected, the cones, placed horizontally, had lower softening points than those placed vertically, the differences, varying with the viscosities of the melting eutectics, being 0° to nearly 200° C. The average difference for the 15 samples tested was 49° C. Owing to the greater force acting on the cross section of a cone placed horizontally, it begins bending at a comparatively early stage of the melting process, so that the softening interval is usually somewhat longer than that of a cone placed vertically.

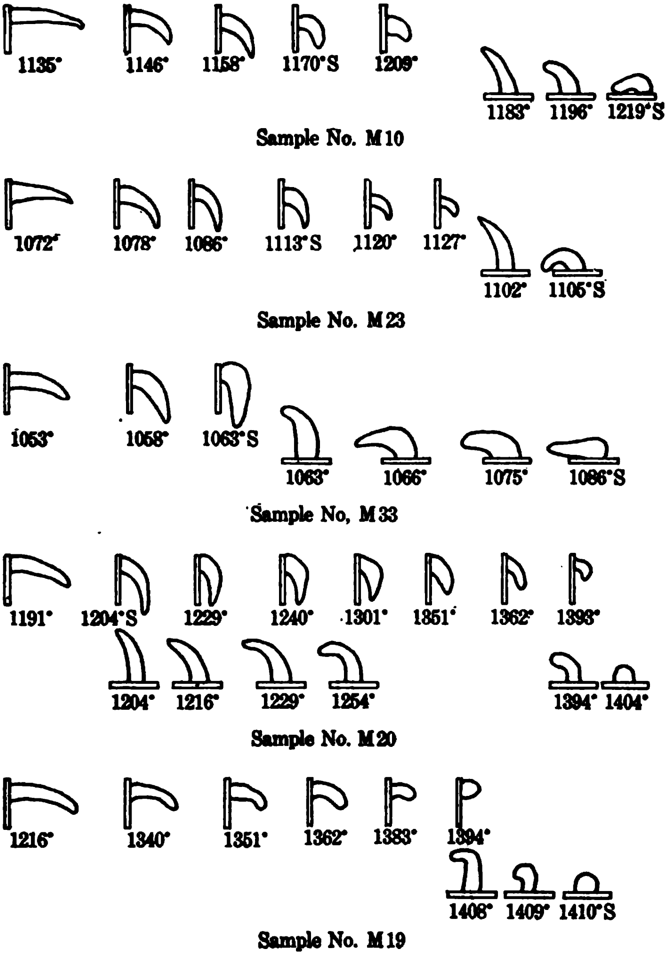

FIGURE 28.—Appearance of vertical and horizontal ash cones at various stages of softening—samples M10, M19, M20, M23, and M33.

Figures 28, 29, and 30 show the appearance of the horizontal and the vertical cones at various stages of softening.

Some of these samples, as M23, M33, M4, 20914, 19160, and 20915, showed for either position final deformation points that were not far apart on the temperature scale, whereas other samples, as M20, 15842, and 16584, reached the so-called "down point" for the horizontal position at a temperature of 100° to 200° C. below the "down point" for the vertical position. In particular, the horizontally placed cone

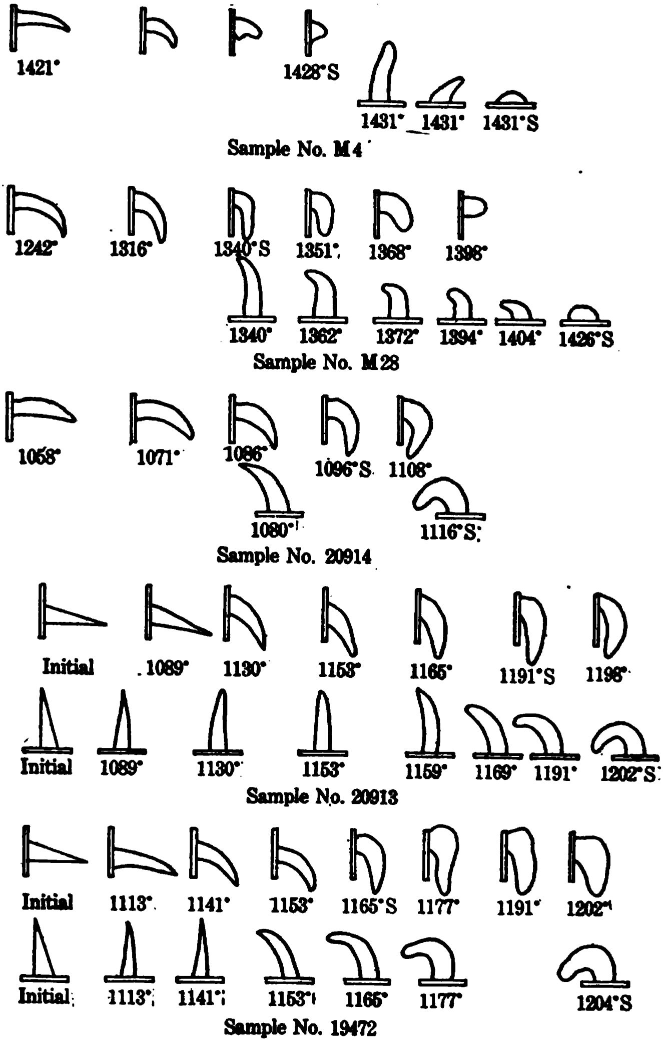

FIGURE 29.—Appearance of vertical and horizontal ash cones at various stages of softening—samples M4, M28, 19472, 20913, and 20914.

from sample 16584 reached the "down point" before any visible deformation took place in the cone placed vertically.

This difference in behavior illustrates the extremely variable character of the fusion process in coal ashes. The sudden formation of a relatively large proportion of fluid eutectic would cause the cone to deform rapidly in either position and at similar temperatures. Sample M4 is an especially good example of such a mixture. The softening interval in either position was not over 7° C., and the down point for either position was practically the same.

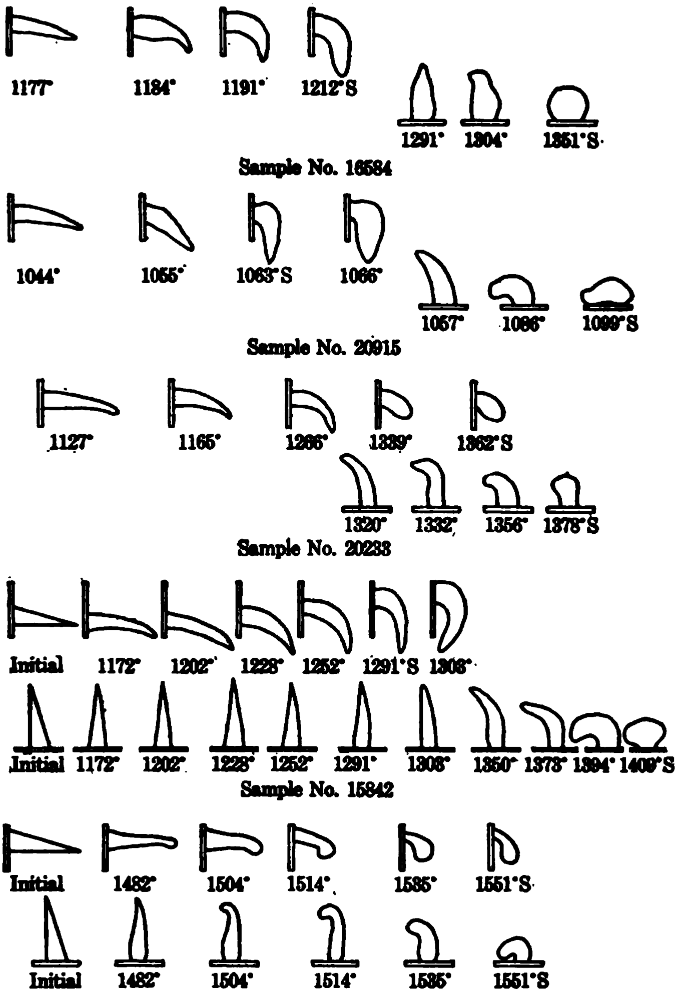

FIGURE 30.—Appearance of vertical and horizontal ash cones at various stages of softening—samples 15842, 16584, 19160, 20233, and 20915.

Sample M19 represents another type of fusion characterized by certain peculiar relations between surface tension and viscosity. The cone, whether in the horizontal or the vertical position, fused to a nearly spherical globule without bending in the usual manner.

It is evident that no fixed difference can be assigned between results obtained by vertical mounting of the cones as compared with horizontal mounting, nor is there any apparent advantage as regards closeness of duplication inherent in either method. The principal difference is that the down point of a vertical cone indicates a temperature at which the melting ash has reached a greater degree of fluidity than that indicated by the deformation point of a horizontal cone. This is undoubtedly a point in favor of the vertically mounted cone, as, according to furnace data thus far accumulated, clinker does not cause serious trouble unless it becomes sufficiently fluid to flow in the fuel bed.

EFFECT OF DEXTRIN BINDER OR RESIDUAL CARBON ON SOFTENING TEMPERATURE OF ASH, AND ITS RELATION TO INTUMESCENCE OF SOFTENING CONE.

SWELLING OF MELTING ASH CONES.

Attention has been called to the pronounced vesicular structure of ash cones fused in a hydrogen atmosphere. A similar decided "bleb" structure, accompanied in many instances by considerable intumescence, has been noted in the majority of ash samples that were fused in mixtures of hydrogen and water vapor. Indeed, some degree of swelling of the softening cone is characteristic of fusions made in any atmosphere containing material proportions of hydrogen. Swelling and intumescence, but to a distinctly less degree, occur also in atmospheres containing carbon monoxide. In atmospheres of air, water vapor, or carbon monoxide intumescence occurs rarely. It appears, then, that intumescence occurs most generally under conditions favoring the reduction of the original ferric oxide of the coal ash.

As other possible causes of swelling there may be considered: (1) Presence of substances that develop an appreciable vapor pressure at the melting temperature; (2) decomposition of sulphates, sulphides, and carbonates; and (3) oxidation of unburned particles of coke in the interior of the cone. Cause 3 especially should receive careful consideration, as even after prolonged heating in a muffle at 750° C. coal ash sometimes contains minute particles of coke, which are visible under the microscope. Unburned dextrin or other organic binding materials may also under certain conditions cause the melting cone to swell.

EXPERIMENTS TO DETERMINE EFFECT OF UNBURNED CARBON IN ASH ON SOFTENING OF CONE.

In order to determine the effect of carbonaceous binder and unburned carbon on the fusion of cones, several samples of various types of ash were reground to a fine powder and reheated five or six hours in an electric muffle at approximately 800° C. When, after this treatment, the ash still showed possible traces of coke, it was again ground in the mortar and heated for several hours in a Meker muffle furnace at a temperature of 900° to 1,000° C., until microscopic examination showed complete absence of all traces of coke. Some of these ashes were slightly "fritted" at this temperature so that repulverization was necessary before molding the cones. The results of analyses of these samples are presented in Table 17 following:

TABLE 17.—*Results of analyses of coal ashes used in experiments to determine influence of carbon on the softening temperature and the intumescence of ash cones.*

Sample No.	SiO_2.	Al_2O_3.[a]	Fe_2O_3.	TiO_2.	CaO.	MgO.	Na_2O.	K_2O.	SO_3.
	P.ct.	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*
86	31.0	22.7	42.8	....	2.1	0.5	....	....	0.6
101	42.1	32.7	19.3	....	2.5	.6	0.9	1.3	1.1
15	54.1	24.8	9.4	2.3	4.0	1.4	1.0	.8	2.8
9	54.8	27.0	7.0	1.3	4.3	1.7	.3	3.1	.6
102	56.9	27.8	9.3	....	1.6	1.1	1.1	1.3	.9
103	52.2	33.4	7.9	....	1.9	.6	1.5	1.4	.9
7	54.1	34.7	4.5	1.5	1.2	.9	.6	2.5	.2

[a] Includes P_2O_5 and TiO_2 where not separately determined.

Seven samples prepared in this way were made into three series of cones, as follows: The cones in series 1 were molded with distilled water only, so that no carbonaceous matter could be present; the cones in series 2 were molded in the usual manner with a 10 per cent dextrin solution; the cones were placed in the furnace without previous burning of the dextrin; the cones in series 3 were molded with 10 per cent dextrin solution, after an amount of sugar carbon equal to 10 per cent of the mixture had been added to the ash; these cones were also placed directly in the furnace without any previous ignition.

One series of softening-temperature tests was made in an atmosphere of equal parts of hydrogen and water vapor, in the furnace shown in Plate IV, *A* (p. 74); another series of tests covering ashes that softened under 1,400° C. was made in an atmosphere containing equal parts of CO and CO_2, in the furnace shown in Plate II (p. 46). The softening temperatures and the degree of intumescence observed are given in Table 18 following.

TABLE 18.—*Results of tests to determine influence of carbon on the softening temperature and on the intumescence of ash cones.*

Rate of temperature increase 5° C. per minute until initial softening of cone; then 2° C. per minute until cone melted.]

ATMOSPHERE OF EQUAL PARTS OF HYDROGEN AND WATER VAPOR.

Lab. No.	Softening temperature.			Differences.		Degree of intumescence.		
	Series 1—cone molded with distilled water only	Series 2—cone molded with 10 per cent dextrin solution.	Series 3—ash containing 10 per cent sugar carbon and dextrin.	Between results of series 1 and 2.	Between results of series 1 and 3.	Series 1—distilled water only.	Series 2—10 per cent dextrin solution.	Series 3—10 per cent sugar carbon and dextrin.
	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*			
22665........	1,080	1,072	1,060	− 8	−20	Marked....	Marked....	Marked.
19150........	1,146	1,182	1,177	+36	+31	.....do......	.....do......	Do.
16585........	1,278	1,291	1,254	+13	−24	Slight......	Slight......	Slight.
15848........	1,325	1,328	1,326	+ 3	+ 1	Moderate...	Moderate...	Moderate.
19155........	1,345	1,351	1,384	+ 6	+39	Slight......	Slight......	Do.
19161........	1,500	1,500	1,512	0	+12	.....do......	.....do......	Slight.
15846........	1,600	1,605	1,596	+ 5	− 4	.....do......	.....do......	Do.
Average.	....	....	....	+ 8	+ 5			

ATMOSPHERE OF EQUAL PARTS OF CO AND CO_2.

Lab. No.	Series 1	Series 2	Series 3	Between results of series 1 and 2.	Between results of series 1 and 3.	Series 1	Series 2	Series 3
22665........	1,077	1,077	1,085	0	+ 8	None.......	None.......	Very slight.
19150........	1,350	1,350	1,320	0	−30	.....do......	.....do......	Do.
16585........	1,301	1,292	1,279	− 9	−22	.....do......	.....do......	None.
15848........	1,340	1,345	1,320	+ 5	−20	Slight......	Moderate...	Slight.
Average.	....	....	....	− 1	−16			

It may be concluded from the results of the tests that the puffing or intumescence is not due primarily to the presence of unburned coke or carbon particles in the coal ash. At the rate of heat increase used in the experiments, namely, 5° C. per minute, small amounts of carbon are probably removed from the system by reacting with the gaseous atmosphere before the sintering temperature is reached. In an atmosphere containing oxygen, 50 per cent water vapor, or 50 per cent carbon dioxide, the solid carbon will be removed as carbon monoxide at temperatures below 1,000° C., provided the rate of heating is not too rapid. This conclusion is also corroborated by the experiments of Marks and those of Palmenberg. Marks[a] has shown that ash cones molded with 10 per cent dextrin solution softened in the oxidizing atmosphere of a Meker furnace at practically the same temperature as similar cones molded with distilled water only. The rate of heat increase was 2° C. per minute.

Palmenberg,[b] who also used a Meker furnace, found that as much as 10 per cent carbon had no appreciable effect on the softening temperature. A few investigators have reported that traces of carbon left in the ash or the presence of an unoxidized organic binder caused intumescence and abnormal deformation points. However, an analysis of their methods of making softening-point determinations has shown that they invariably heated the cone rapidly—40° to 70° C. per minute. Under these conditions it is quite possible that carbon was not removed by the surrounding gases before the softening temperature was reached.

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[a] Marks, L. S., The clinkering of coal: Trans. Am. Soc. Mech. Eng., vol. 36, 1914, p. 810.

[b] Palmenberg, O. W., Discussion of Marks's paper, The clinkering of coal: Trans. Am. Soc. Mech. Eng., vol. 36, 1914, p. 824.

MOLYBDENUM FURNACE AS STANDARD FURNACE FOR FUSION TESTS.

The molybdenum furnace was used for a period of one year in routine and in special softening-temperature determinations of coal ash, clays, and similar materials, thus affording an excellent opportunity to pass judgment on its practical usefulness as a standard type of furnace for determining the fusibility of coal ash. Any standard method for the determination of the fusibility of coal ash should answer the following requirements:

(1) The atmosphere should reduce the iron component of the ash to the ferrous state in which it can exert its maximum fluxing action.

(2) The method should give consistent and reproducable results when used by different chemists.

(3) The method should be simple and rapid.

(4) The furnace should be comparatively cheap and easily operated and repaired with the means available in the ordinary commercial laboratory.

If these criteria be applied to the molybdenum furnace with an atmosphere of hydrogen and water vapor as tried for a year in the Bureau of Mines laboratory, the method may be said to meet requirements 1 and 2, but to fail with respect to requirements 3 and 4.

In recent years the price of molybdenum wire has advanced materially so that the cost of renewing heating elements is a serious item. With temperatures up to 1,450° C. the element is fairly durable, but if the furnace is frequently heated to higher temperatures, as 1,600° C., oxidation by the water vapor causes the molybdenum wire to waste away and eventually burn out.

The replacement of such units, although comparatively unimportant in a laboratory equipped for renewing electric furnaces, is a more serious matter to the busy commercial laboratory. Such laboratories do not have the time nor the means to make the various electrical repairs and adjustments required in the somewhat complicated electrical system of the molybdenum furnace and satuator.

A simple form of gas furnace would be cheaper and more easily repaired; it could also be adapted to making a number of determinations simultaneously. Hence the possibilities of the gas-heated furnace were studied to determine whether the combustion could be so controlled as to produce an atmosphere of approximately constant composition and of the proper reducing character.

FUSIBILITY OF COAL ASH AS DETERMINED IN GAS FURNACE.

THEORETICAL CONSIDERATIONS.

As mentioned previously, exceedingly variable results for softening temperature have been obtained in tests made in muffle furnaces. These variations were due to the variable amounts of gas and products of combustion that penetrated the muffle and reacted with the coal ash. Now, if a furnace be constructed with a very porous muffle or one with suitable openings so that the combustion gases have free access to the cones, it should be possible to subject the cones to a reducing atmosphere merely by operating the furnace with a deficient air supply. The proportion of reducing gases in the furnace atmosphere could thus be increased to a certain maximum limit at which combustion slows down to a rate at which the desired temperature can not be attained. If this maximum amount of reducing gas does not reduce the iron constituents of the ash to metallic iron, the most desirable method of operating the gas furnace becomes simple, namely, using the lowest possible proportion of air in the mixture that will sustain the required intensity of combustion.

In the experiments described subsequently no more than traces of metallic iron were ever found under maximum reducing conditions.

DESCRIPTION OF FURNACE.

An investigation of the various gas furnaces on the market led to the selection of the No. 3 melter's furnace (Pl. IV, *B*, p. 74) of the American Gas Furnace Co. Ricketts[a] has used this furnace for a number of years for determining ash-softening temperatures in the laboratory of the New York Edison Co. This type of pot furnace is especially suitable for fusion determinations in that the burners, three in number, are on a tangent near the base of the furnace, thus producing a rotary flame which completely surrounds the crucible in which the cones are placed. The whirling flame heats the crucible uniformly and when the furnace is operated with excess gas supply, a reducing atmosphere is maintained within the crucible.

The furnace proper consists of three easily replaceable fire-clay parts, namely, a lower cylinder containing the three tangential tuyères, and forming the bottom of the furnace, a removable upper cylinder 7 inches in internal diameter and 7 inches high, and a cover plate 1½ inches thick, having a vent hole in the center for the flue gas.

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[a] Ricketts, E. B., Discussion of paper by L. Marks, The clinkering of coal: Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 801–833.

The manufacturers were asked to modify the stock design by furnishing the upper cylinder with two holes in the side—a 2-inch observation hole with its center 4 inches from the top of the cylinder (excluding cover plate), and a ¾-inch thermocouple hole 90° to the right of the observation hole; the bottoms of these two holes are in the same horizontal plane. The interior of the furnace is cylindrical and approximately 7 inches in diameter and 11 inches high. A counterbalanced sheet-iron canopy 24 inches in diameter was connected with a telescopic 8-inch flue to an exhaust system to conduct the hot gases out of the room. The canopy is shown in a raised position in Plate IV, *B*; when in use it is pulled down to the furnace cover.

The joint between the upper and the lower cylinder was made fairly gas tight by spreading on the top of the lower cylinder a thick paste of alundum cement to a depth of one-half inch and then firmly pressing the upper cylinder into place. The outer part of the joint was also smoothed over with alundum cement.

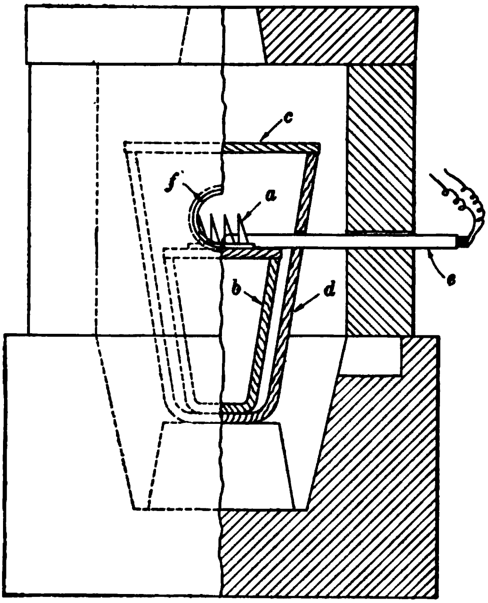

FIGURE 31.—Section of No. 3 melter's furnace arranged for fusion tests.

The interior of the furnace as arranged for making a test is shown in figure 31. The ash cones *a* are supported on a plate of alundum cement serving as a cover for the crucible *b*, which is 4 inches high and 3 inches in diameter at the bottom, outside dimensions. The cone-supporting crucible is within the crucible *d*, which with the cover *c* serves as a muffle. Crucible *d* is 7 inches high, 5 inches in diameter at the top, and 3 inches in diameter at the bottom, and is provided with observation and thermocouple holes corresponding to those in the furnace cylinder. The platinum and platinum-rhodium thermoelement is protected from the furnace gases by a glazed Marquardt porcelain tube *e* one-fourth inch in diameter. A fused silica tube *f*, 1⅛ inches in external diameter and 6 inches long, is placed in the 2-inch observation hole of the furnace, the inner end projecting through the wall of the crucible *d*, and the other end projecting out of the furnace a distance of 2 inches. A brass sleeve carrying a thin glass window was slipped on the outer end of the observation tube to prevent the escape of burning gas, which would have interfered with convenient observation of the cones.

The fire-clay furnace cylinders and cover will withstand a temperature of 1,650° C. It has, however, been difficult to obtain suitable refractory crucibles. Fused-silica ware was first used but was later abandoned on account of high first cost and short life owing to crystallization; alundum ware also went to pieces after a few runs. Finally the ordinary hard burned-clay assay crucibles were tried and found to be suitable, provided they were not heated to a temperature exceeding 1,500° C.

The average life of a crucible under these conditions is six runs; however, they cost only a few cents each, and are quickly replaced. Experiments are being made with various materials in order to obtain a cheap and durable material that may be used for temperatures up to 1,650° C.

METHOD OF OPERATION.

Natural gas, that used by the city of Pittsburgh, Pa., was supplied to the furnace at a pressure of approximately 10 to 12 inches water gage, through a ¾-inch pipe; air was supplied from a Root blower through a 1½-inch pipe. The blower developed a pressure of 2 to 3 pounds to the square inch. As this pressure was inadequate for attaining temperatures above 1,500° C. an ordinary oxygen tank was connected to the air line and the air was enriched with oxygen whereby temperatures up to the melting point of platinum (1,755° C.) were readily obtained.

With stronger air pressure, at least 5 pounds per square inch, temperatures up to 1,700° C. should be easily obtained without recourse to oxygen.

The coal ash was prepared and the cones were mounted in the manner described on pages 29–30.

Although several cones can be heated simultaneously in the gas furnace as shown in figure 31, all tests described in the following experiments were made with single cones, in order to eliminate any possible influence of volatile constituents of one cone on the softening point of another cone. The mounted cone having been placed in proper position in the crucible *d* (fig. 31) with the cover *c* in place, the gas was turned on, ignited, and allowed to burn about 10 minutes before the cover plate of the furnace was replaced; during this time the air and gas was gradually increased so as to slowly heat the crucible without cracking it. After the cover plate was in place on the furnace, the proportion of air was increased sufficiently to cause the combustion to take place just above the tuyères, and yet maintain a flame at least 6 inches high above the opening in the cover plate. From this point the temperature was gradually increased by a suitable adjustment of gas and air until it was at least 200° C. below the probable down point of the cone; then a temperature increase of 5° C. per minute was started and maintained until deformation commenced. When the cone started to bend the rate of heat increase was reduced to 2° C. per minute, and this rate was maintained until the end of the test. During the entire period of heating a reducing atmosphere was maintained in the furnace by using the minimum amount of air necessary for attaining the desired temperatures. A luminous reducing flame issued from the opening of the cover plate to a height of at least 6 inches, except at the higher temperatures above 1,450° C., which could not be attained without a larger proportion of air to gas. Fortunately a strongly reducing atmosphere is not so essential at the higher temperatures, as refractory ashes, owing to their low content of iron oxide, are only slightly affected by oxidizing or reducing atmospheres.

COMPOSITION OF ATMOSPHERE SURROUNDING CONES.

While the furnace was operated under the conditions described above, samples of gas were taken from the interior of the muffle crucible *d* (fig. 31) at various temperatures between 800° and 1,500° C. The results of analyses of the gas samples, given in Table 19 following, show a fairly uniform composition over the given temperature range. Practically all the oxygen of the air had been consumed.

TABLE 19.—*Results of analyses of gas samples taken at various temperatures from No. 3 melter's furnace.*

Temperature where sample was taken.	Percentage by volume of—							Ratio.	
	CO_2.	CO.	H_2.	O_2.	C_2H_4, etc.	CH_4.	N_2.	$\frac{CO_2}{CO+CO_2}$	$\frac{CO}{CO+CO_2}$
° *C.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
800	6.1	8.3	7.0	0.0	0.6	1.6	76.4	42	58
900	6.1	7.9	7.8	.5	.3	1.8	75.6	44	56
1,000	6.4	8.5	7.7	0	.2	.4	76.8	43	57
1,100	5.9	8.3	8.2	.3	.5	.5	76.3	42	58
1,200	6.2	8.5	9.1	.4	.2	.3	75.3	42	58
1,300	6.4	8.5	8.6	.1	.1	.3	76.0	43	57
1,400	6.9	7.3	6.5	.2	.2	.3	78.6	49	51
1,500	9.3	6.6	6.5	.2	.0	.3	77.1	59	41

These analyses do not show the real composition of the furnace atmosphere in that the water vapor was not determined. However, it is possible to calculate the percentage of water vapor from the analysis of the natural gas used and the products of combustion, as follows:

CALCULATION OF PERCENTAGE OF WATER VAPOR IN GAS SAMPLE.

Composition of natural gas used.

Constituent.	Per cent.
Methane (CH_4)	78.7
Ethane (C_2H_6)	20.3
Nitrogen (N_2)	1.0
	100.0

78.7 c.c. CH_4 forms on combustion 78.7 c.c. $CO+CO_2$ and 157.4 c.c. H_2+H_2O.
20.3 c.c. C_2H_6 forms on combustion 40.6 c.c. $CO+CO_2$ and 60.9 c.c. H_2+H_2O.
119.3 218.3

$$\text{Ratio of } H_2+H_2O \text{ to } CO+CO_2 \text{ in products of combustion} = \frac{218.3}{119.3} = 1.83$$

Hence water vapor in products of combustion $= (CO+CO_2)\ 1.83-H_2$.[a]

The percentage of water vapor was calculated in this way for each analysis, and the results were combined with the results of the analysis in order to obtain the actual composition of the furnace atmosphere at various temperatures. The values so found are given in Table 20 following.

TABLE 20.—*Composition of furnace atmosphere at various temperatures.*

Temperature.	Percentage by volume of—								Reducing gases.[a]	Oxidizing gases.[b]	Ratio of reducing gases to oxidizing gases.
	N_2.	C_2H_4.	CH_4.	H_2.	CO.	CO_2.	H_2O.	O_2.			
° *C.*											
800......	64.0	0.5	1.3	5.9	6.9	5.1	16.1	0.0	14.6	21.2	41:59
900......	64.3	.3	1.4	6.6	6.7	5.2	15.1	.4	14.0	20.7	40:60
1,000.....	64.1	.2	.4	6.4	7.1	5.4	16.4	.0	14.1	21.8	39:61
1,100....	64.8	.4	.4	7.0	7.0	5.0	15.1	.3	15.2	20.4	42:58
1,200.....	63.9	.2	.3	7.7	7.2	5.3	15.1	.3	15.4	20.7	43:57
1,300.....	63.9	.1	.3	7.3	7.2	5.4	15.7	.1	14.9	21.2	41:59
1,400.....	65.8	.2	.2	5.4	6.1	5.8	16.3	.2	11.9	22.3	35:65
1,500.....	62.9	.0	.3	5.3	5.3	7.6	18.4	.2	10.9	26.2	29:71

[a] $C_2H_4+CH_4+H_2+CO$. [b] $CO_2+H_2O+O_2$.

Methane (CH_4), ethane (C_2H_4), hydrogen (H_2), and carbon monoxide (CO) are reducing gases; oxygen (O_2), carbon dioxide (CO_2), and water vapor (H_2O) are oxidizing gases; nitrogen may be regarded as a neutral gas having neither oxidizing nor reducing properties. On this basis the ratio of reducing to oxidizing gases was fairly constant at 40 to 60 in the temperature range between 800° and 1,300° C.; above 1,300° C. the atmosphere became more oxidizing, the ratio being 29 to 71 at 1,500° C. There is seemingly no danger of obtaining in the gas furnace an atmosphere that is too reducing in character. However, care must be observed to maintain the largest possible excess of gas over air; otherwise the minimum softening point corresponding to a ferrous iron slag will not be obtained.

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[a] Neglecting water vapor in gas and air supplied to the furnace and assuming no other forms of carbon in combustion products.

TESTING FOR MAGNETIC SLAG AND METALLIC IRON.

In connection with the fusion tests made in atmospheres of equal parts of hydrogen and water vapor and of equal parts of carbon dioxide and carbon monoxide, it was shown that ash cones fused in such atmospheres showed little or no magnetism when suspended in the field of a strong electromagnet. This test was therefore used for determining the efficiency of reduction in the gas furnace. The electromagnet (fig. 32) consisted of a rod of Swedish iron, *a*, 6¾ inches long and 1 inch in diameter, upon which were wound 250 feet of No. 18 B. & S. gage insulated copper annunciator wire, *b*, in 10 layers, the whole being covered with electrician's tape to keep the wire in place. The fused cone was placed in the copper stirrup, *c*, which was free to swing on a thread supported from above. A current of 3½ amperes was used to energize the magnet.

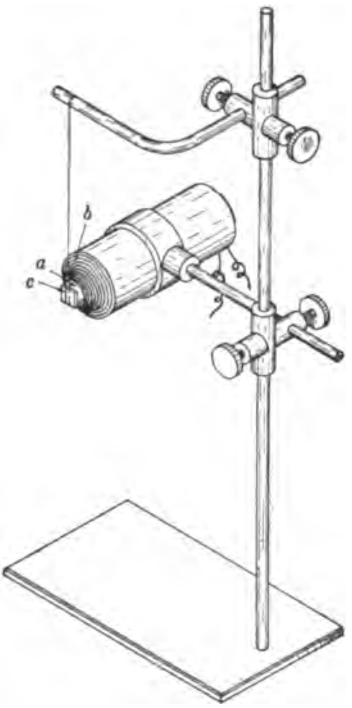

FIGURE 32.—Electromagnet used to determine magnetic properties of fused cones.

A dilute solution of copper sulphate was poured over a freshly ground surface of a fused cone. Metallic iron was then easily detected when this surface was examined under the microscope, as any particles of iron became copper plated.

TEMPERATURE MEASUREMENTS.

Most of the temperature measurements were made with a thermoelement of platinum and platinum-rhodium and a Siemens and Halske high-resistance millivoltmeter. The cold junction was kept at the temperature of melting ice. The thermoelement was protected from the furnace gases by a glazed Marquardt porcelain tube of five-sixteenth inch external diameter. This outfit could be safely used up to temperatures of 1,550° C. For higher temperatures an optical pyrometer of the Holborn-Kurlbaum type was used. The accuracy of both the thermocouple and the optical pyrometer was checked at regular intervals by determining the melting points of gold (1,063° C.), diopside (1,391° C.), and nickel (1,450° C.), as in the earlier experiments.

The whirling flame produced by the three tangential burners heated the muffle crucible so evenly that it was difficult to see the cones and observe their deformation until means was provided for locally cooling the cones slightly below the temperature of the crucible wall. For this purpose a ¼-inch Marquardt porcelain tube was inserted through the openings provided for the thermocouple tube; the tube was so placed that when it was connected with the compressed-air pipe a jet of air was directed against the cones, cooling them momentarily, and thus rendering them visible.

RESULTS OF TESTS.

To furnish a basis for comparison of the results obtainable in the No. 3 melter's furnace with the results of fusion tests with other equipment, determinations were made of the softening temperatures of the same 48 coal ashes that had previously been fused in air, in an atmosphere of equal parts of hydrogen and water vapor, and in an atmosphere of equal parts of carbon monoxide and carbon dioxide. The size of cone and rate of heating was exactly the same in each of the different atmospheres. The results obtained are given in Table 13 (p. 68) and in figures 24 and 25 (pp. 70 and 71). It will be seen that most of the gas-furnace results agree fairly well with those obtained in the CO-CO_2 atmosphere, the average difference for the two series being only 10° C. and the average softening point curve *d* (fig. 24) for the gas furnace almost coinciding with the CO-CO_2 curve *c*. There is, however, a difference, varying from 30° to 70° C., between the average softening point curve *d* and the average softening point curve *b*, representing the results obtained in the atmosphere of hydrogen and water vapor, which produced the lowest softening temperatures.

COMPARISON OF RESULTS OBTAINED IN TWO DIFFERENT TYPES OF GAS FURNACES OPERATED UNDER MAXIMUM REDUCING CONDITIONS.

The experiments described showed that it was possible to obtain duplicate results in softening-temperature determinations made under maximum reducing conditions in the No. 3 melter's furnace.

The next step in investigating the suitability of the gas furnace for use in a standard method was to determine whether comparable results would be obtained in different types of gas furnaces, provided the furnace construction was such as to allow free access of the combustion gases to the ash cones.

As a different type of furnace, a No. 29 Meker muffle furnace was modified as shown in figure 33 by removing the muffle and replacing it with a porous alundum tube, *a*, 1¼ inches in internal diameter, the open front of the furnace around the alundum tube was closed with refractory fire clay, *b;* a thin rectangular plate *c*, 1½ by 3 inches, composed of kaolin and alumina mixed in equal proportions, was placed over the burner and directly under the alundum tube *a*, to spread the flame and cause uniformly distributed heating.

Within the tube *a* were placed two short alundum capsules *d* and *e*, the former with its closed end in front, near which was placed the cone *f*, mounted as usual in a base of kaolin and alumina. Radiation was cut down by placing the capsule *e* in front of the cone; a small opening in the closed end of this capsule permitted observation and measurement of the temperature with an optical pyrometer of the Holborn-Kurlbaum type. The temperature measurements were checked in the usual manner by observing the melting points of pure gold wire and crystals of diopside. The burner connections were so arranged that either natural gas or coal gas could be used, with air at a pressure of 2 pounds per square inch. Owing to the low air pressure it was necessary to enrich the air with oxygen in order to attain the highest temperatures.

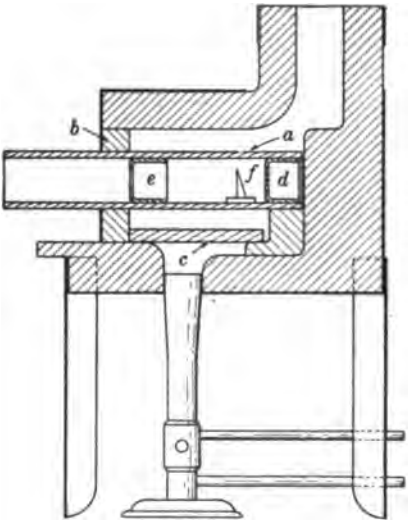

FIGURE 33.—Modified No. 29 Meker furnace.

The method of making and mounting the ash cone, its size, and the rate of heating it were the same as in the tests with the No. 3 gas furnace. The furnace was operated at all times under maximum reducing conditions, a flame issuing from the chimney to a height of approximately 6 inches. Table 21 following shows the results of these tests in comparison with those with the No. 3 melter's furnace.

Table 21.—*Results of tests to determine softening temperatures in No. 3 melter's furnace and in modified Meker furnace.*

Ash sample No.	Softening temperatures in—			Differences.	
	No. 3 furnace with natural gas.	Meker furnace with natural gas.	Meker furnace with artificial gas.	2 and 3.	3 and 4.
1	2	3	4	5	6
	°C.	*°C.*	*°C.*	*°C.*	*°C.*
86	1,076	1,087	1,074	−11	+13
88	1,125	1,134	1,124	− 9	+10
83	1,145	1,156	1,115	−11	+41
11	1,156	1,165	1,137	− 9	+28
69	1,166	1,179	1,177	−13	+ 2
100	1,172	1,153	1,153	+19	0
90	1,177	1,185	1,184	− 8	+ 1
93	1,198	1,195	1,146	+ 3	+49
17	1,218	1,212	1,189	+ 6	+23
68	1,229	1,234	1,224	− 5	+10
97	1,236	1,266	1,276	−30	−10
76	1,250	1,248	1,219	+ 2	+29
13	1,262	1,288	1,329	−26	−41
18	1,277	1,271	1,224	+ 6	+47
72	1,304	1,318	1,316	−14	+ 2
67	1,316	1,329	1,304	−13	+25
74	1,327	1,295	1,295	+32	0
95	1,325	1,283	1,291	+42	− 8
89	1,325	1,295	1,288	+30	+ 7
91	1,341	1,283	1,318	+58	−35
77	1,401	1,358	1,358	+43	0
Average				+ 4½	+ 9

A representative lot of 21 coal ashes, varying in softening temperatures from 1,070° to 1,400° C., were tested in the two furnaces, the determinations in the No. 3 melter's furnace being made by one of the authors,[a] and the determinations in the Meker furnace being made by another member[b] of the laboratory force. Thus the use of a reducing atmosphere in a gas furnace was put to a more severe test by having different operators as well as different types of furnaces and pyrometers. The agreement in the results obtained, as shown in Table 21 and in figure 34 is even better than was expected and is within the requirements of a standard method. The largest difference was 45° C., whereas 75 per cent of the samples checked within 25° C. The average difference for the series was 4½° C.

COMPARISON OF RESULTS OBTAINED WITH NATURAL AND WITH ARTIFICIAL GAS.

The tests described were all made with natural gas as the furnace fuel. The results of tests with artificial gas shown in the table and in figure 34 are explained later. The combustible constituents of the

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[a] A. E. Hall, assistant chemist.

[b] H. A. Depew, junior physical chemist.

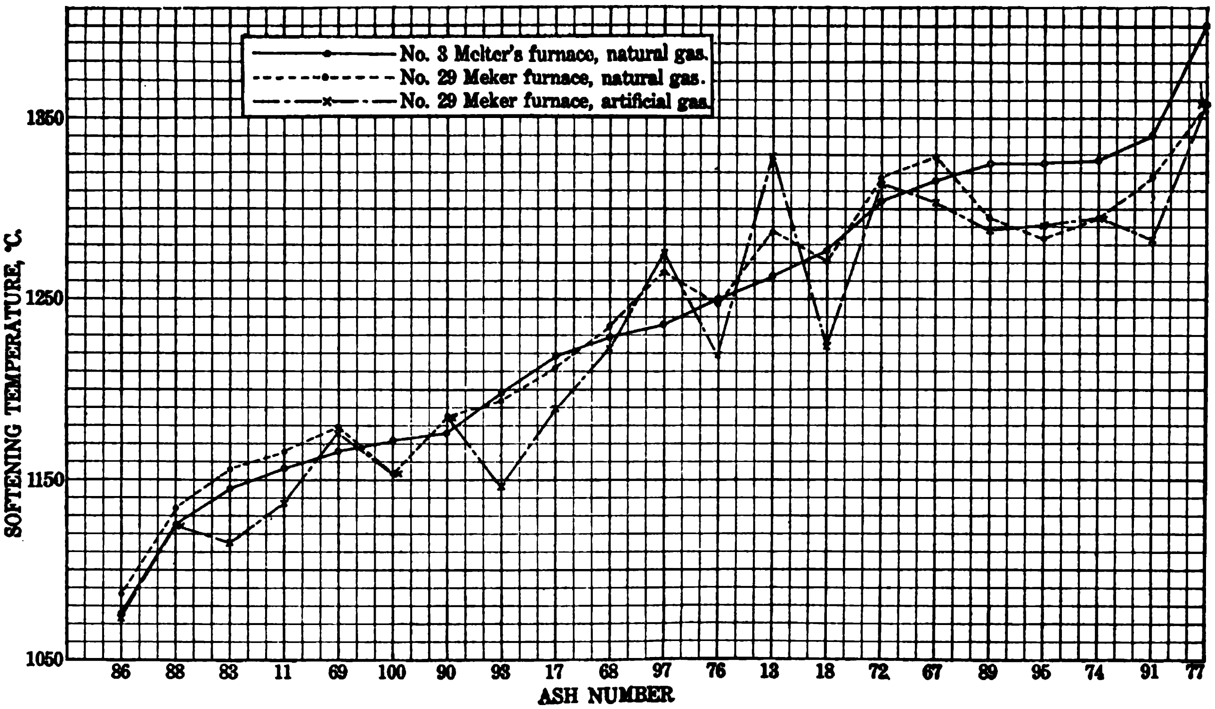

FIGURE 34.—Curves showing softening temperatures determined in No. 3 gas furnace and in modified No. 29, Meker furnace.

natural gas used in the city of Pittsburgh, Pa., comprise paraffin hydrocarbons only, the composition being as follows:[a]

Analysis of natural gas used at Pittsburgh, Pa.

Constituent.	Per cent.
Nitrogen (N_2)	1.6
Methane (CH_4)	84.7
Ethane (C_2H_6)	9.4
Propane (C_3H_8)	3.0
Butane (C_4H_{10})	1.3
	100.0

In order to determine whether the composition of the fuel gas used had any influence on the softening temperature of an ash when heated under reducing conditions, the softening points of the same lot of ash samples were again determined in the Meker furnace under exactly the same conditions as before except that artificial gas was used instead of natural gas. The artificial gas used in Pittsburgh is made up of approximately 1 part of carburetted water gas and 3 parts of coal gas. Burrell, Seibert, and Robertson[b] give the following complete analysis for this gas:

Analysis of Pittsburgh artificial gas.

Constituent.	Formula.	Per cent.
Carbon dioxide	(CO_2)	2.63
Oxygen	(O_2)	.81
Carbon monoxide	(CO)	13.25
Hydrogen	(H_2)	37.33
Methane	(CH_4)	31.13
Ethane	(C_2H_6)	2.10
Propane	(C_3H_8)	.43
Ethylene	(C_2H_4)	6.05
Propylene	(C_3H_6)	.60
Butylene	(C_4H_8)	.11
Benzene	(C_6H_6)	1.33
Nitrogen	(N_2)	4.23
		100.00

The results obtained with the artificial gas have been given in Table 21 and in figure 34. The determinations in artificial gas averaged 10° C. lower than those in natural gas. The largest deviation was 49° C., and only 5 samples in a total of 21 deviated more than 30° C. These differences are little greater than would be expected in a parallel series of softening temperature determinations in which the same gas was used; hence the use of natural gas in one laboratory and artificial gas in another should not interpose any obstacle in the way of obtaining closely similar results.

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[a] Burrell, G. A., Seibert, F. M., and Robertson, I. W., Analysis of natural gas and illuminating gas by fractional distillation at low temperatures and pressures: Tech. Paper 104, Bureau of Mines, 1915, p. 16.

[b] Same work, p. 16 and p. 28.

IGNITION OF ASH IN OXYGEN.

The effect of unburned carbon in the ash in causing puffing or intumescence of the melting ash has been discussed, especially with reference to softening tests made in an atmosphere of hydrogen and water vapor. In the H_2–H_2O atmosphere many of the cones had a marked tendency to swell even when the ash had been repeatedly ground and reignited in air and was finally made up into cones without the use of any carbonaceous binding material. Experiments made with the addition of small amounts of sugar carbon or dextrin did not produce any appreciable difference in the softening temperature or swelling tendencies of the ash; hence it was concluded that under the slow rates of temperature increase (2° C. per minute) used, the carbonaceous material reacted with the atmosphere and was removed from the system before the ash began to soften, and that the swelling in H_2–H_2O atmospheres was due to causes other than the presence of carbon.

In CO–CO_2 atmospheres and in the reducing atmosphere of the gas furnace, swelling of the cones was comparatively infrequent. However, a particularly troublesome ash in this respect led to igniting the ash in a current of pure oxygen for several hours at a temperature of 850° C. This treatment practically eliminated swelling of the cone in the reducing atmosphere of the gas furnace, and greatly reduced the degree of swelling when the ash was fused in the H_2–H_2O atmosphere. As swelling of the cone makes the softening point less distinct, and practically prevents the determination of the fluid temperature, it was deemed advisable to regrind the ashes and subject them to a second ignition in oxygen for a period of two hours, at a temperature of 800° to 850° C.

This ignition was conveniently made in the improvised electric furnace shown in figure 35. It consisted of a fused-silica tube *a*, 2¾ inches in internal diameter and 18 inches long, and having a wall one-fourth inch thick. It was open at one end and drawn down at the other end to a tube *b*, of three-eighths inch external diameter. On this tub was wound 90 feet of nichrome ribbon *c*, one-eighth inch wide and 0.032 inch thick (No. 20 B. & S. gage), the turns being as close as possible without touching; the winding was covered with a coating of alundum cement. The silica tube was supported at each end by asbestos (transite) board three-fourths inch thick, which was cut to fit snugly into the sheet-iron shell of the furnace. The space between the silica tube and the shell was packed with kieselguhr (infusorial earth). The sheet-iron cylindrical jacket was 10 inches in diameter and 17½ inches long. Oxygen was supplied from a tank through a rubber tube and a bubbling bottle at the rate of 3 to 5 bubbles per second, the rubber tube being connected to the silica tube at *b*.

Before the ash was ignited it was ground for 20 or 30 minutes in a mechanical agate mortar grinder, which reduced the ash to 200 mesh and finer. Five to ten grams of this finely ground ash was placed in a silica or porcelain capsule *d* 1¾ inches in diameter and five-eighths inch deep, and was transferred to the furnace. Six of these could be placed in the furnace at one time. The open end of the furnace was closed with a snugly fitting plug of "transite." The thermocouple was inserted through a hole in this plug. After one or two runs with the thermocouple, the temperature could be estimated with sufficient accuracy by the eye.

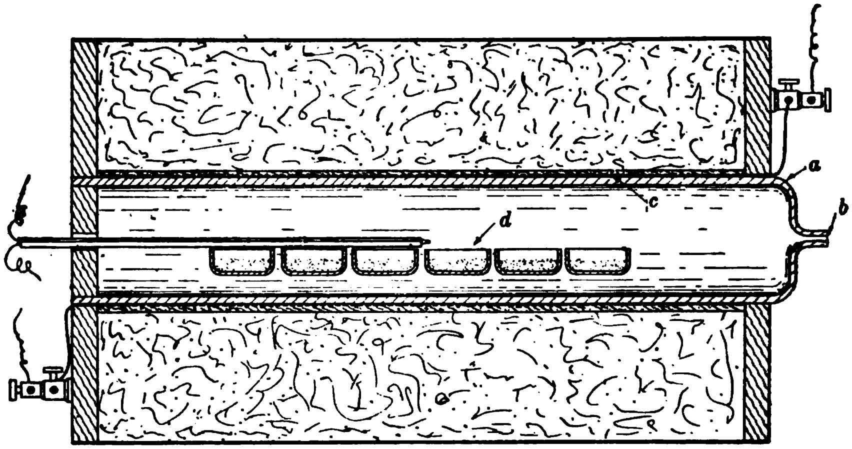

FIGURE 35.—Electric furnace for igniting ash in atmosphere of oxygen.

In heating this furnace from room temperature to 800° C., a current of 5 amperes was used during the first 10 minutes; and then 10 amperes until 800° C. was reached, 30 to 40 minutes being required in all. Five to six amperes was required to maintain the temperature at 800° C. The maximum voltage was 110 volts.

This method of ignition was used in all subsequent experiments.

INFLUENCE OF SIZE AND SHAPE OF CONE.

The ordinary Seger cone used in the ceramic industries is so proportioned that on softening it describes a smooth bend or half circle, the tip touching the base. In the early part of this investigation it was found that similar cones made of molded coal ash did not, as a rule, bend as uniformly as the standard Seger cones. Certain ashes had a decided tendency to flow down into a somewhat spherical lump without bending. Inclination of the cone did not solve the difficulty but, on the contrary, introduced more variation in the results, owing to shrinking and warping of the cone as the sintering temperature was approached. The most uniform results appeared to be obtained with a rather slender cone placed in a vertical position. A cone 1 inch high, having an equilateral triangular base, each side of which was three-sixteenths inch long, was adopted as a tentative standard and used in most of the tests previously described.

On extending this investigation to a wide variety of coal ashes some objections to the slender form of cone became evident, in that the softening-temperature determinations of certain types of ashes could not be duplicated within the usual limits (30° C.). These differences were due to irregularities in the manner of deformation. Some cones flowed or "slumped" down into a lump without bending; other cones bent over in the usual manner. The down point of such cones was always lower, sometimes as much as 200° C. Previously described experiments (pp. 79–86) have shown the impossibility of making the cones composed of certain types of ashes describe a smooth bend on softening even when the cones are mounted in a horizontal position. Furthermore, the deformation temperature of inclined or horizontally mounted cones represents in many instances a comparatively incomplete state of fusion. The authors believe that serious clinker trouble in a furnace corresponds to a more complete fusion of ash, a state in which the resulting slag flows from the force of the gravity alone, without the aid of a bending moment. Reasoning from this point of view it was decided to use a cone with a wider base in proportion to the altitude so that the vertically mounted cone would never bend but always flow or "slump" down into a spherical lump, the temperature at which the cone attained this condition to be taken as the "deformation point" or "softening temperature."

In order to determine the best proportions of base to altitude, 12 ashes, whose softening temperatures ranged from 1,050° to 1,550° C., were molded into cones of the following sizes: ³⁄₁₆ by ½ inch, ³⁄₁₆ by ⅝ inch, ³⁄₁₆ by 1 inch, and ¼ by ¾ inch. These ashes were selected as far as possible from those that had previously given trouble in irregular deformation. The cones were prepared in the usual manner and one of each size was mounted on the same kind of refractory base, and was placed in the No. 3 gas furnace in the position shown in figure 31. An exploration of the interior of the muffle crucible with the thermocouple showed a maximum variation in temperature of only 5° C.; hence each of the four different sizes of cones of a given ash was subected to the same heat treatment, and any variation of results must have been due only to differences in the size and shape of the cones. The furnace was operated with a maximum reducing flame. From a temperature of 800° C. the rate of heat increase was 5° C. per minute until initial deformation of any cone was noticed; the rate of heat increase was then 2° C. per minute until the last cone had either bent with the tip touching the base, or, failing to bend, had collapsed into a spherical lump. The temperature at which each cone reached one or the other of these conditions was recorded as the softening temperature. All the determinations were made in duplicate in order to obtain some data on the tendency toward variation in the manner of deformation of the several different sizes of cones.

RESULTS OF EXPERIMENTS.

In nearly all the tests the slender cone, 1 by $\frac{3}{16}$ inch, described the usual Seger-cone bend, whereas practically each of the other cones settled down into a somewhat spherical lump without bending. The results of the duplicate determinations for each size of cone, which are given in Table 22 following, show that the slender cones, which bent in the tests, were more susceptible to variable results than the cones with wider bases, which softened to a lump, the average difference in results of duplicate tests of a cone $\frac{3}{16}$ by 1 inch being 18.5° C., as compared to 6°, 6.5°, and 8° C., for the nonbending cones.

TABLE 22.—*Results of tests to determine softening temperatures in No. 3 melter's furnace with 4 different sizes of cones.*

[Results in centigrade degrees.]

Ash sample No.	No. 1 cones, 1 inch by $\frac{3}{16}$ inches.			No. 2 cones, $\frac{3}{4}$ by $\frac{1}{4}$ inch.			No. 3 cones, $\frac{5}{8}$ by $\frac{3}{16}$ inch.			No. 4 cones, $\frac{1}{2}$ by $\frac{3}{16}$ inch.			Difference, No. 1 cones and average for No. 2 and No. 3 cones.
	Softening temperature.	Average.	Difference.	Softening temperature.	Average.	Difference.	Softening temperature.	Average.	Difference.	Softening temperature.	Average.	Difference.	
88......	1,111 1,124	1,117	13	1,122 1,124	1,123	2	1,124 1,124	1,124	0	1,124 1,124	1,124	0	− 7
104.....	1,062 1,072	1,067	10	1,132 1,142	1,137	10	1,132 1.132	1,132	0	1,101 1,132	1,116	31	− 68
105.....	1,190 1,195	1,193	5	1,195 1,195	1.195	0	1,195 1,195	1,195	0	1,195 1,195	1,195	0	− 2
108.....	1,163 1,169	1,166	6	1,201 1,236	1,218	35	1,211 1,236	1,223	25	1,201 1,206	1,204	5	− 55
107.....	1,236 1,257	1,247	21	1,246 1,257	1,252	11	1,246 1,257	1,252	11	1,236 1.236	1,236	0	− 5
80......	1,153 1,201	1,177	48	1,331 1,342	1,337	11	1,331 1,342	1,337	11	1,331 1,342	1,337	11	−160
67......	1,298 1,356	1,327	58	1,353 1,361	1,357	8	1,353 1,361	1,357	8	1,353 1,356	1,355	3	− 30
74......	1,321 1,353	1,337	32	1,353 1,364	1,359	11	1,353 1,364	1,359	11	1,353 1,364	1,359	11	− 22
109.....	1,391 1,407	1,399	16	1,407 1.407	1,407	0	1,407 1,407	1,407	0	1,407 1,407	1,407	0	− 8
110.....	1,496 1,496	1,496	0	1,505 1,507	1,506	2	1,505 1,507	1,506	2	1,496 1,507	1,502	11	− 10
75......	1,496 1,501	1,499	5	1,507 1,512	1,510	5	1,507 1,512	1,510	5	1,507 1,512	1,510	5	− 11
106.....	1,518 1,531	1,525	13	1,531 1,531	1,531	0	1,531 1,531	1,531	0	1,531 1,531	1,531	0	− 6
Average	......	......	19	......	......	8	......	......	6	......	......	6½	− 32

The effect of varying the size and the shape of the cones on the deformation points is clearly shown in figure 36, in which the average softening temperature of each ash and of cones of each different size is plotted. The results for the No. 2 and No. 3 cones, which had the same ratios of base to altitude, are always within 10 degrees of each other. The results for the No. 4 cone, the shortest tried, also agree fairly well with those for the No. 2 and No. 3 cones. All of these cones deformed to a lump without bending. On the other hand, all of the No. 1 cones, the slender cones that usually bent on softening, had a lower softening point than the cones of the other three sizes, this difference ranging from 2° to 160° C. The observed softening point of a cone made from ash No. 80 may vary from 1,180° to 1,340° C., depending on whether the cone starts to bend at the incipient softening or whether it flows down to a lump without bending. As such variations are likely to occur with bending cones,

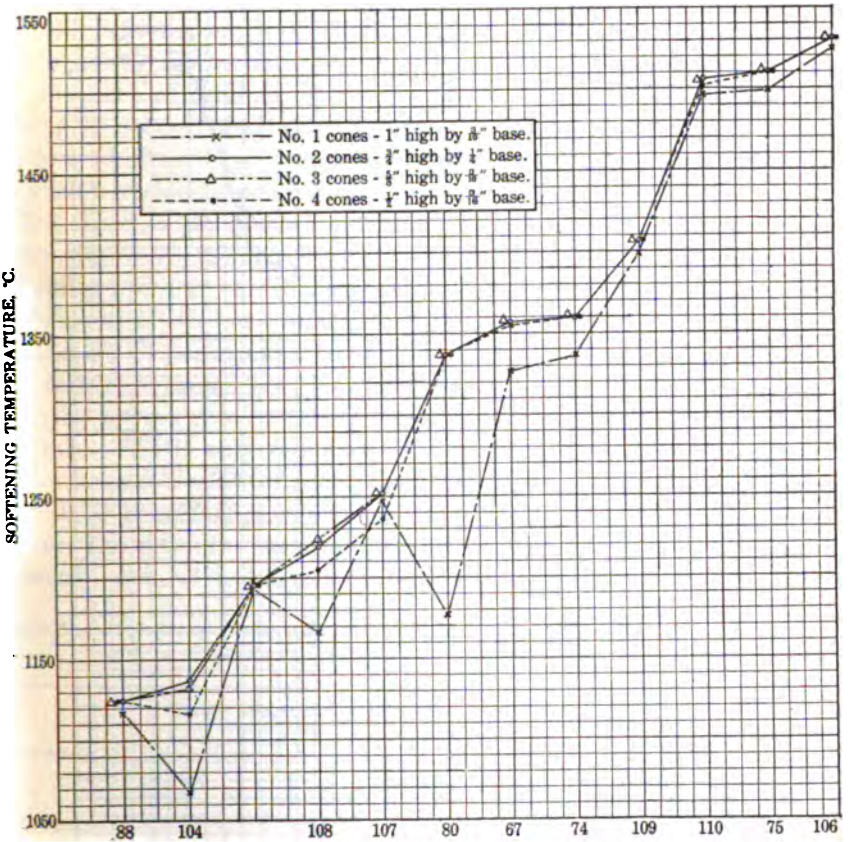

FIGURE 36.—Curves showing softening temperatures obtained in No. 3 melter's furnace with four different sizes of cones.

it seems best to rule out entirely this type of test piece and depend on one so proportioned that the force of gravity and surface tension will change the form from that of a pyramid or cone to that of an approximate sphere. Although perhaps the temperature at which this transition takes place can not be quite so definitely fixed as can the temperature at which the tip of a bending cone touches the base, the change is certainly less subject to large variations due to irregular bending, and in the opinion of the authors more nearly represents the state of fusion of ash when it forms troublesome clinkers.

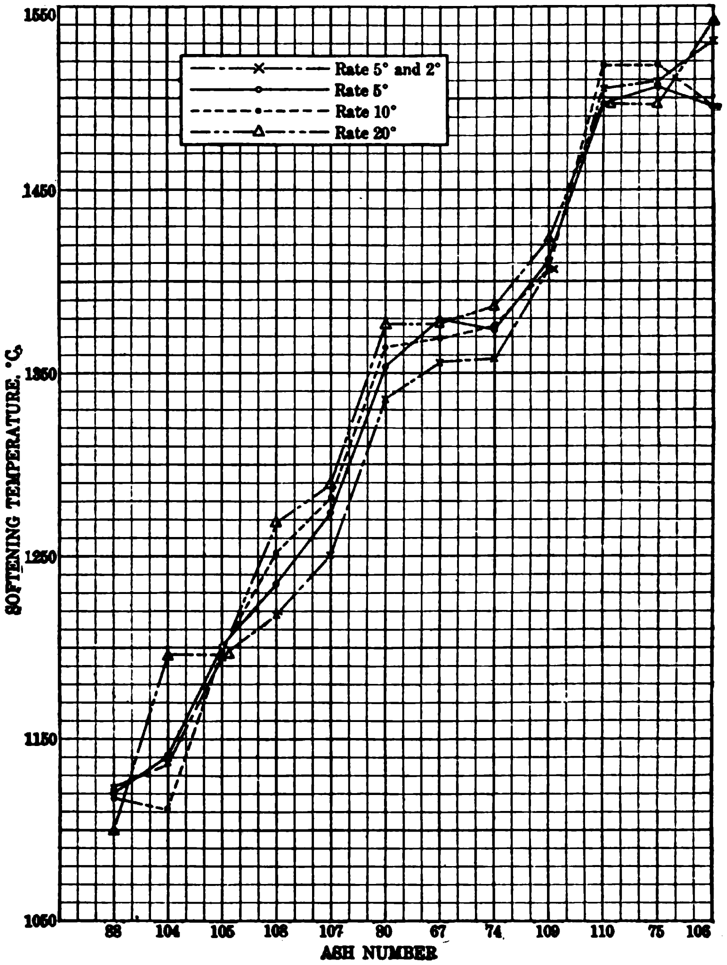

FIGURE 37.—Curves showing softening temperatures obtained in No. 3 melter's furnace at four different rates of heating.

As a result of the experiments described, a standard size of cone having an altitude of three-fourths inch, and a triangular base of one-fourth inch was used in the subsequent experiments. The softening temperature was taken at the point where the cone deformed to a somewhat spherical lump, as shown in Nos. 2, 3, and 4 cones in Plate III, *B* (p. 66).

INFLUENCE OF RATE OF HEATING.

Based on the results of experiments to determine the influence of rate of heating in oxidizing atmospheres of air and in reducing atmospheres of hydrogen, a temperature increase of 5° C. per minute to initial deformation, and then an increase of 2° C. per minute to final deformation was adopted as a standard rate in the experimental work so far described. In view of the importance of making the time of conducting a test as short as possible in commercial testing it seemed desirable to repeat these rate experiments under the new conditions prevailing in the gas furnace when operated under maximum reducing conditions.

The same 12 samples of coal ashes used in the "size-of-cone" experiments were molded into cones ¼ by ¾ inch and heated to the softening point in the No. 3 melter's furnace at four different rates, as follows: 5° C. per minute until initial softening, then 2° C. per minute, 5° C. per minute, 10° C. per minute, and 20° C. per minute. The results obtained are given in Table 23 and in figure 37. In general, the lowest rate of heating produced the lowest softening temperature. With only one exception the results of the 5° and 10° rates were within 20° of each other, and this exception deviated only 30°. It is therefore recommended that the rate of heating be kept between these limits. As a number of cones are usually heated simultaneously, it is advisable to start with a temperature of 800° C.

TABLE 23.—*Results of tests to determine softening temperatures in No. 3 melter's furnace at four different rates of heating.*

Ash sample No.	Rate of temperature increase per minute, ° C.				Differences, ° C.					
	5 and 2[a].	5	10	20	5 and 2 and 5.	5 and 2 and 10.	5 and 2 and 20.	5 and 10.	5 and 20.	10 and 20.
88	1,123	1,120	1,118	1,072 *1,127* 1,099	+ 3	+ 5	+24	+ 2	+21	+19
104	1,137	1,141	1,112	1,189 *1,204* 1,196	− 4	+25	−59	+29	−55	−84
105	1,195	1,201	1,199	1,196	− 6	− 4	− 1	+ 2	+ 5	+ 3
108	1,218	1,235	1,252	1,268	−17	−34	−50	−17	−33	−16
107	1,251	1,273	1,282	1,289	−22	−31	−38	− 9	−16	− 7
80	1,336	1,353	1,364	1,377	−17	−28	−41	−11	−24	−13
67	1,357	1,379	1,369	1,377	−22	−12	−20	+10	+ 2	− 8
74	1,358	1,373	1,375	1,386	−15	−17	−28	− 2	−13	−11
109	1,407	1,412	1,407	1,423	− 7	0	−16	+ 5	−11	−16
110	1,506	1,496	1,518	1,497	+10	−12	+ 9	−22	− 1	+21
75	1,509	1,507	1,518	1,497	+ 2	− 9	+12	−11	+10	+21
106	1,531	1,496	1,496	1,542	+35	+35	−11	0	−46	−46
Average	....	....	....	....	− 5	− 7	−18	− 2	−13½	−11½

[a] 5° C. per minute until initial deformation of cone; then 2° C. per minute.

DETERMINATION OF FLUID TEMPERATURE.

That the softening temperature of an ash, without any other information as to the change of viscosity of the mixture with temperature, must necessarily be a rather inadequate indication of probable clinkering properties has been generally recognized by most investigators who have studied this subject. Some ashes melt to a comparatively fluid slag in a short temperature range, 10° to 20° C.; others soften very slowly, there being an interval of 50° to 150° C.

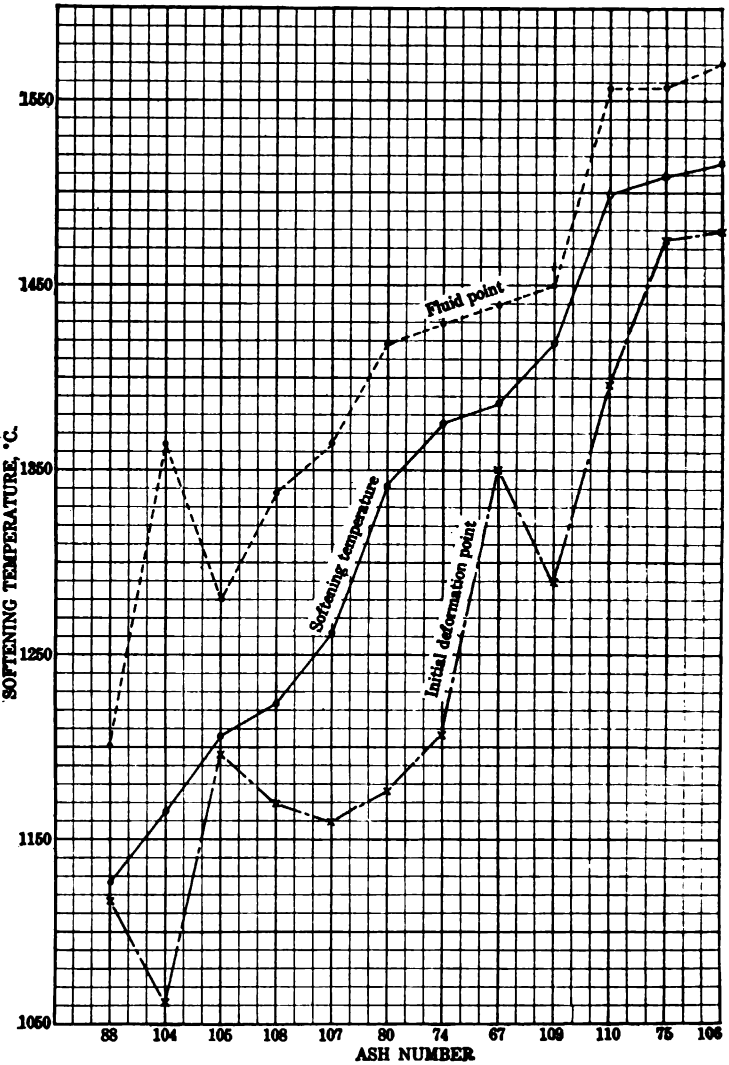

FIGURE 38.—Curves showing temperatures of initial deformation and softening and fluid temperatures of 12 ashes in No. 3 melter's furnace.

between the initial deformation of the cone and its final softening point. This temperature range has been termed by the authors the "softening interval." It is believed that the length of the softening interval, when considered with the softening point, will ultimately be proved to have some connection to clinker formation. Furthermore, at the softening temperature the fusing ash is yet in a pasty and viscous condition; the fusion process has not reached the point where the slag is fluid enough to run and drip through the grate bars of a furnace, as sometimes happens in particularly troublesome clinker formation.

Therefore it is desirable by some simple method to determine the approximate fluid temperature of an ash. One method is to continue heating the cone, after the softening point is reached, to the temperature at which the spherical lump of slag becomes fluid and spreads out in a thin layer over the base in the manner of cone 5 in Plate III, *B* (p. 66). On trying this experiment with the same 12 samples of ashes used in the preceding experiments, it was found entirely practicable to determine this approximate fluid temperature. It was, of course, impossible to reproduce these fluid temperatures in duplicate tests as closely as the softening temperatures, as the end point was less definite than the point at which the cone became a spherical lump. The fluid point, however, was sufficiently defined to have some significance. For example, referring to figure 38, in which the results of these tests are plotted, ash sample 104 began to deform at 1,060° C., it reached the softening temperature at 1,165° C., and became fluid at 1,365° C.; the total temperature interval from initial deformation to fluid point was 305° C. On the other hand, ash sample 88, which began to deform at a higher temperature—1,117° C.—became fluid at a much lower temperature, namely, 1,200° C., the total temperature interval being only 83° C. This would appear to represent the type of ash that might cause troublesome clinkers; that is, the slag would probably spread out over the grate bars in the furnace at a comparatively low temperature, choking off the draft and interfering with combustion to a much greater degree than would ash sample 104. All the "critical points"—initial deformation point, softening temperature, and fluid point—should therefore be taken into consideration in judging the probable clinkering properties of an ash. These three points give approximate data on the change in viscosity with temperature, a change that varies greatly in different ashes. All the results obtained in this series of tests are given in Table 24 following.

Table 24.—*Results of tests to determine initial-deformation, softening, and fluid temperatures of 12 samples of ashes in No. 3 melter's furnace.*

[Rate of temperature increase, 5° C. per minute; cones ¾ by ¼ inch; values in table in centigrade degrees.]

Ash sample No.	Critical points.			Fusion range.		
	Initial-deformation temperature.	Softening temperature.	Fluid temperature.	Start to down point, or softening interval.	Down point to fluid, or flowing interval.	Start to fluid, or total fusion interval.
88	1,117	1,127	1,201	10	74	84
104	1,062	1,165	1,364	103	199	302
105	1,196	1,206	1,280	10	74	84
108	1,169	1,223	1,337	54	114	168
107	1,159	1,262	1,364	103	102	205
80	1,176	1,342	1,418	166	76	242
74	1,206	1,375	1,429	169	54	223
67	1,350	1,386	1,439	36	53	89
109	1,289	1,418	1,450	129	32	161
110	1,396	1,499	1,557	103	58	161
75	1,474	1,509	1,557	35	48	83
106	1,479	1,516	[a] 1,570	37	54	91

[a] Estimated above 1,557° C.

DISCUSSION OF RESULTS OBTAINED IN GAS FURNACE WITH REDUCING ATMOSPHERE.

It was found possible to obtain consistent check results when the No. 3 melter's furnace and the modified Meker furnace were operated under maximum reducing conditions. Such conditions were obtained by allowing the combustion gases to circulate freely about the ash cones, and by so regulating the proportions of gas to air that a reducing flame at least 6 inches high burned at the outlet of the furnace. The softening temperatures thus obtained were generally between the values obtained in atmospheres of equal parts of H_2 and H_2O and atmospheres of equal parts of CO and CO_2. Usually they were very close to the results obtained in the CO–CO_2 atmospheres. Most of the iron oxide was reduced to the ferrous form, as shown by the nonmagnetic or only slightly magnetic nature of the fused cones. No metallic iron was found. A given ash sample showed practically the same softening temperature; whether natural or artificial gas was used in the furnace made no material difference. Analysis of the products of combustion from the interior of the furnace showed a ratio by volume of 40 per cent reducing to 60 per cent oxidizing gases, which is within the limits found necessary for a minimum softening temperature in CO–CO_2 or H_2–H_2O mixtures, and also approximates the composition of the atmosphere that prevails in the fuel bed of a furnace, as shown by the work of Kreisinger, Ovitz, and Augustine.[a]

In view of these facts, the method of determining the fusibility of coal ash, in a gas furnace having a reducing atmosphere, may be accepted as answering the theoretical requirements of a standard method. A furnace of the type of the No. 3 melter's furnace used in the tests was found to possess the following practical advantages for use in commercial laboratories: (1) It is a standard type of furnace obtainable on the market at a low cost; (2) it is easily operated and repaired with the facilities available in the ordinary laboratory of a commercial plant; (3) owing to the tangential burners and swirling flame the required reducing atmosphere and a very uniform temperature can be maintained around the ash cones; (4) either a thermocouple or an optical pyrometer can be used (the thermocouple must, of course, be protected by a glazed Marquardt porcelain tube, and its use is limited to temperatures under 1,550° C.); (5) the fusibility of six different samples may be determined simultaneously, thus greatly decreasing the cost of a determination.

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[a] Kreisinger, Henry, Ovitz, F. K., and Augustine, C. E., Combustion in the fuel bed of hand-fired furnaces: Tech. Paper 137, Bureau of Mines, 1916, p. 61.

RECOMMENDED PROCEDURE FOR STANDARD GAS-FURNACE METHOD.

PREPARING THE ASH.

Fifty to one hundred grams of 60-mesh coal is spread out on a 6-inch fire-clay roasting dish, and completely converted to ash in a muffle furnace at a temperature of 800° to 900° C. Five to ten grams of this ash is transferred to an agate mortar[a] and ground to a fineness of 200 mesh. The ash is then placed in a silica or porcelain capsule, five-eighths of an inch deep and 1¾ inches in diameter, and ignited for a period of two hours in a current of oxygen, at a temperature of 800° to 850° C. This ignition is made to insure complete and uniform oxidation of the ash.

PREPARATION OF CONES.

The ignited ash is moistened with distilled water or 10 per cent dextrin solution and is worked into a plastic mass with a spatula or pestle. If the ash is finely ground a skilled operator can mold and mount cones without using any binding material; however, the authors were not able to detect any deleterious effect from using a 10 per cent solution of dextrin, provided the mounted cone was ignited at a red heat in an open muffle furnace before the fusion test was made. Cones made with dextrin solution are less fragile.

The plastic material is molded into small triangular pyramids three-fourths of an inch high and one-fourth of an inch wide at the side of the base. The pyramids are made by firmly pressing the plastic material with a steel spatula into a brass mold of the dimensions mentioned, the mold being similar to that shown in figure 7 (p. 29). The surface is then struck off smooth and the cone removed from the mold by applying a small knife blade at the base. By lubricating the mold at the start with a little vaseline the cone can be removed at once without waiting for it to dry. When the cones are dry they are mounted in a refractory base composed of equal parts of kaolin and alumina. The base mixture is moistened to make it workable, and a part of it is spread out on a sheet-iron plate. The cone is then mounted in a vertical position in a small hole made in the base, and base material is put into the hole around the bottom of the cone to fill the crevices and make the cone stand firmly. Usually five or six cones are mounted in one base in the manner shown in Plate III, *B* (p. 66). The sheet-iron plate with the test piece is then dried on a hot plate. If dextrin has been used as binder the cones are ignited at a red heat for 30 minutes in an open muffle furnace to remove the carbonaceous material.

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[a] A mechanical agate-mortar grinder will save time where many determinations are made.

METHOD OF HEATING.

The test piece is placed in the muffle crucible of the melter's furnace in the position shown in figure 31 (p. 91), the loosely fitting cover *c* is placed on the crucible, and the gas is ignited. It is necessary to allow the gas to burn about 10 minutes to heat the furnace parts before the large cover plate of the furnace is replaced; otherwise the flame is apt to blow out. During this time the flow of the gas and air is gradually increased at such a rate as will not cause the crucible to crack. After the cover plate is on the furnace, the volume of gas and air is increased sufficiently to cause the combustion to take place just above the tuyères and yet maintain a yellowish flame at least 6 inches above the opening in the cover plate. While such a flame is maintained above the furnace, the temperature is gradually increased, by a suitable adjustment of gas and air, to 800° C., when the rate of heat increase is slowed down to not less than 5° C. and not more than 10° C. per minute. This rate is maintained until the end of the test. It is also important that the 6-inch reducing flame be maintained at the furnace vent throughout the test, if possible, and at all events the temperature should be kept up to 1,450° C. After a test has been completed the supply of gas and air is turned off gradually to avoid cracking the muffle crucible. The critical points observed in the test are defined as follows:

(1) *The initial deformation temperature.*—The temperature at which the first rounding or bending of the apex of the cone takes place, as shown in cone 1 of Plate III, *B*. Such bending must not be confused with a shrinking or warping of the cone.

(2) *The softening temperature.*—The temperature at which the cone has fused down to a spherical lump as shown in cones 2 and 3 of Plate III, *B*. Cone 4 has almost reached the softening temperature.

(3) *The fluid temperature.*—The temperature at which the cone has spread out over the base in a flat layer, as represented by cone 5 in Plate III, *B*.

In order to avoid the confusion that might arise in an effort to report all of the critical points mentioned it is recommended that only the softening temperature be reported as such, the other two points being given in terms of temperature intervals above and below the softening temperature. For convenience these intervals are reported as follows:

(1) *The softening interval*, which is the difference between the softening temperature and the temperature of initial deformation.

(2) *The flowing interval*, which is the difference between the fluid temperature and the softening temperature.

Following is a specimen report in which these intervals are used:

Ash No. 104.

	° C.
Softening temperature	1, 165
Softening interval	105
Flowing interval	200

Owing to the difficulty of obtaining on the market highly refractory muffle crucibles, commercial laboratories will probably not attempt to carry fusion tests above 1,500° C.; the great majority of coal ashes will soften below this temperature. Any ash that withstands 1,500° C. is in the refractory class and is acceptable as regards fusibility. Within these temperature limits the ordinary Denver fire-clay crucible No. K, 7¼ inches high and 4⅝ inches in diameter at the top may be used. Crucibles should be ordered by the hundred, and the manufacturers should be required to drill two holes in the side of each, an observation hole 2 inches in diameter with its center 2 inches from the top of the crucible and a thermocouple hole three-fourths of an inch in diameter 90° to the right of the observation hole. The bottom of this hole should be in the same horizontal plane as the bottom of the observation hole.

A Denver fire-clay crucible No. E, 4½ inches high and 3 inches in diameter at the top, is suitable for supporting the ash cones in their proper position in the large crucible.

If the maximum temperature limit is placed at 1,500° or 1,650° C., a thermocouple made of platinum and platinum-rhodium, with a high-resistance millivoltmeter or a Northrup pyrovolter, is most convenient and accurate for measuring the temperature. The thermocouple must be protected from the furnace gases with a glazed Marquardt porcelain tube. The outside diameter of this tube need not exceed five-sixteenths of an inch. The outfit should be checked frequently by mounting small pieces of pure gold or nickel wire or foil in place of the cones and observing their melting points in exactly the same manner as the melting points of the cones are observed. With a strong reducing atmosphere, 1,450° C. should be obtained for the melting point of the nickel and 1,063° C. for the gold. The thermocouple and the millivoltmeter should also from time to time be standardized throughout the temperature range for which they are used by a suitably equipped standardizing laboratory such as that of the United States Bureau of Standards.

If it should be desired to determine softening points above 1,500° C., special muffle crucibles of fused silica, magnesia, or corundite will be required, and the temperature measurements will of necessity be made with an optical pyrometer, preferably of the Holborn-Kurlbaum or Wanner type.

SUMMARY AND CONCLUSIONS.

Attention is called to the empirical nature of the "fusing or softening-temperature" test and the necessity of adopting some standard and duplicatable method if it is to be used in specifications for the purchase of coal.

The influence of the more important factors affecting the softening temperatures of the coal ashes tested may be summarized as follows:

Fineness of ash.—Ash ground to an impalpable powder tended to soften at a slightly lower temperature than ash that would pass a 100-mesh screen. The difference averaged 6° C., and in no test exceeded 40° C. Ash pulverized to at least 200 mesh was preferred, as it could be molded into more substantial cones than 100-mesh material.

Inclination of cones.—Mounting the cones with a considerable inclination—35° or 40° from the vertical—caused premature deformation points in some cones, which bent only at or near the base at a comparatively early stage in the fusion process, before the ash had softened to the flowing point. Vertical or nearly vertical cones were more free from this source of variation. Horizontally mounted cones usually gave lower softening points than vertical cones, the difference varying from 0° to nearly 200° C. The average difference for 15 samples was 49° C. In general, horizontal or inclined cones indicated a softening point at too early a stage in the fusion process; hence vertically mounted cones are recommended.

Size and shape of cone.—The deformation of a cone depended to some degree on its size and shape. Within limits of ½ to 2½ inches high the actual size had little influence provided the shape was the same; that is, the ratio between the dimensions of the base and of the altitude. Cones made of coal ash did not bend as uniformly as the standard pyrometric cones. Attempts to modify the ratio of base to altitude in order to obtain a more uniform deformation, like the usual Seger-cone bend, resulted in the selection of a rather slender type of cone—each side of base three-sixteenths of an inch; altitude, 1 inch—as having less tendency to soften to a lump without bending. However, later experience showed that certain ashes when molded into slender cones would bend when only a relatively small part of the ash mixture had fused, thus indicating a premature softening temperature; it was also impossible to prevent some ashes from fusing down to a lump even with the slender type of cone. It was, therefore, decided to eliminate entirely the bending tendency by increasing the width of the base; and to take as the softening temperature the point where the cone had softened and flowed down to a somewhat spherical lump. The optimum size for this purpose was determined experimentally and found to be a cone three-fourths of an inch high and one-fourth of an inch wide at the base. This end point was found reproducable within 30° C., and was practically free from the occasional large variations experienced with the slender bending cones.

Carbonaceous matter in the ash.—Tests in which all traces of carbon were removed and in which the cones were made up exclusively with distilled water, showed no appreciable difference in softening temperature when compared with similar tests in which the same ashes were made into cones with 10 per cent dextrin solution which was not burned out before the cones were inserted in the furnace. These tests were made in atmospheres of air, of hydrogen and water vapor, and of carbon monoxide and carbon dioxide at rates of heating not exceeding 5° C. per minute. However, in certain tests in the gas furnace with a reducing atmosphere, more rapid rates of heating caused swelling and intumescence, which was much less pronounced when the ash was previously ignited in oxygen for two hours. In routine work, ignition of the reground ash in oxygen promotes more definite and reliable softening points, especially when the original ash has been incompletely ignited.

Rate of heating.—In general, slower rates of heating gave lower softening points. Rates slower than 2° C. or faster than 10° C. per minute are not advisable. The effect of variations between these limits in purely oxidizing atmospheres was negligible. In a reducing atmosphere of hydrogen the lowest and most uniform results were obtained with a heat increase of 2° C. per minute; however, a temperature increase of 5° C. per minute until initial deformation of the cone occurred, and then reducing the rate to 2° C. per minute gave practically the same results and saved considerable time in making the test. In the gas furnace with reducing atmosphere, the optimum rate of temperature increase was between 5° and 10° C. per minute. These limits are recommended for a standard rate of heating.

Oxidizing or reducing atmosphere.—The atmosphere in which the ash was heated proved by far the most important factor in causing large variations in the softening temperature. The highest softening points were obtained either in an atmosphere of air (platinum-wire resistance furnace), or in a strongly reducing atmosphere of carbon monoxide (Northrup furnace), which prevented the iron oxide from acting as a fluxing agent by reducing it to metallic iron before the softening of the ash began. The lowest softening temperatures were obtained in those atmospheres of mixed gases in which reduction of ferric oxide proceeded mainly to ferrous oxide, the most active phase in the reduction of iron ore as regards slag formation at lower temperatures. The maximum variation in softening temperatures due to different atmospheres ranged from 134° to 396° C.

Fusibility of ash in various mixtures of hydrogen and water vapor.—The softening temperatures of five different coal ashes were determined in various mixtures of hydrogen and water vapor, ranging from 100 per cent hydrogen to 100 per cent water vapor. These results, plotted in the form of curves, showed that for each of the ashes tested there was a high softening temperature in pure hydrogen on one end owing to reduction of iron oxide to metallic iron; a similar high softening temperature in water vapor or air on the other end owing to the iron oxide remaining for the most part in the form of ferric iron or magnetite; and a somewhat lower softening temperature in the middle part, in an atmosphere ranging from 30 to 70 per cent water vapor, owing to the reduction of most of the iron to the ferrous state, in which it combined to form fusible ferrous silicates.

Fusibility of ash in various mixtures of carbon monoxide and carbon dioxide.—Softening-temperature tests in various mixtures of carbon monoxide and carbon dioxide were also made with four of the ashes that had been tested in mixtures of hydrogen and water vapor. Softening-temperature curves similar to the curves obtained in tests with atmospheres of hydrogen and water vapor were obtained, each having a minimum region of fusion between approximate limits of 75 and 10 per cent carbon monoxide. However, it was found necessary to heat the ash cones more slowly in CO–CO_2 atmospheres to attain approximately the same minimum point as was found in H_2-H_2O atmospheres at faster rates of heating. Even with a slower rate of heating in the former atmosphere the average difference in the minimum softening temperatures (down points) of a series of ashes tested in the two atmospheres was 66.1° C. The maximum difference was 197° C. The results obtained in H_2–H_2O atmospheres were generally lower than in CO–CO_2 atmospheres. The initial deformation points were practically the same in CO–CO_2 and in H_2–H_2O atmospheres.

Fusibility of ash in the reducing atmosphere of a gas furnace.—It was found possible so to operate two types of gas furnaces, in which the combustion products had free access to the coal ash, that an atmosphere of approximately 60 per cent oxidizing and 40 per cent reducing gases was maintained around the coal ash. Softening temperatures similar to those previously obtained in mixtures containing equal parts of carbon monoxide and carbon dioxide were thus obtained in a much simpler type of furnace. By operating the gas furnace with the maximum excess of gas over air, the desired reduction of ferric to ferrous iron was obtained without the production of any metallic iron.

State of oxidation of iron in ash cones fused in various atmospheres.—Ash cones fused in oxidizing atmospheres of air, carbon dioxide, and water vapor contained 67 to 88 per cent of the total iron in the ferric form; in reducing atmospheres of hydrogen and carbon monoxide, 49 to 78 per cent of the iron was present as metallic iron. In mixtures containing equal parts of H_2 and H_2O or of CO and CO_2 78 to 94 per cent of the iron was present as ferrous iron.

State of oxidation of iron in clinker and slags.—Analyses of clinker slags from two different boiler furnaces and one hand-fired experimental furnace showed that fuel-bed conditions are such as to favor the formation of clinkers containing iron principally in the ferrous state as was found in ash cones fused in mixtures of equal parts of H_2 and H_2O or of CO and CO_2.

Standard method of determining the softening temperature of coal ash.—The method of determining the softening temperature of coal ash in a gas furnace with a reducing atmosphere is recommended as a standard method. The furnace is simple, it can be easily operated, and a number of tests can be made simultaneously. When the furnace is operated under the prescribed reducing conditions reproducable results can be obtained, approximating those of the more definite mixtures of equal proportions of CO and CO_2. A standard furnace design and procedure is recommended.

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3. ——— The fusing temperature of coal ash. Power, vol. 34, 1911, pp. 802–806; New England Eng., vol. 4, 1912, pp. 7–15. Describes tests showing direct relation between fusing temperature of ash and clinkering. Amount of clinker from any one coal is affected by conditions of draft, rate of combustion, and thickness of fuel bed.

4. Blake, W. H. Coal selection and combustion. Coll. Guard., vol. 103, 1912, p. 1089. Sulphur is not the only impurity causing clinker trouble—coal with 5 per cent sulphur has been burned without trouble. Forced or induced draft may cause clinker trouble. Advises use of steam under grate.

5. Breckinridge, L. P. A study of 400 steaming tests. Bull. 325, U. S. Geol. Survey, 1907, pp. 28–29, 32–44. Curves are given showing relations of percentage of sulphur in coal, percentage of clinker in refuse, and boiler efficiency. Other curves and tables show relations of various factors to clinkering. General conclusion is that sulphur affects efficiency by formation of clinker.

6. Burgwin, Jr., H. K. Relation of composition of ash in coal to its fusing temperature. Jour. Ind. and Eng. Chem., vol. 6, 1914, pp. 694–695. Critically discusses Palmenberg's conclusion that "there is no relation between the clinkering quality of a coal and its sulphur or iron content." Shows by two charts that where the Fe_2O_3 in ash is less than 10 per cent, the fusing temperature is not below 1,399° C. (2,550° F.); where the Fe_2O_3 in ash is more than 20 per cent, the fusing temperature is never above 1,399° C.; that where $S+Fe_2O_3$ in coal is below 3 per cent, the fusing temperature is above 1,399° C.; and that where $S+Fe_2O_3$ in coal is above 3 per cent the fusing temperature is below 1,399° C. There being only 3 exceptions out of 45 observations, Burgwin concludes that, although, as Palmenberg states, "there is no definite fusing temperature to fit a definite percentage," an ash of a coal containing more than 3 per cent of $S+Fe_2O_3$ will fuse at a temperature below 1,399° C., and that it is undesirable and impossible to keep heat of boiler furnace below this temperature, with even moderately hard driving.

7. Cobb, J. W. Coal ash. Jour. Soc. Chem. Ind., vol. 23, 1904, pp. 11–13. Describes method of determining fusing temperature of ash with Seger cones and thermocouple in a gas furnace. Advises use of oxidizing atmosphere in fusion tests.

8. Constam, E. J. Ueber Schmelztemperaturen von Kohlenaschen. Ztschr. Ver. Gas- und Wasserfachmänner in Oest.-Ungarn, Oct. 15, 1913. Abstracted in Jour. Gas Lighting, vol. 124, 1913, p. 522. Describes origin of ash. Concludes that amount of ash does not influence melting point; that melting point is characteristic of coal used; and that the melting point is not affected by coking. Recommends that coke for central heating plant should have an ash fusing above 1,300° C.; anthracite and boiler coal, above 1,400° C.; and locomotive and producer coal, above 1,500° C. Carbon resistance furnace was used for fusing-temperature tests. Temperatures were determined with Holborn-Kurlbaum optical pyrometer.

9. Elwood, W. F. Efficiency valuation of fuels; importance of other factors than B. t. u. and the fusing point of ash. Coll. Eng., vol. 34, 1913, pp. 23–25. States that fusing temperature of ash has no practical value in evaluating coal. Improper firing is responsible for most clinker trouble. Clinkering should be avoided by proper methods of stoking.

10. Fieldner, A. C. Notes on the sampling and analysis of coal. Tech. Paper 76, Bureau of Mines, 1914, pp. 30, 33–34. Discusses relation of composition of ash and sulphur content of coal to fusibility of ash, also effect of various sulphides of iron on clinker formation.

11. Fieldner, A. C., and Feild, A. L. The fusibility of coal ash in mixtures of hydrogen and water vapor. Jour. Ind. and Eng. Chem., vol. 7, 1915, pp. 742–747. Coal ash was found to soften and fuse at lower temperatures in mixtures of approximately equal parts of hydrogen and water vapor than in either atmosphere by itself. In this mixture the iron of the ash was reduced principally to ferrous iron, thus tending to form the more fusible ferrous silicate slags, similar to those actually found in clinkers from furnaces.

12. ———. A new method and furnace for the determination of the softening temperature of coal ash under fuel-bed conditions. Jour. Ind. and Eng. Chem., vol. 7, 1915, pp. 829–835. Describes a molybdenum-wire electric furnace and saturator for making softening-temperature tests of coal ash in atmospheres of hydrogen and water vapor.

13. Fieldner, A. C., and Hall, A. E. The fusibility of coal ash in various atmospheres. Jour. Ind. and Eng. Chem., vol. 7, 1915, pp. 399–406, 474–481. A detailed report of a comprehensive investigation of the various factors that influence the softening temperature of coal ash as determined in various types of gas and electric laboratory furnaces.

14. Fieldner, A. C., Hall, A. E., and Feild, A. L. The fusibility of coal ash. Proc. Am. Soc. Testing Materials, vol. 15, pt. 2, 1915, pp. 297–347. A general résumé of all investigations conducted by Bureau of Mines on the fusibility of coal ash.

15. Hubley, F. C. Clinkering properties of coal. Proc. Eng. Club of Philadelphia, vol. 32, January, 1915, pp. 35–85; reviewed in Power, vol. 40, 1914, p. 796; Jour. Am. Soc. Mec. Eng., vol. 37, 1915, pp. 208–211; Power, vol. 44, 1916, pp. 523–25, 591–93. Describes the cone and the fusiometer methods of determining fusing temperature of coal ash. Latter method consists of making a cylinder of the ash one-half inch high by five-eighths inch in diameter under high pressure, placing it between

two carbon rods in gas furnace with 1½ pounds pressure on it, and heating until pointer attached to rod shows it to be half its original height. Hubley says softening range (temperature between first indication of softening and final point) is important; long range gives troublesome, gummy clinker, whereas short range gives porous, easily broken clinker. Specification of fusing temperature dependent on load conditions, etc.

16. KREISINGER, HENRY. Hand-firing soft coal under power-plant boilers. Tech Paper 80, Bureau of Mines, 1915, pp. 38–47. Clinker is fused ash; its formation is dependant on composition of ash and condition of fuel bed. Chief causes of clinkering are thick fuel bed, excessive stirring of fire, burning coal in ash pit, much slack in coal, closed ash-pit doors and preheating air admitted under grate. Precautions against clinkering are: carry thin fire, avoid excessive stirring, fire in small charges, keep burning coal from ash pit, keep ash-pit doors open, and, in very bad cases, use of limestone spread over grate. Gives instructions for burning coal containing much slack.

17. LE CHATELIER, HENRI. Étude sur la fusibilité des cendre des combustibles. Bull. Soc. D'Encourg. pour l'Ind. Nat., t. 102, 1902, pp. 223–229; abstracted in Jour. Soc. Chem. Ind., vol. 21. 1902, p. 459. Made fusing temperature tests by Seger-cone method. Found considerable variation in coals from same mine.

18. MARKS, L. S. Clinkering of coal; results of tests for effect of various constituents of the ash. Eng. News, vol. 64, 1910, p. 623. Added various ash-forming constituents of coal to a clinkering West Virginia coal and noted the effect on clinker formation. As remedy for clinkering advises reduction of combustion temperature or use of steam under grate.

19.——— The clinkering of coal. Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 801–815; Jour. Am. Soc. Mech. Eng., vol. 37, 1915, p. 206; Power, vol. 40, 1914, pp. 932–934. Gives as main factors influencing fusing temperature of ash: Character of atmosphere, size of cone, position of cone, situation of cone in furnace, and method of supporting cone. Advises horizontally mounted cone in oxidizing atmosphere. A general relation exists between fusing temperature and clinker trouble, but is not sufficiently definite to be of much practical use.

20. PALMENBERG, O. W. The clinkering of coal and its composition. The fusion temperature is not to be judged by the sulphur or iron content. Iron Age, vol. 93, 1914, pp. 410–411. Gives fusing temperatures of 38 coals with percentages of ash and sulphur in each, and percentage of Fe_2O_3 in the ash of each. Palmenberg concludes that no relation is shown between the percentages of sulphur and iron and the fusing temperatures.

21. ——— Discussion of L. S. Marks's paper on "The clinkering of coal": Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 824–828; Jour. Am. Soc Mech. Eng., vol. 37, 1915, pp. 211–212. Describes method of determining fusing temperature. Uses Meker gas furnace and optical pyrometer, and molds ash into a long slender cone.

22. ——— Fusion temperature of ash. Black Diamond, vol. 56, 1916, p. 403. Discusses effect of furnace atmosphere on the iron constituents of coal ash.

23. ——— The relation of composition of ash in coal to its fusing temperature. Jour. Ind. and Eng. Chem., vol. 6, 1914, pp. 277–279. *See also* No. 20.

24. **Parr**, S. W. Honeycomb and clinker formation. Railway Rev., vol. 55, 1914, pp. 14–15. Paper read before convention of International Railway Fuel Association. Discusses nature of the chemical combinations in coals that have a tendency to produce honeycomb and clinker formations in locomotive boilers. In the absence of air, pyrite (FeS_2) is decomposed by heat to ferrous sulphide, which is easily fusible and starts the formation of clinker and honeycomb. Clinker trouble can be minimized by eliminating "fines" especially where forced draft is used. A thin fire should be maintained with sufficient excess of air to burn FeS_2 to Fe_3O_4 without forming FeS.

25. **Parson**, C. H. Steam for preventing clinker. Power, vol. 33, 1911, p. 240. Describes different kinds of clinker, notes losses incident to clinker troubles, and states that steam jet prevents clinker formations.

26. **Phillips**, W. B. Further notes on coal impurities. Coal Age, vol. 2, 1912, pp. 111–112. Discusses papers of Bailey and Wadleigh on clinkering and fusibility of coal ash. Takes exception to their statements that there is no relation between the fusing temperature and the percentage of sulphur in the coal. Pyrite must be regarded as a source of clinker trouble.

27. **Prost**, **Eugene**. Recherches sur les relations existant entre le degré de fusibilité et la composition des cendres de houille. Rev. univ. des mines, t. 31, 1895, pp. 87–98; Moniteur sci., t. 46, 1895, pp. 560–565. *See also* English abstracts of his articles in Coll. Guard., vol. 70, 1895, p. 796; vol. 74, 1897, p. 602; vol. 75, 1898, p. 473. Gives a number of ash analyses and relative fusibilities, and derives a formula for expressing the relative fusibility of ash from its composition. For further details see this bulletin, p. 2

28. **Quickel**, R. D. The clinkering of mixed coals. Coll. Eng., vol. 34, 1914, pp. 368–371. Coals that do not form clinker when used alone may do so when mixed. Records experience with certain southern coals in locomotive practice. Describes two hard and molasses types of clinker. Gives tables of analyses and of silicate calculations.

29. **Ricketts**, E. B. Discussion of L. S. Marks's paper, The clinkering of coal. Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 830–833; Jour. Am. Soc. Mech. Eng., vol. 37, 1915, p. 213. Gives determinations of fusing temperatures in gas furnace with Wanner pyrometer and horizontally mounted cones. Gives a curve, "Efficiency *vs.* ash-fusion temperature," derived from boiler tests. Considers fusing-temperature tests good indication of clinkering qualities.

30. **Stuckenberg**, W. I. and **Kohout**, J. F. Some points about fusion temperature of ash. Black Diamond, vol. 56, 1916, p. 234. Uses cone method and Scimatco pyrometer. Tests with raw and washed coals showed similar fusing temperatures, although the ash content and the sulphur content were lowered by washing. Authors assert that fusing temperature depends mostly on intrinsic ash and is not affected by extraneous impurities.

31. **Tuspin**, S. U. The formation of clinker in coal. Elec. World. vol. 65, 1915, pp. 1184–1185. General discussion of reasons and remedies for clinkering.

32. **Voight**, **Karl**. Ueber die Schmelzbarkeit der Aschen von Brennstoffen. Ztschr. angew. Chem., Bd. 21, 1908, pp. 2461–2462. Claims that fusing-temperature test is of no practical value in buying coal.

33. Wadleigh, F. R. Notes on the impurities in coal. Coal Age, vol. 1, 1912, pp. 1206–1209. A general article on the nature of clinker formation, kinds of clinker, and the determination of clinkering properties from the fusing temperature of the ash and from analyses of the ash.

34. Wing, J. F. Management of water-gas generators. Am. Gas Light Jour., vol. 92, 1910, pp. 730–732 and 772–776. Describes water-gas producer in which an extra grate is used and temperature is raised to make the clinker flow down upon the lower grate, whence it is easily removed.

35. Editorial. The steam jet as a means of preventing clinkering in the boiler furnace. Eng. News, vol. 64, 1910, p. 632. Advises the use of steam jet as clinker preventive, and urges further study of the problem.

CONTRIBUTIONS TO THE PHYSICAL CHEMISTRY OF SLAG AND SILICATES.

36. Akermann, R. Ueber die zum Schmelzen verschiedener Hochofenschlacken erforderlich Wärmemenge. Stahl u. Eisen, Jahrg. 6, 1886, pp. 281–301, 387–396. Gives heat of formation, latent heat of fusion, and other physical data for various slags. One of the earliest works on the subject.

37. Allen, E. T., and White, W. P. Diopside and its relation to calcium and magnesium metasilicates. Am. Jour. Sci., vol. 27, 1909, pp. 1–47. Describes results of using thermal method of determining melting-point curve of various mixtures of $CaSiO_3$ and $MgSiO_3$. Describes eutectics, giving melting points. Only one stable form, identical with diopside was found; it had a melting point of 1,380° C.

38. Ashley, H. E. Slag constitution studied by means of the triaxial diagram with rectangular coördinates. Trans. Am. Inst. Min. Eng., vol. 31, 1901, pp. 855–883. Discusses use of the triaxial diagram in plotting the data obtained by Hofmann on the formation temperatures of slags.

39. Babu, L. Métallurgie générâle. Paris, 1904, vol. 1, pp. 470–543. Discusses general properties of slags, silicate degree, and formation and melting temperatures, and gives diagrams for two or more than two constituents.

40. Bleininger, A. V. Some aspects of the physical chemistry of fusing silicates. Trans. Am. Cer. Soc., vol. 9, 1907, pp. 419–460; Sprechsaal, Jahrg. 42, 1909, pp. 170–172. Enumerates various methods of attack upon ceramic physical-chemical problems and calls attention to some of the laws involved.

41. Bleininger, A. V., and Brown, G. H. The behavior of fire bricks under load conditions at a temperature of 1,300° C. Trans. Am. Cer. Soc., vol. 12, 1910, p. 337. Gives full description and results of test.

42. ——— The testing of clay refractories, with special reference to their load-carrying capacity at furnace temperatures. Tech. Paper No. 7, U. S. Bureau of Standards, 1911, 78 pp. Discusses composition and fusion of refractory clays, effect of heat on pure clays; effect of accessory constituents such as quartz, alumina, titanium oxide, iron oxide, alkalies, mica, lime, and magnesia upon the softening temperature of fire clays; effect of fluxes on the refractoriness; Richter's law; Bischof's refractory quotient; and effect of the RO in the Seger-cone series. Describes a granular-carbon resistance furnace used in determining the softening point of the fire clays.

43. Day, A. L., and Shepard, E. S. The lime-silica system of minerals. Am. Jour. Sci. vol. 22, 1906, pp. 265–302. An exhaustive study of this binary system, by physico-chemical methods, thermal and microscopic. Calls attention to the fact that softening-temperature curves of mechanical mixtures molded in the form of cones do not give reliable data regarding melting points, as a cone, when heated, begins to weaken as soon as the eutectic begins to melt, and further progress is governed entirely by the relative quantity of eutectic present and its viscosity after melting. In general, the formation temperature is below the melting point of the fused mineral except where the constituents are coarsely ground and poorly mixed.

44. Doelter y Cisterich, C. A. Handbuch der Mineralchemie. Dresden, 1912, Bd. 1, pp. 628, 925–955. Discusses silicate formation, melting points, etc. Gives tables of melting points of common minerals, comparing points of different observers. Discusses various methods of determining melting point, temperatures, viscosity, and crystal formation. Describes metallurgical slags and their viscosity, crystallization of the slag minerals, etc.

45. ——— Die Schmelzen des metakalzium-silikates, $CaSiO_3$ und des Alumosilikates, $Ca_3Al_2Si_2O_{10}$. Tonind. Ztg., Bd. 35, 1911, p. 118. Discusses measured electrical resistances.

46. Endell, K., and Rieke, R. Ueber die Umwandlungen des Kieselsäureanhydrid bei höheren Temperaturen. Ztschr. anorg. Chem., Bd. 79, 1912, pp. 239–259. Describes experiments to determine melting and transformation points of various silica minerals in iridium furnace, with Ir-Ir Ru thermocouple.

47. Feild, A. L. A method for measuring the viscosity of blast-furnace slag at high temperatures. Tech. Paper 157, Bureau of Mines, 1916, 27 pages. In the method described, the liquid slag is confined between two concentric cylinders of graphite. The outer is rotated and the inner is suspended and the torque exerted upon it is measured. The average viscosity at 1,500° C. of 8 slags measured was 301 (H_2O at 20° C. =1), which is less than that of castor oil at room temperature but greater than that of olive oil. Apparatus could be used with temperatures up to 1,600° C., but was not feasible with iron-bearing slags on account of graphite parts and reducing atmosphere.

48. Fenner, C. N. The stability relations of the silica minerals. Am. Jour. Sci. vol. 36, 1913, pp. 331–384. A study of the inversion temperatures of quartz, tridymite, and cristobalite. Gives melting point of cristobalite as 1,625° C.

49. Fulton, C. H. Principles of metallurgy. 1910. pp. 146–196, 245–288, 290–293. Discusses measurement of high temperatures, and gives composition, formation, and melting temperature of various silicate slags, with tables and curves. Outlines physical properties such as viscosity, specific heat, heat of formation, and latent heat of fusion, and discusses the system of Fe-FeS.

50. Greiner, F. E. Ueber die Abhängigkeit der Viscosität in Silikat-schmelzen von ihrere chemischen Zusammensetzung. Inaug. dissertation, Univ. Jena. 1907. Describes tests in which with Na_2SiO_3 taken as a standard, $FeSiO_3$, $MnSiO_3$, $Fe_2\ (SiO_3)_3$, $MgSiO_3$, lowered the viscosity in the order named. $CaSiO_3$ and especially $Al_2\ (SiO_3)_3$ raised the viscosity. SiO_2 raised the viscosity, and B_2O_3 and WO_3 lowered it. An apparatus similar to Tamman's was used.

51. Hofman, H. O. General metallurgy. 1913, pp. 433–470. Gives composition, formation temperatures, and melting temperatures of various silicate slags, with table and curves. Discusses physical properties.

52. Howe, H. M. Notes on the use of the triaxial diagram and triangular pyramid for graphical illustration. Trans. Am. Inst. Min. Eng., vol. 28, 1898, pp. 346–354. States that formation temperature is higher than fusion temperature.

53. Johnson, J. E., jr. The effect of alumina in blast-furnace slags. Trans. Am. Inst. Min. Eng., vol. 44, 1912, pp. 123–142. Gives effect of Al_2O_3 and MgO in blast-furnace slags. Considers influence of viscosity and methods of measuring it.

54. Juptner, H. The constitution of slags and the part they play in the metallurgy of iron. Jour. Iron and Steel Inst., vol. 58, pt. 2, 1900, pp. 276–310. Gives three main divisions of slags—silicates, phosphates, and oxides. Refers to work of Ledebur, Akermann, Gredt, Vogt, Hofman, and others.

55. ——— Grundzüge der Siderologie. Leipzig, Bd. 1, 1900, 315 pp.; Bd. 2, 1902, 408 pp. Gives table of slag minerals.

56. Konstantinov, N., and Sélivanov, B. The artificial preparation and fusibility of iron calcium silicates. Ztschr. Kryst. u. Min., Bd. 55, 1915, p. 198; Chem. Abs., vol. 10, 1916, p. 441. Describes investigation of system $FeSiO_3$–$CaSiO_3$ by thermal and optical methods. $FeSiO_3$ crystallized well only when slowly cooled, and solidified at 1,145° to 1,160° C.

57. Percy, John. Metallurgy. London, 1875, pp. 46–86. Discussion of composition and physical properties of slags, mainly silicates.

58. Peters, E. D. Principles of copper smelting. 1907, pp. 339–400. Describes practical study of slags, giving composition, melting temperatures, and formation temperatures of copper slags.

59. Rankin, G. A., and Merwin, H. E. The ternary system CaO–Al_2O_3–MgO. Jour. Am. Chem. Soc., vol. 38, 1916, pp. 568–588. A physico-chemical study in which the method of quenching was used to determine the melting temperatures. Concentration-temperature models and equilibrium diagrams are given.

60. Rankin, G. A., and Wright, F. E. The ternary system CaO–Al_2O_3–SiO_2. Am. Jour. Sci., vol. 39, 1915, pp. 1–79. A summary of the complete investigation of this system, giving tables of melting points of minerals, compounds, and eutectics, with triaxial diagrams and concentration-temperature models.

61. Sélivanov, B. Préparation artificielle, fusibilité et quelques autres propriétes des silicates du systeme $2FeO.SiO_2+2CaO.SiO_2$. Rev. Soc. Russe Métal., 1915, p. 328; Rev. Métall., Extraits, t. 12, 1915, pp. 309–318. Chem. Abs., vol. 10, 1916, p. 1157. Describes preparation, fusibility, and some other properties of silicates of the system $2FeOSiO_2 2CaOSiO_2$. Weighed quantities of FeO, CaO, and SiO_2 were heated in a platinum boat to a certain temperature for a given time, in an atmosphere of nitrogen, and allowed to cool. As a result of heating, the FeO forms Fe and Fe_3O_4 in some degree. No melting temperatures are given in abstract account.

62. Shephard, E. S., and Rankin, G. A. The binary system of alumina with silicia, lime, and magnesia. Am. Jour. Sci., vol. 28, 1909, pp. 293–333. Ztschr. anorg. Chem., Bd. 68, 1910, pp. 370–420. A thermal and microscopic study of the binary systems. Temperature-concentration diagrams are given, showing the melting points of compounds and eutectics.

63. **Sosman, R. B.** The common refractory oxides. Jour. Ind. and Eng. Chem., vol. 8, 1916, pp. 985–990. Summary of reliable data on melting and inversion temperatures, with diagrams of two and three component systems of the oxides SiO_2, Al_2O_3, MgO, CaO, Fe_2O_3, Fe_3O_4, and FeO.

64. **Sosman, R. B., and Merwin, H. E.** Preliminary report on the system lime-ferric oxide. Jour. Wash. Acad. Sci., vol. 6, 1916, pp. 532–537. States that Fe_2O_3 dissociates to some degree in all melted mixtures of CaO and Fe_2O_3, especially when the latter is in excess. Mixtures containing less than 50 molecular per cent CaO were not heated above 1,250° C.; only two binary compounds could be found, $CaO.Fe_2O_3$ and $2CaO.Fe_2O_3$. $CaO.Fe_2O_3$ dissociated at 1,216° C., liquefying in part; $2CaO.Fe_2O_3$ dissociated at 1,436° C., liquefying in part.

65. **Turner, T.** The physical and chemical properties of slags. Jour. Soc. Chem. Ind., vol. 24, 1905, pp. 1142–1149. Presents record of experiments at University of Birmingham with blast-furnace slag, in which thermal method of obtaining cooling curves was used. Concludes that silicate slags are heterogeneous mixtures with no definite melting point to which the phase rule is applicable.

66. **Vogt, J. H. L.** Die Silicatschmelzlösungen. Christiania, vol. 1, 1903, 161 pp.; vol. 2, 1904, 235 pp. Volume 1 deals with minerals found in silicate fusions with special reference to the dependence of mineral formation upon the composition of the solution. Volume 2 deals with the lowering of the melting points of silicate fusions (comparable to lowering of freezing point in solutions by dissolved substance). A general study of slag and slag minerals from the point of view of the physical chemist.

67. **Wallace, R. C.** Ueber die binären System des Natriumsilicats mit Lithium-, Magnesium-, Calcium-, Strontium-, und Barium-metasilicat; des Lithium metasilicats mit Kalium-, Magnesium-, Calcium-, Strontium-, und Barium-metasilicat; und über das Dreistoffsystem Na_2O-Al_2O_3-SiO_2. Ztschr. anorg. Chem., Bd. 63, 1909, pp. 1–48. Presents cooling curves obtained by the thermal method; gives conclusions regarding solubility, eutectics, and different crystal phases, and discusses dissociation, decomposition, reduction, and oxidation of substances occurring in, or related to, coal ash and the formation of clinker.

68. **Baur, E., and Glaessner, A.** Gleichgewichte der Eisenoxyde mit Kohlenoxyd und Kohlensäure. Ztschr. physik. Chem., Bd. 43, 1903, pp. 354–368. Discusses the reduction of Fe_3O_4 to FeO and FeO to Fe by CO. Gives tables and curves of results of study of the following reactions:

$$3FeO+CO_2 \rightleftarrows Fe_3O_4+CO$$
$$Fe+CO_2 \rightleftarrows FeO+CO$$

See pages 53 and 54, this bulletin.

69. **Chaudron, G.** Réactions réversibles de l'eau sur le fer et sur l'oxyde ferreux. Compt. rend., t. 159, 1914, pp. 237–239. Gives two series of equilibrium data corresponding to following reactions:

$$Fe+H_2O \rightleftarrows FeO+H_2$$
$$3FeO+H_2O \rightleftarrows Fe_3O_4+H_2$$

See also pages 44 to 46 of this bulletin.

70. **Deville, H. St. Claire.** Action de l'eau sur le fer et de l'hydrogéne sur l'oxyde de fer. Compt. rend., t. 70, 1870, pp. 1105–1111 and 1201–1207. Presents equilibrium data corresponding to reaction:

$$3Fe+4H_2O \rightleftarrows Fe_3O_4+4H_2$$

71. **Falcke, V.** Die Reaktionen zwischen Eisenoxydul und Kohle und zwischen Kohlenoxyd und Eisen. Ber. deut. chem. Gesell., Bd. 46, 1913, pp. 743–750. States that addition of amorphous carbon has no effect on the pressure produced when FeO (containing 0.5 per cent C) and Fe are heated together at 530° to 660° C., and that the reaction between FeO and CO is different when the CO is passed over the FeO than when the two are heated together in a closed vessel; in the reaction first mentioned much carbon is deposited; in the second reaction no carbon is deposited.

72. ———. Die Reaktionen zwischen Eisenoxydul und Kohle und zwischen Kohlenoxyd und Eisen. Ztschr. Electrochem., Bd. 22, 1916, pp. 121–133; abstracted in Jour. Chem. Soc., vol. 110, pt. 2, 1916, p. 484. Compares results obtained by author with those of Schenck.

73. **Glaser, Ferdinand.** Ueber Reduktion von Metalloxyden im Wasserstoffstrom. Ztschr. anorg. Chem., Bd. 36, 1903, pp. 1–35. Treats of reduction of various metallic oxides (including Fe_2O_3) in current of hydrogen.

74. **Hilpert, S.** Ueber die Reduktion von Eisenoxyd durch Wasserstoff und Kohlenoxyd. Ber. deut. chem. Gesell., Bd. 42, 1909, pp. 4575–4581. Temperatures are given for the beginning of reduction of iron oxide prepared by various methods. States that FeO and Fe_3O_4 in a pure form could not be obtained, each sample showing at once all the stages of reduction. On reducing with CO the products were contaminated with C, seemingly in the form of cementite.

75. ———. Ueber die Sauerstoff-Abgabe des Eisenoxyd, bei hohen Temperaturen. Ber. deut. chem. Gesell., Bd. 42, 1909, pp. 4893–4895. States that results varied according to source and previous treatment of oxide. The reaction took place slowly.

76. **Hilpert, S., and Beyer, J.** Ueber Eisenoxyduloxyde und Eisenoxydul. Ber. deut. chem. Gesell., Bd. 44, 1911, pp. 1608–1619. Describes a method for preparing pure Fe_3O_4 by heating Fe_2O_3 in H_2 containing a definite amount of water vapor. Attempts to prepare FeO by increasing the amount of water vapor and raising the temperature always gave some Fe, and at 1,100° C. the product still contained 1.5 per cent Fe_3O_4. A method was devised for making comparative measurements of the magnetic properties of the products in powdered form. The presence of solid solutions with Fe_3O_4 was shown.

77. **Hofman, H. O., and Mostowitsch, W.** The behavior of calcium sulphate at elevated temperatures with some fluxes. Trans. Am. Inst. Min. Eng., vol. 39, 1908, pp. 628–653. Describes experiments made in platinum-foil furnace, in pure, dry air with fluxes, SiO_2, Fe_2O_3, and PbO (separately) with $CaSO_4$. Hydrous $CaSO_4$ loses its combined water at 900° C.; anhydrous $CaSO_4$ is unchanged to 1,200° C., where dissociation begins, with separation of SO_2 and O_2, the rate and degree of dissociation depending on time and temperature. $CaSO_4$ fuses at 1,360° C., being somewhat decomposed. SiO_2 decomposes $CaSO_4$, decomposition beginning at 1,000° C. and finishing at 1,250° C., products being CaO, SiO_2, and gases. It appears that the formation temperature of calcium silicate lies below melting point. Pure Fe_2O_3 begins to decom-

pose $CaSO_4$ at 1,100° C., decomposition ending at 1,250° C. A higher temperature is required than with SiO_2, but the action is more rapid. The decomposition product is $CaO.Fe_2O_3$ (calcium ferrite), which fuses at 1,250° C. This product can dissolve and decompose $CaSO_4$.

78. ———. The behavior of calcium sulphate at elevated temperatures with some fluxes. Trans. Am. Inst. Min. Eng., vol. 40, 1909, pp. 807–808. Authors found that Fe_2O_3 remained unchanged at 1,360° C. in dry air, but at 1,375° C. it lost O_2 (4.4 per cent), became magnetic, and contained some FeO.

79. HOFMAN, H. O., AND WANJUKOW, W. The decomposition of metallic sulphates at elevated temperatures in a current of dry air. Trans. Am. Inst. Min. Eng., vol. 43, 1912, pp. 523–577. Heated in current of pure, dry air, $FeSO_4$ fuses to $Fe_2O_3.2SO_3$, which dissociates into Fe_2O_3 and SO_3, noticeably at 492° C., and markedly at 550° to 560° C. $Al_2(SO_4)_3$ begins to dissociate at 590° C., disassociation becoming marked at 639° C. Dissociation of $MgSO_4$ begins at 890° C.

80. HOSTETTER, J. C., AND SOSMAN, R. B. The dissociation of ferric oxide in air. Jour. Am. Chem. Soc., vol. 38, 1916, pp. 1188–1198. Describes experiments that showed a measureable dissociation of Fe_2O_3 in air at all temperatures between 1,100° and 1,300° C. and indicated that dissociation increases with increase of temperature.

81. JOHNSON, W. MCA. The reducibility of metallic oxides as affected by heat treatment. Trans. Am. Inst. Min. Eng., vol. 47, 1914, pp. 219–231. Discusses H. O. Hofman's experiments to determine comparative reducing effects of CH_4, CO, illuminating gas, and producer gas on hard hematite. The tests showed that, irrespective of the reversibility of the reaction, the progress of the reduction was not simply a function of the time and the temperature.

82. JOHNSTON, JOHN. The thermal dissociation of $CaCo_3$. Jour. Am. Chem. Soc., vol. 32, 1910, pp. 938–946. States that dissociation begins at 587° C. and is complete at 898° C., at atmospheric pressure.

83. KINNISON, C. S. A note on the reduction of Fe_2O_3. Trans. Am. Cer. Soc., vol. 16, 1914, pp. 136–143. The reaction $3Fe_2O_3 \rightleftarrows 2Fe_3O_4 + O_2$ in clays was studied by adding 5 per cent Fe_2O_3 in the form of $Fe_2(OH)_6$ to kaolin and various mixtures of SiO_2 and kaolin and heating the mixture until a black color developed. Author concludes that (1) the smaller the Al_2O_3–SiO_2 ratio the lower the temperature of formation of "ferrous silicate"; (2) at comparatively low temperatures fluxes augment the coloring power of iron; and (3) in mixtures with large amounts of fluxes, "reduction is associated with solution." The dissociation temperature is about 1,210° to 1,220° C. Fluxes contribute to lowering of the transition point.

84. KREMANN, R. K. The application of physico-chemical theory to technical processes and manufacturing methods. London, 1913, pp. 115–131. Gives equilibrium data on the systems of Fe and its oxides and of C and its oxides.

85. MATHESIUS, LUDWIG. Untersuchungen über die Reduzierbarkeit von Eisenerzen in strömenden Gasen. Stahl und Eisen, Bd. 34, 1914, pp. 866–872. Describes method for determination of Fe, Fe″, Fe‴. Ores were heated at 600°, 700°, and 900° C. in a current of illuminating gas. Concludes that true equilibrium is not reached in a moving gas stream and that the degree of reduction is largely dependent on the nature of the ore and the exposed surface. Reduction is difficult in a sintered product.

86. **Preuner, G.** Das Gleichgewicht zwischen Eisen, Eisenoxyduloxyd, Wasserstoff und Wasserdampf. Ztschr. phys. Chem., Bd. 47, 1904, pp. 385–417. Gives equilibrium data for system Fe, Fe_3O_4, H_2, and H_2O. (See pp. 44–46 of this bulletin.)

87. **Rhead, T. F. E., and Wheeler, R. V.** The effect of temperature on the equilibrium $2CO \rightleftarrows CO_2+C$. Jour. Chem. Soc., vol. 97, 1910, pp. 2178–2179. Wood charcoal was heated 48 to 240 hours at temperatures of 850° to 1,200° C., and the equilibrium constant K was calculated at the various temperatures from the analysis of the resulting gases.

88. ——— The effect of temperature and of pressure on the equilibrium $2CO \rightleftarrows CO_2+C$. Jour. Chem. Soc., vol. 99, 1911, pp. 1140–1153. Tables are given showing the composition of the resulting gases at temperatures of 800° to 1,100° C. and pressures of 1.11 to 1.33 atmospheres.

89. ——— The rate of reduction of carbon dioxide by carbon. Jour. Chem. Soc., vol. 101, 1912, pp. 831–845. Critical discussion of results of other investigators. Concludes that in manufacture of producer gas temperatures should be 1,100° C. or more to insure reduction of CO_2 to CO.

90. **Schenck, Rudolph.** Physikalische Chemie der Metalle. Halle, 1909, 193 pp. Pages 112 to 136 are on oxidation and reduction; pages 137 to 165 treat of blast-furnace gases and equilibrium of various systems, such as CO_2, CO, C, Fe, Fe_2O_3, etc.

91. **Sosman, R. B.** Some problems of the oxides of iron. Jour. Wash. Acad. Sci., vol. 7, 1917, pp. 55–72. Reviews present knowledge regarding the oxides of iron relative to the phase rule. Discusses the relation of iron oxides to certain geological problems.

92. **Sosman, R. B., and Hostetter, J. C.** The oxides of iron. 1. Solid solutions in the system Fe_2O_3–Fe_3O_4. Jour. Am. Chem. Soc., vol. 38, 1916, pp. 807–833. Reports investigation undertaken as basis for study of iron-bearing silicates at high temperatures, giving results of measurements of dissociation pressures of the iron oxides made in vacuum furnace.

93. ———. The reduction of iron oxides by platinum, with a note on the susceptibility of iron-bearing platinum. Jour. Wash. Acad. Sci., vol. 5, 1915, pp. 293–303. States that platinum acts on hematite and magnetite at 1,200° C. under low O_2 pressures, absorbing Fe and giving off O_2; reacts in the same way with magnetite at 1,600° C. and O_2 at atmospheric pressure.

94. **Stoffel, A.** Ueber die Reaktion zwischen Kohlenoxyd und Eisen. Ztschr. anorg. Chem., Bd. 84, 1914, pp. 56–76. Reports results of a study of velocity of formation of $Fe(CO_5)$ from Fe and CO and effect of various factors on the velocity. When CO was passed over Fe powder at a pressure of 1 atmosphere the reaction was at first rapid, becoming slower as the amount of CO absorbed increased, but still continuing after one week. Decomposition occurred above 60° to 70° C. and was complete at 100° C.

95. **Walden, P. T.** On the dissociation pressure of ferric oxide. Jour. Am. Chem. Soc., vol. 30, 1908, p. 1350. Author determined the pressure curve corresponding to the reaction $3Fe_2O_3 \rightleftarrows 2Fe_3O_4+O$. At about 1,350° C. the pressure was found to be 160 mm.

96. **Wartenberg, H. von.** Ueber die Reduktion der Kieselsäure. Ztschr. anorg. Chem., Bd. 79, 1912, pp. 71–87. States that partial pressure of Si vapor, assuming that it is monatomic, is 3×10^{-6} at 1,315° C., and that SiO_2 is reduced and sublimes in H_2 and also in CO at 1,350° to 1,400° C.

MELTING AND SOFTENING TEMPERATURES OF OXIDES, SULPHIDES, SILICATES, MIXTURES, AND SLAGS.

97. Akermann, R. Ueber die Bildungstemperaturen der Hochofenschlacken. Stahl und Eisen, Jahrg. 10, 1890, pp. 424–425; abstracted in Jour. Iron and Steel Inst., 1890, pt. 2, p. 760. Discusses Gredt's determinations of the formation temperatures of slags and compares his results with those obtained by the calorimetric method. The latter method gave lower results. Gives the specific heat of blast-furnace slags as between 0.29 and 0.33.

98. Becker, E. Ueber das Zustandsdiagramm Schwefel-Eisen und den durch Schwefel hervorgurfen Rotbruch. Stahl und Eisen, Bd. 32, 1912, pp. 1017–1021; abstracted in Jour. Iron and Steel Inst., vol. 86, pt. 2, 1912, pp. 604–605. Deals with equilibrium of the system iron-sulphur; gives eutectic mixture as 84.6 per cent iron sulphide and 15.4 per cent iron, melting at 985°.

99. Bishof, Carl. Relative Schmelzbarkeit der Silikate des Eisens, des Kalks, der Magnesia und der Thonerde. Dingl. poly. Jour., Bd. 165, 1862, pp. 378–381. Gives relative fusibilities of iron and lime silicates.

100. Bleininger, A. V., and Brown, G. H. The testing of clay refractories, with special reference to their load-carrying capacities at furnace temperatures. Tech. Paper No. 7, U. S. Bureau of Standards, 1911, 78 pp. Discusses Seger cones and their uses; influence of viscosity; and effects of various constituents of fire clays upon the softening temperatures. *See also* No. 42.

101. Boudouard, O. Experiments on the fusibility of slags. Jour. Iron and Steel Inst., vol. 67, 1905, pp. 339–382. Fletcher gas furnace was used for temperatures up to 1,450° C., and Deville furnace for higher temperatures. Gives softening temperatures of silicates of Al_2O_3 and of CaO; of aluminates of CaO; and of aluminocalcic silicates. Results are represented by triaxial diagrams. Also gives softening temperatures of a number of industrial slags.

102. Cobb, J. W. The synthesis of a glaze, glass, or other complex silicate. Jour. Soc. Chem. Ind., vol. 29, 1910, pp. 69–74, 250–259, 333–336, 399–404, 608–614, 799–802. Experiments were performed in laboratory muffle furnace and in large works furnace. Draws conclusions on progress of slag formation by determining ingredients that were (1) "quickly soluble" (SiO_2, CaO, etc., extracted by N–HCl in one hour), (2) "slowly soluble" (successive treatment with N–HCl, 24-hour periods, until only trace resulted) and (3) "silica soluble in Na_2CO_3" (SiO_2 extracted by hot 5 per cent Na_2CO_3 in 15 minutes). Describes physical condition or degree of progress toward fusion. Gives results with two samples of broken glassware.

103. Day, A. L., and Sosman, R. B. The melting points of minerals in the light of recent investigations on the gas thermometer. Am. Jour. Sci., vol. 31, 1911, pp. 341–349. Gives a series of melting points, with bibliography.

104. Dittler, Emil. The melting points of potassium-sodium feldspars. Tech. Min. Petr. Mittl., Bd. 31, 1914, pp. 513–522; Keram. Rundschau., Bd. 21, 1914, pp. 390–391; Chem. Abs., vol. 8, 1914, p. 222. Presents table showing chemical analysis and mineralogical constitution of feldspars used and their fusion temperatures. Melting points of albite silicates lowered by addition of orthoclase silicates up to 40 to 43 per cent of latter; with larger proportion the melting point is raised.

105. **Flach, G.** Ueber die Wirkung von Metalloxyden auf den Schmelzpunkte von Quarz-Zettlitzer-Kaolin-Mischungen. Sprechsaal. Jahrg. 44, 1911, pp. 171–173, 187–189, 205–207, 219–220, and 626. Gives tables showing effects of various oxides, among them CaO, MgO, Na_2O, K_2O, FeO, on mixtures of quartz and Zettlitz kaolin. Temperature measurements expressed in terms of Seger cones. Revision of works of Richter, Bishof, Seger, and others.

106. **French, A. T.** Some analyses of copper blast furnace slags and determinations of their melting points. Mining Sci., vol. 61, 1910, pp. 133–134. Used muffle furnace and Seger cones; did not consider temperature even throughout muffle. Results, for a series of 10 slags, show that slag of highest melting point contained most Al_2O_3 and that slag of lowest melting point contained least Al_2O_3.

107. **Fulton, C. H.** The constitution and melting points of a series of copper slags. Trans. Am. Inst. Min. Eng., vol. 44, 1912, pp. 751–780. Used a carbon-resistance furnace and, by thermal method, determined melting points of a number of copper slags. Concludes that slags of lowest melting points lie between 1.4 and 1.65, silicate degree.

108. **Gredt, Paul.** Ueber die Bildungs-temperaturen der Hochofenschlacken. Stahl und Eisen, Bd. 9, 1889, pp. 756–759. Gives formation temperatures of a number of synthetic blast-furnace slags, composed of SiO_2, Al_2O_3, and CaO, and SiO_2, Al_2O_3, CaO, and MgO. Expresses temperatures in terms of Seger cones.

109. **Hofman, H. O.** Some experiments determining the refractoriness of fire clays. Trans. Am. Inst. Min. Eng., vol. 24, 1894, pp. 42–66. He states that Seger cones are not true pyrometers, as the change from a mixture to a compound is not a function of temperature alone, but depends also on the manner of firing, the time of heating, and the form of furnace.

110. ———. The temperatures at which certain ferrous and calcic silicates are formed in fusion, and the effect upon these temperatures of the presence of certain metallic oxides. Trans. Am. Inst. Min. Eng., vol. 29, 1899, pp. 682–721. Describes softening-temperature determinations made in a gas furnace having a reducing atmosphere, the slag mixture being molded into small cones, $\frac{25}{32}$ by $\frac{3}{8}$ inch, and the temperatures being measured with Seger cones. Gives table showing effect on softening point of replacing SiO_2 with Al_2O_3 (temperature increases with increase of Al_2O_3); also table showing effect of replacing FeO and CaO with Al_2O_3.

111. **Hofman, H. O., and Stoughton, B.** Does the size of the particle have any influence in determining the resistance of fire clays to heat and to fluxes? Trans. Am. Inst. Min. Eng., vol. 28, 1898, pp. 440–444. Concludes that, as tables show only slight difference in temperature of fusion between pulverized and rough samples, the laboratory method of grinding to a powder will give results directly comparable with those obtained in practice on larger and coarser samples.

112. **Howe, H. M.** Preliminary note on the thermal properties of slags. Trans. Am. Inst. Min. Eng., vol. 18, 1890, pp. 724–747. Used a gas furnace and calorimetric method of determining temperature. Gives softening temperatures of various CaO, MgO, and SiO_2 mixtures, both with and without previous heating to higher temperature. Gives melting point of $CaO.SiO_2$ as 1,381° C. Thinks that formation temperatures are higher than melting temperatures.

113. ———. Metallurgical laboratory notes. Boston Testing Laboratories, Boston, 1902, pp. 46–69. Discusses melting points of silicates. Uses cone method, making pyramid with ¾-inch base and 1½ inches high, of 100-mesh material, with a binder of 10 to 15 per cent vaseline mixed with heavy lubricating oil.

114. Johnson, W. McA. A new apparatus for determining the melting points of slags. Jour. Chem. Met. Min. Soc. South Africa, vol. 7, 1906–7, p. 98. Slag allowed to drop through hole in cylinder of Acheson graphite.

115. Kanolt, C. W. Melting points of fire bricks. Tech. Paper No. 10, U. S. Bureau of Standards, 1912, 17 pp. Gives melting points of 54 samples of fire bricks, including fire clay, bauxite, silica, magnesia, and chromite brick, determined in Arsem vacuum furnace with Morse optical pyrometer. Melting points were as follows: Kaolin, 1,740° C.; pure Al_2O_3, 2,010° C.; pure SiO_2, 1,750° C.; bauxite, 1,820° C.; bauxite clay, 1,795° C.; and chromite, 2,180° C.

116. ———. Melting points of some refractory oxides. United States Bureau of Standards Bull., vol. 10, No. 2, 212, 1914, pp. 295–314. With same methods as described in Technologic Paper 10, author determined following melting points: MgO, 2,800° C.; CaO, 2,572° C.; Al_2O_3, 2,050° C.; Cr_2O_3, 1,990° C.; and Pt. 1,755° C.

117. Kultascheff, N. V. Ueber die Schmelzpunkte von Calciumsilikat ($CaSiO_3$), Natriumsilikat (Na_2SiO_3) und ihren Mischungen. Ztschr. anorg. Chem. Bd. 35, 1903, pp. 187–193. Gives tables of melting points, from pure Na_2SiO_3 at 1,007° C. to mixture of 15 per cent Na_2SiO_3 and 85 per cent $CaSiO_3$ at 1,150° C. (Melting pure $CaSiO_3$ above 1,400° C). Used thermal method in getting freezing-point curve.

118. Leitmeier, H. von. Zur Kenntnis der Schmelzpunkte von Silikaten. Der Einfluss der Korngrösse auf den Schmelzpunkte. Bestimmung des Schmelzpunktes einiger Silikate durch längeres Erhitzen. Ztschr. anorg. Chem. Bd. 81, 1913, pp. 209–232. Used Heraeus platinum furnace, with thermocouple. Gives sintering and melting points for series of natural minerals, showing influence of size of particles. Difference for coarse (3 to 4 mm. diameter) and fine (40 mm. in diameter) particles ranged up to 70° C. Leitmeier's melting temperatures are really softening temperatures, that is, none represent the equilibrium point of solid liquid.

119. Loebe, R., and Becker, E. Das System Eisen-Schwefeleisen. Ztschr. anorg. Chem., Bd. 77, 1912, pp. 301–319. The two components form no mixed crystals, though Fe may be slightly soluble in FeS. Melting point of FeS is 1,198° C.

120. Mathesius, W. Die Entstehung der Schlacken in hüttenmännischen Prozessen; die Konstitution der Schlacken, ihre industrielle Verwertung. Stahl und Eisen, Bd. 24, 1904, pp. 1000–1007. Discusses origin and utilization of slags; does not give formation or melting points or results of original work.

121. Nacken, R. Vergleich der optischen und der thermischen Methode zur Bestimmung von Schmelz-temperaturen. (Compares optical and thermal methods for the determination of melting points.) Centralbl. Min. Geol. 1913, pp. 328–337; Chem. Abs. vol. 7, 1913, p. 3097. Shows discrepancies existing in values given in literature. Author states that by great care he obtained optical results that agree well with those obtained by thermal method.

122. ———. Ueber die Bestimmung der Schmelz-temperaturen von Silikaten auf optischen Wege (determination of fusion temperature of silicates by an optical method). Silikat Ztschr., Jahrg. 1, 1913, pp. 65–67; Chem. Abs., vol. 7, 1913, p. 4053. Substance was heated in platinum-resistance furnace, and observations were made through polarizing microscope; melting point is given as temperature at which interference colors disappear.

123. Rieke, R. Das Richters'sche Gesetz. Ein Beitrag zur Kenntnis der Flussmittelwirkung. Sprechsaal, Jahrg. 43, 1910, pp. 198–199, 214–216, 229–232. Discusses an extensive investigation of the fluxing action of the basic oxides, used in the ceramic industries, on Zettlitz kaolin and mixtures of quartz and kaolin. The raw materials, as kaolin, quartz, limestone, iron oxide, etc., were ground wet in a ball mill. Some of the mixtures were ground dry, ignited in a kiln, and again ground. The softening points of the mixtures were determined in a carbon resistance furnace by comparison with standard pyrometric cones.

124. ———. Ueber die Schmelz-barkeit von Kalk-Tonerde-Kieselsäure Mischungen. Stahl und Eisen, Bd. 28, 1908, pp. 16–19; Abs. Jour. Soc. Chem. Ind., vol. 27, 1908, p. 123. Discusses fusibility of lime-alumina-silica mixtures. The mixture was molded into small cones, and the softening points were measured by Seger cones. Mixtures with more than 50 per cent CaO crumbled into powder when cooled slowly.

125. Ruff, Otto. Arbeiten im Gebiet hoher Temperaturen: Ueber das Schmelzen und Verdampfen unserer feuerbeständigsten Oxyde im elektrischen Vakuumofen. Ztschr. anorg. Chem., Bd. 82, 1913, pp. 373–400. Further work on material mentioned in No. 126 following.

126. ———. Ueber einen elektrischen Vakuumofen. Ber. Deut. Chem. Gesell., Bd. 43, 1910, pp. 1564–1574. Describes carbon-tube-resistance vacuum furnace and its uses in melting and vaporizing oxides. Gives melting points, as follows: Pt, 1,745° to 1,755° C.; kaolin, 1,912° C.; and Al_2O_3(in N_2), 2,065° C.

127. Ruff, O., and Goeke, O. Ueber das Schmelzen und Verdampfen unserer sogenannten hochfeuerfesten Stoffe. Ztschr. angew. Chem., Bd. 24, 1911, pp. 1459–1465. Discusses investigations on the melting and vaporization of highly refractory materials. An electric vacuum furnace and a Wanner pyrometer were used. Gives melting point of Al_2O_3 at 2,020° C.

128. Seger, H. A. Collected writings; translated by members of the American Ceramic Society and edited by A. V. Bleininger. Easton, Pa., 1902, 2 vols., 1157 pp. Pages 519 to 520 are on "The effect of titanic acid on the fusibility of fire clays"; author considers that TiO_2 has greater fluxing power than SiO_2. Pages 1005 to 1027 give data on use of Seger cones.

129. Smolensky, S. Schmelzversuche mit Bisilikaten und Titanaten. Ztschr. anorg. Chem., Bd. 73, 1911–12, pp. 293–303. States that thermal method was used to determine melting points of various mixtures of $CaSiO_3$ and $CaTiO_3$, $BaSiO_3$, and $BaTiO_3$, and $MnSiO_3$ and $MnTiO_3$. A kryptol resistance furnace and a Pt–PtRh thermocouple were used. In general, the addition of TiO_2 lowered the melting point.

130. Sprechsaal Kalender. Schmelzpunkte. Jahrg. 8, Coburg, 1916. Gives melting points of oxides, silicates, and various mixtures.

131. STEFFE, HERMAN. Ueber die Bildungstemperaturen einiger Eisenoxydul-Kalkschlacken und einiger kalkfreien Eisenoxydul-schlacken, deren Kenntniss für das Verschmelzen der Bleierze Bedeutung hat. Dissertation. Berlin, 1906. Gives formation temperatures of some ferro-calcic and ferrous slags. Observations made of sintering temperature of mixture when heated in a platinum boat in a current of nitrogen. Temperatures were measured with thermocouple.

132. STEIN, G. Ueber die Darstellung einiger Silikate. Ztschr. anorg. Chem., Bd. 55, 1907, pp. 159–174. The optical method, with Wanner pyrometer, was used for determining melting and inversion points of various silicates (Al_2O_3, FeO, etc.). Discusses formation of Ca, Sr, and Ba carbides in the presence of SiO_2.

133. TREITSCHKE, W., AND TAMMANN, G. Ueber das Zustands-diagramm von Eisen und Schwefel. Ztschr. anorg. Chem., Bd. 49, 1906, pp. 320–335. Determines cooling curve by thermal method, obtaining two "stop points"—one at 970° C. and one at 1,400° C.

134. WHITE, W. P. Melting-point determinations. Am. Jour. Sci., vol. 28, 1909, pp. 453–473. Gives theoretical and mathematical considerations regarding the thermal method for determining melting points. Cites reasons for the variations that have been observed and mentions precautions.

135. ———. Melting-point methods at high temperatures. Am. Jour. Sci., vol. 28, 1909, pp. 474–489. Describes furnace and thermo elements used, giving best methods for making melting-point determinations up to 1,600°.

136. ANONYMOUS. Formation temperatures of ferrous oxide-lime-silica slags. Trans. Am. Inst. Min. Eng., vol. 31, 1901, p. 862; curves plotted from Hofman's paper. Trans. Am. Inst. Min. Eng., vol. 29, 1899, pp. 682–721, showing effect on fusion temperature of replacing CaO by MgO and slag constituents by Al_2O_3. *See also* No. 63.

137. ———. Schmelzpunkte von Oxygen, Silikaten, Boraten und Aluminaten und deren eutektischen Gemischen. Sprechsaal, Jahrg., 44, 1911, pp. 710–713, 729–731, and 744–746. Presents, from various sources, a tabulation of results conveniently arranged.

PYROMETRIC CONES; THEIR USE IN MEASURING TEMPERATURE.

138. AMERICAN CERAMIC SOCIETY. Report of committee on Seger cones. Trans., vol. 3, 1901, pp. 180–192. The committee was appointed to investigate reasons for unsatisfactory behavior of low-melting cones, mainly those containing Fe_2O_3. Suggests that boric acid, affected by S in the gas, and perhaps $CaCo_3$, changed to $CaSO_4$ by S in gas may be the cause of trouble.

139. ASHLEY, H. F. A comparison of the various makes of Seger cones on the market. Trans. Am. Cer. Soc., vol. 8, 1906, pp. 159–167. Compares German, English, and various American makes of cones in 48-hour burns in ceramic kilns.

140. BROWN, G. H., AND MURRAY, G. A. The function of time in the vitrification of clays. Tech. Paper No. 17, U. S. Bureau of Standards, 1913, 26 pp. Shows that rate of heating has decided bearing on the softening points of Seger cones and, in general, that the effect of time in firing of clay products is important.

141. Geigsbeck, S. The fusing points of Seger cones expressed in degrees. Trans. Am. Cer. Soc., vol. 6, 1904, pp. 94–108. Discusses the advisability of expressing the melting points of cones in degrees. Orton and others opposed to such procedure.

142. ———. The melting points of pyrometric cones under various conditions. Trans. Am. Cer. Soc., vol. 14, 1912, pp. 849–871. Concludes that rate of heating has much to do with melting points of Seger cones and that they can not be used for measuring temperature.

143. Gorton E. E. Remarks on Seger cones. Trans. Am. Cer. Soc., vol. 2, 1900, pp. 60–73. General discussion of application of Seger cones to ceramic industry, with especial reference to variable indications of the low-melting cones containing Fe_2O_3.

144. Hoffman, Dr. G. Prüfung der Segerkegel. Tonind. Ztg., Bd. 33, 1909, pp. 1577–1580. Measured softening points of cones 26 to 35 in an iridium tube furnace using Ir-Ir-Ru thermocouple. His results are lower than estimated melting temperatures by 5° to 60° C.

145. ———. Prüfung der Segerkegel. Tonid. Ztg., Bd. 35, 1911, pp. 1099–1100; Sprechsal, Bd. 44, 1911, pp. 143–145. Continuation of work described in 144, in which author compares softening temperatures as obtained in iridium furnace with optical pyrometer or thermocouple, in the laboratory of the Reichsanstalt, with average values obtained in industrial ceramic kilns. The fusion temperatures are 35° to 95° C. lower in ceramic kilns than the accepted Seger cone figures, and 40° to 110° C. lower than results obtained in iridium furnace.

146. Rieke, R. Die Schmelzpunkte der Segerkegel 022–15. Sprechsaal, Bd. 44, 1911, pp. 726–729, 741–744; Tonind. Ztg., Bd. 35, 1911, pp. 1751–1757. Although some cones fused at a temperature lower than accepted figures, on slow heating other cones showed opposite behavior. All cones fused at a lower temperature when heated slowly than with rapid heating; differences were as great as 60° to 100° C.

147. Simonis, M. Segerkegel 022 bis 7. Tonind. Ztg., Bd. 32, 1908, pp. 1764–1768. The object of the investigation described in this article was to overcome troubles encountered with cones that fuse at low temperatures. Cones tested were free from lead.

148. Sosman, R. B. The physical chemistry of Seger cones. Trans. Am. Cer. Soc., vol. 15, 1913, pp. 482–498. Shows the influence of eutectics on the deformation temperatures of Seger cones. States that the existance and influence of these eutectics makes temperature measurements by cones uncertain.

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PYROMETERS AND THE MEASUREMENT OF HIGH TEMPERATURE.

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of solid and liquid copper was measured with thermocouple and various optical pyrometers.

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LABORATORY FURNACES FOR HIGH-TEMPERATURE WORK.

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167. Day, A. L., and Allen, E. T. Temperature measurements up to 1,600° C. Phys. Review, vol. 19, 1904, pp. 177–186. Describes platinum-wire (internal winding) vertical furnace for melting-point determinations.

168. Field, A. L. An electrical-contact vapor-pressure thermoregulator. Jour. Am. Chem. Soc., vol. 36, 1914, pp. 72–76. Describes U-tube, saturated-vapor type of thermoregulator, which is used in connection with molybdenum-wire furnace described in this bulletin on pp. 46 to 49.

169. Frederich, K. Über einiger neuere Formen von elektrisch geheizten Laboratoriumsöfen (several new forms of laboratory furnaces). Metallurgie, Jahrg. 4, 1907, pp. 778–781; Chem. Abs., vol. 2, 1908, p. 1084. Describes platinum-foil and kryptol furnaces for high-temperature work.

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172. Heraeus, W. C. Ueber elektrische Laboratoriumsöfen. Ztschr. Elektrochem., Bd. 18, 1912, p. 143. Comparison of efficiency of Pt and Ni wound furnaces.

173. Hoffman, G. Prüfung der Segerkegel. Sprechsaal, Bd. 44, 1911, pp. 143–145. Describes an iridium tube furnace.

174. Jeffries, Zay. Notes on the granular electric furnace. Met. and Chem. Eng., vol. 12, 1914, pp. 154–157. Describes a convenient type of carbon-resistance furnace, with a resistor of kryptol, granular carbon or graphite, and alundum refractories. Approximately 3 kilowatts is required to attain temperature of 1,600° C.

175. Le Blanc, M. Widerstandofen mit elektrisch geheiztem Nickeldraht. Ztschr. Elektrochem., Bd. 15, 1909, pp. 683–687. Describes furnace of 2-mm. nickel wire wound on Berlin porcelain, used up to 1,300° C. maximum.

176. Malm, W. R. A high-temperature experimental furnace. Met. and Chem. Eng., vol. 13, 1915, p. 70. Describes a crucible-type arc furnace especially adapted for fusion of ferroalloys. Takes 30 to 50 amperes, giving ½-inch arc.

177. Northrup, E. F. A new high-temperature electric furnace. Met. and Chem. Eng., vol. 12, 1914, pp. 31–33. Describes a vertical type furnace, with Acheson graphite heating element, the whole inclosed in Monel-

metal case, with a heating space 12 inches long by 1¼ inches in diameter. Maximum capacity, 170 amperes at 23 volts. Used for study of properties of metals at high temperatures.

178. ———. Cascade attachment for graphite furnace. Met. Chem. Eng., vol. 12, 1914, pp. 305–306. Discusses a supplementary device for furnace described in No. 177 to raise temperature to melting point of tungsten or higher. Maximum current consumption of 9.5 kilowatts.

179. Pip, W. Zwei neue Laboratoriumsöfen für hohe Temperaturen. Ztschr. Elektrochem., Bd., 16, 1910, pp. 664–667. Describes an oven constructed of MgO and heated by gas, and an oven of the carbon-resistance type.

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182. Tucker. S. A. Electric tube furnace for temperature measurements. Trans. Am. Electrochem. Soc., vol. 11, 1907, pp. 303–306. Describes a carbon-tube furnace with resistor 540 mm. long; part in center turned down thin to increase resistance; about 1 inch in inside diameter; ¼-inch walls in thick part. Tube surrounded by petroleum coke; whole incased in brick; water-cooled brass electrodes soldered on tube. About 15 volts required to attain a temperature of 3,000° C. A maximum current of 850 amperes is required.

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184. ———. Platinum resistance furnace for melting points and combustions. Jour. Am. Chem. Soc., vol. 29, 1907, pp. 1442–1444. Register is Pt. tape wound on quartz tube, used only for temperatures up to 1,200° C.

185. Tucker, S. A., and Moody, H. R. Improved electric furnace for laboratory use. Jour. Am. Chem. Soc:, vol. 23, 1901, pp 473–476. Describes an electric-arc furnace, Moisson type, with capacity of 8½ to 12½ kilowatts at 70 to 90 volts. Either water-jacketed or copper-plated electrodes.

186. Ubbelohde, L. Electrische Laboratoriumsöfen mit Wicklung aus unedlen Metalle. Ztschr. Electrochemie, Bd. 17, 1911, pp. 1002–1003. Compares Ni and Pt wound furnaces for low-temperature work.

187. Winn, R., and Dantsizen, C. Small electric furnace with heating element of ductile tungsten or ductile molybdenum. Jour. Ind. and Eng. Chem., vol. 3, 1911, pp. 770–771. Discusses construction of two types—vertical, crucible type, and horizontal, tube type, each with resistor of ductile tungsten or molybdenum wire protected from oxidation by an atmosphere of H_2

See also Nos. 13, 46, 62, 126.

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INDEX.

A. Page.

- Alabama, coal from, ash analyses of ... 27
- Alaska, coal from, ash analyses of ... 28
- Allison, V. C., acknowledgment to ... 118
- Alumina, melting point of ... 15
- American bed, Ala., coal from, ash analysis of 27
- Anderson bed, Oreg., coal from, ash analysis of ... 26
- Arkansas, coal from, ash analyses of ... 28
- Ash, alumina in ... 25
  - carbon in, effect of on softening temperature of ... 115
    - figure showing ... 70, 71
  - containing Fe_2O_3, softening temperature of ... 68
  - composition of ... 13
  - definition of ... 13
  - deformation point of, definition of ... 30
  - extraneous, nature of ... 14
  - fineness of, effect of on softening temperature of ... 114
  - fusing temperature of, variations in ... 1
  - ignition of in oxygen, furnace for, figure showing ... 102
    - method of ... 101
  - inherent, nature of ... 14
  - intrinsic, nature of ... 13
  - iron in ... 25
    - effect of on fusibility of ... 21
    - reduction of ... 110
  - lime in ... 25
  - method of fusion of ... 10
  - preparation of, for fusibility tests ... 29
  - silica in ... 25
  - softening of, effect of iron content on ... 21
    - factors influencing ... 20
    - importance of data on ... 1
    - in hydrogen and water vapor mixtures ... 50
    - standard method of determining, importance of ... 44
- Ash cones, appearance of in softening, figure showing ... 31
  - critical temperatures in heating of, figure showing ... 108
  - deformed, views of ... 43
  - effect of fineness of ash on fusibility of ... 33
    - of furnace atmospheres on softening temperature of ... 116
    - of inclination on softening temperature of ... 33
    - of position on softening temperature of ... 80
    - of rate of heating on softening temperature of ... 35
    - of rate of heating on softening temperature of ... 56, 115
    - figure showing ... 34, 35, 106
    - of shape of, on softening temperature of ... 103, 104
- Ash cones, effect of size of, on softening temperature of ... 32, 103, 104
    - on melting temperature of, figure showing ... 105
  - fluid temperature of, determination of ... 108
  - from Northrup furnace, appearance of ... 43
  - fused, appearance of, figure showing ... 42, 83, 84, 85
    - carbides in, formation of ... 47
    - iron content of ... 65
      - state of ... 64
    - magnetism of ... 73
  - heating of, standard method for ... 112
  - preparation of, standard method for ... 111
  - softening interval of, definition of ... 109
  - softening points of ... 34, 35
  - softening temperature of, in different atmospheres ... 63, 68
      - figure showing ... 57, 58, 59, 60
  - softening temperature tests of, results of. 61, 82
  - swelling of, causes of ... 86
  - vertical position of, advantages of ... 86
  - *See also* Cones, Seger Cones.
- Atmospheres. *See* Reducing atmospheres.
- Augustine, C. E., cited ... 53, 55

B.

- Bailey, E. G., on determination of fusing temperature of ash ... 4
- Bibliography ... 119-138
- Binder, dextrin, effect of, on fusibility of ash cones ... 36
- Bleininger, A. V., acknowledgment to ... 117
- Bureau of Mines, work of ... 1

C.

- Carbides in fused cones, formation of ... 67
- Carbon, unburned, effect of, on softening of ash ... 87, 115
  - *See also* Ash.
- Carbon dioxide and carbon monoxide atmospheres, difficulty in making tests with ... 73
  - effect of, on melting temperature of ash .. 82, 116
- Carbon dioxide used in tests, preparation of. 55
- Carbon-resistance furnace, reducing atmosphere in ... 39
  - results of tests in ... 32
    - figure showing ... 40
  - *See also* Furnace, carbon-resistance.
- Carbon monoxide, reduction of iron in atmosphere of ... 38, 39
  - used in tests, preparation of ... 55
- Calcite in coal, occurrence of ... 14
- Calkins, W. B., determination of fusing temperature of ash by ... 4
- Carney bed, Wyo., coal from, ash analysis of .. 28

Page.
Chandron, G., on equilibria of iron and its oxides in hydrogen and water vapor 45
China, coal from, ash anlyses of 28
Clement, J. K., cited 55
Clinker, iron content of 65
  view of 66
Clinkering of ash in coal, causes effecting 4, 6, 21, 44
  relation of melting interval to 18
  relation of softening temperatures to 63
Clinker slags, state of iron in 64
Coal, impurities in, float-and-sink method for separation of 13
  silica in, occurrence of 14
Coals tested for ash fusibility, source of 26
  See also States and beds named.
Coal ash, analyses of 87
  mixtures, analyses of 82
  See also Ash.
Coal Creek bed, Tenn., coal from, ash analysis of 27, 28
Cobb, J. W., determination of fusing temperature of ash by 3
Cones, "down point" of, values for 82
  electromagnet for testing, figure showing. 95
  position of, effect on softening temperature of 114
  shape of, effect on softening temperature of 115
  size of, effect on softening temperature of. 114
  See also Ash cones, Seger cones.
Constam, E. J., determination of melting temperature of ash by 6
Crucibles, for standard tests of fusibility, description of 113

D.

Day, A. L., cited 18, 19
  on melting point of minerals 15
Deformation point, definition of 30, 50
Dennis, L. M., cited 55
Depew, H. A., acknowledgment to 118
Deville, S. C. H., cited 44
Dextrin binder, effect of, on fusibility of ash cone 36
"Down point," definition of 81
  values for 82

E.

Electromagnet, for testing fused cones, figure showing 95
England, coal from, ash analysis of 28
Equilibrium data in the system CO, CO_2, Fe, FeO, Fe_3O_4, figure showing 53
Equilibrium diagram of water vapor, hydrogen, iron, and iron oxides, figure showing 46
Eutectic point of lime-silica system, figure showing 17

F.

Feild, A. L., cited 19
Ferric oxide, dissociation temperature of 20
Fieldner, A. C., cited 29, 32

Page.
Flowing interval, definition of 113
Fluid temperature, definition of 112
  determination of 108
Fraser, J. C. W., cited 55
Freeport bed, Pa., coal from ash, analysis of 28
Furnace, carbon-resistance, figure showing 24
  for fusibility tests, requirements of 89
  gas, reduction of iron in ash cones in 38
    tests with, description of 92
      discussion of 110
  molybdenum, figure showing 23
  Northrup, figure showing 25
  silica-tube, figure showing 46, 47
  softening points of ash in, figure showing 38
  See also Carbon-resistance, molybdenum, platinum, Northrup, Meker, melter's, and Heraeus furnace.
Fusiometer, description of 10
Fusion temperature, methods used in determining 30
Fusion point, definition of 30, 50

G.

Gas, artificial, results of melting temperature tests with 98
    analysis of 100
  natural, analysis of 100
Gas furnace, advantages of 89
  atmosphere in, analyses of 93, 94
Goodwin, E. G., acknowledgment to 118
Greiner, E., cited 21
Gypsum in coal, occurrence of 14

H.

Hall, A. E., cited 32
Heraeus furnace, figure showing 22
  saturator for, figure showing 47, 48
Hill, H. H., acknowledgment to 118
Hitchcock, E. A., boiler test of coals by 4
Hubley, F. C., on determination of fusibility of ash 9
Hulett, G. A., acknowledgment to 117
Hydrogen, equilibrium of with water vapor, iron, and iron oxides 44, 45
  fusibility of ash cones in, figure showing 34
  use of, in fusibility tests 49
Hydrogen and water-vapor atmosphere, effect of on softening temperature of ash 116
  preparation of 78

I.

Illinois, coal from, ash analyses of 28
Indiana, coal from, ash analysis of 27
Initial deformation of cones, definition of 112
  values for 82
Intumescence of ash cones, causes of 86
Iron, equilibrium of, with hydrogen and water vapor 45
  in ash, effect of reducing atmospheres on. 116
  in fused ash and in clinker, state of 64, 117
Iron oxides, in ash, effect of on fusibility 20, 78
  reduction of 110

J.

Jellico bed, Tenn., coal from, ash analyses of 27, 28

K.

Page.

- Kaolinite, melting point of.......... 15
- Kanolt, C. W., cited.......... 16
- Kansas, coal from, ash analysis of.......... 28
- Kreisinger, Henry, cited.......... 110
  - on oxygen in fuel bed of coal fires.......... 53

L.

- Le Chatelier, Henry, determination of fusing temperature of ash by.......... 3
- Lime, melting point of.......... 16
  - in ash, percentage of.......... 25
- Lime-silica system, concentration-temperature diagram for, figure showing.......... 17
  - eutectic point of, figure showing.......... 17
- Lower Kittanning, Miller, or B bed, Pa., coal from, ash analyses of.......... 28
- Lower Weir-Pittsburg bed, Kans., coal from, ash analyses of.......... 28

M.

- Magnesia, melting point of.......... 16
  - percentage of in ash.......... 25
- Magnetism of fused ash cones.......... 73
  - method of determining.......... 95
- Magnetite, melting point of.......... 16
- Marks, L. S., cited.......... 1,88
  - determination of fusibility of ash by.......... 6
  - on differences in softening temperatures of ash.......... 37
  - on position of ash cones in fusibility tests.......... 79
  - on use of Seger cones.......... 5
- Mary Lee bed, Ala., coal from, ash analysis of.......... 27
- Maryland, coal from, ash analysis of.......... 27
- Mathesius, Ludwig, cited.......... 64
- Meker furnace, figure showing.......... 22,97
  - results of softening temperature tests in.......... 32,98
- Melter's furnace, advantages of for softening tests.......... 111
  - atmosphere in, analyses of.......... 93,94
  - description of.......... 90
  - figure showing.......... 91
  - results of tests in.......... 96,98,104,107
- Melting interval, determination of.......... 18
- Michigan, coal from, ash analysis of.......... 28
- Miller or "B" bed, Pa., coal from, ash analyses of.......... 28
- Mingo bed, Tenn., coal from, ash analyses of.......... 27
- Molybdenum furnace, advantages of.......... 89
  - ash cones from, appearance of.......... 43
  - effect of hydrogen atmosphere in, figure showing.......... 41
  - experiments with, description of.......... 35
  - modified, description of.......... 74
    - figure, showing.......... 75,76
    - view of.......... 75
  - saturator for, description of.......... 48
  - softening point tests in, results of.......... 32,33,36
    - figure showing.......... 41
  - view of.......... 43
  - *See also* Furnace, molybdenum.
- Mueller, W. A., acknowledgment to.......... 118

N.

Page.

- Natural gas, analysis of.......... 100
  - results of softening temperature tests with.......... 98
    - figure showing.......... 98
- Nickel Plate bed, Ala., coal from, ash analysis of.......... 27
- No. 2 bed, Mich., coal from, ash analysis of.......... 28
- No. 5 bed, Ind., coal from, ash analysis of.......... 27
- No. 6 bed, Ill., coal from, ash analyses of.......... 28
- North Carolina, peat from, ash analysis of.......... 28
- North Dakota, lignite from, ash analysis of.......... 28
- Northrup furnace, ash cones from, appearance of.......... 43
  - nature of reducing atmospheres in.......... 39
  - result of softening point tests in, figure showing.......... 40,41
  - view of.......... 42
  - *See also* Furnace.

O.

- Oregon, coal from, ash analysis of.......... 28
- Osgood, F. D., acknowledgment to.......... 118
- Ovitz, F. K., cited.......... 53
- Oxides. *See* Iron oxides.

P.

- Palmenberg, O. W., cited.......... 88
  - on determination of fusibility of coal ash.......... 11
  - on position of ash cone in fusibility tests.......... 79
- Pennsylvania, coal from, ash analyses of.......... 27,28
- Peters, E. D., on fusibility of ferric oxide in ores.......... 21
- Pittsburgh bed, Md., coal from, ash analysis of.......... 27
- Pittsburgh bed, Pa., coal from, ash analyses of.......... 28
- Pittsburgh bed, W. Va., coal from, ash analyses of.......... 27
- Platinum furnace, ash cones from, appearance of.......... 42
  - experiments with, description of.......... 36
  - result of softening point tests in, figure showing.......... 38,41
  - *See also* Furnace.
- Pocahontas bed, W. Va., coal from, ash analyses of.......... 28
- Pocahontas No. 3 bed, Va., coal from, ash analyses of.......... 27,28
- Preuler, G., cited.......... 44
- Prost, Eugene, determination of fusion temperature of ash by.......... 2
- Pyrite in coal, occurrence of.......... 14
- Pyrometer, optical, use of.......... 31
  - Wanner, use of.......... 32

Q.

- Quartz, melting point of.......... 15

R.

- Rankin, G. A., cited.......... 17
- Rate of heating, effect of, on softening temperature.......... 115
- Red Stone bed, W. Va., coal from, ash analysis of.......... 27

Page.

- Reducing atmosphere, tests with 110
  - effect of, on softening temperature of ash. 116
  - softening temperature of ash cones in ... 63
  - used in tests 49
- Reduction of iron oxide in ash, effect of 21
- Ricketts, E. B., cited 79, 90
  - on determination of fusibility of coal ash. 12

S.

- Saginaw bed, Mich., coal from, ash analysis of 28
- Saturator for hydrogen and water vapor mixtures, description of 48
  - figure showing 47
  - for modified molybdenum furnace, description of 78
  - figure showing 76
- Seger cones, experiments with, description of. 22–43
  - preparation of 29
  - sizes of, used in test 30
  - softening temperatures of 72
  - use of, by Hubley 9
    - by Le Chatelier 3
    - by Marks 5, 7, 8
    - by New York Edison Co 12
    - by Palmenberg 11
    - to determine softening temperature .. 19
  - *See also* Ash cones, cones.
- Sewell bed, W. Va., coal from, ash analyses of. 28
  - low ash content of 14
- Selvig, W. A., acknowledgment to 118
- Shepard, E. S., cited 19
- Silica in ash of fuels tested 25
  - in coal, occurrence of 14
- Silica-tube furnace, figure showing 46, 47
  - *See also* Heraeus furnace.
- Silicates, ferrous and ferro-calcic, formation temperatures of 21
- Sillimanite in fuel-bed clinker 66
- Slags. *See* Clinker slags.
- "Softening interval," definition of 109, 113
- Softening point, definition of 50, 81
  - of ash cones, figure showing 34, 35
- Softening temperature, definition of 112
  - determination of 30
  - factors influencing 20
  - of ash cones, effect of position on 80
    - in reducing atmospheres 63
  - of ash containing Fe_2O_3 68
    - figure showing 70, 71
  - relation of, to temperature of complete melting 19
  - tests, standard, crucibles for, description of 116
- Softening temperatures in different furnaces, figure showing 38, 40, 41
  - in reducing atmospheres, figure showing. 57
  - of Seger cones in reducing atmospheres and air 72
    - in water vapor and hydrogen mixtures, figure showing 51, 52
- Sosman, R. B., on melting points of minerals. 15
- Spadra bed, Ark., coal from, ash analyses of. 28
- Stanton, S. M., cited 29
- Steffe, Herman., cited 21
- Sterling bed, Tenn., coal from, ash analyses of 26
- Swelling of ash cones, causes of 96

T.

- Taylor, G. G., acknowledgment to 118
- Temperature, maximum limit of, in commercial tests of ash 113
  - *See also* Ash cones, fusing temperature, softening temperature.
- Temperature measurements, methods of making 30, 49, 95
- Tennessee, coal from, ash analyses of 27, 28
- Thermocouple, for standard softening-temperature tests, description of 113
  - use of in softening-point tests 31, 96
- Texas, coal from, ash analyses of 28
- Thompson bed, Ala., coal from, ash analyses of 27

U.

- Upper and Lower Banner bed, Va., coal from, ash analyses of 28
- Utah, coal from, ash analyses of 28

V.

- Virginia, coal from, ash analyses of 27
- Viscosity of fused ash, effect of, on rate of deformation of cones 52

W.

- Wanner pyrometer, use of, in tests 49
  - *See also* Pyrometer.
- Washington, coal from, ash analyses of 28
- Water vapor, equilibrium of, with iron and iron oxides 44
- West Virginia, coal from, ash analyses of 27, 28
- Widow Kennedy bed, Va., coal from, ash analyses of 28
- Wright, F. E., cited 17
- Wyoming, coal from, ash analyses of 28

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Bulletin 130

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, Secretary
BUREAU OF MINES
VAN. H. MANNING, Director

BLAST-FURNACE BREAKOUTS, EXPLOSIONS, AND SLIPS, AND METHODS OF PREVENTION

BY

F. H. WILLCOX

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CONTENTS.

Page.

- Introduction 1
- Acknowledgments 1
- Part I. Blast-furnace breakouts 2
  - Purpose of investigation 2
  - Definition of the term "breakout" 2
  - Explosions from molten iron 2
  - Hazards from breakouts 2
    - Breakouts of iron 2
    - Breakouts of slag or cinder 3
    - Breakouts of the blast, gas, and coke 3
  - Bosh breakouts 3
    - Causes of bosh breakouts 3
    - Position of bosh in furnace 3
    - Development of present types of bosh 4
    - Banded boshes 6
      - Use of plate boxes 8
      - Brickwork construction between plates 9
      - Strength of bosh brickwork against displacement 9
      - Placing of plates 10
      - Cooling effect of plates 10
      - Cooling-water supply 11
      - Construction of bosh bands 12
    - Jacketed boshes 13
      - Durability of brick lining 13
      - Jackets compared with plate-and-band construction 13
      - Jacket construction 14
      - Factors tending to destroy jacket 15
        - Positive water cooling essential 15
        - Formation of skulls 18
        - Erosive action of charge 18
      - Use of auxiliary sprays 18
    - Other types of bosh construction 19
    - Causes of bosh breakouts 20
      - Examples of bosh failures 20
        - Bursting of bosh bands 20
        - Escape of iron into cooling-water ditch 21
        - Falling scaffold 22
        - Failure following hot spots on jacket 22
        - Failure from bursting of riveted seam 22
        - Failure from a heavy slip 22
      - Future hazard from bosh breakouts 23
  - Tuyère-breast breakouts 23
    - Construction of tuyère breast 23
    - Factors tending to destroy the brickwork 23
      - Softening of the brickwork 23
      - Erosion by the blast 24
      - Slagging of the brickwork 24
      - Pressure within tuyère breast 26

Part I. Blast-furnace breakouts—Continued. Page.

Tuyère-breast breakouts—Continued.

Use of cooling plates in tuyère breast 25
Protection of brickwork above the tuyères 26
Protection of brickwork below the tuyères 26
Omission of plates beneath the tuyère coolers 27
Use of plate boxes 28
Reinforcement of brickwork 28
Packing of tuyère coolers 30
Blowpipes and eyesights 31
Types of eyesights 32
Examples of tuyère-breast breakouts 33
Prevention of tuyère-breast or bosh breakouts 35
Importance of correct construction 35
Cooling 35
Regularity of furnace operation 35
Arrangement of cast house and signals 36
Other precautions in cast house 36
Hearth breakouts 36
Breakouts through the hearth jacket 37
Thinning of the hearth walls 37
Abrasion by the blast 37
Attack of slag 38
Cracking of the walls 39
Development of breakouts through the jacket 40
Thickness of the hearth walls 41
Use of packing between hearth wall and jacket 41
Types of jackets 42
The rolled steel-plate jacket 42
Jackets with cooling pipe cast in 43
Other types of jackets 43
Cooling the jacket 44
Breakouts below the tapping hole 45
Danger in using spray-cooled, steel-plate jackets 45
Formation of "salamanders" 45
Causes of destroying hearth bottom and walls 46
Slagging action of pig iron 46
Buoyant effect of superincumbent iron 47
Separation of graphite 47
Rate of lowering of hearth bottom 48
Example of a furnace "losing its iron" 49
Deep salamanders a source of danger 50
Typical examples of hearth breakouts 51
Example No. 1 51
Example No. 2 52
Example No. 3 52
Example No. 4 52
Example No. 5 53
Example No. 6 53
Example No. 7 54
Example No. 8 54
Example No. 9 55

Part I. Blast-furnace breakouts—Continued.

Page.

Hearth breakouts—Continued.

Typical examples of hearth breakouts—Continued.

Example No. 10 55
Example No. 11 56
Example No. 12 56
Prevention of breakouts through the hearth wall or jacket 57
Use of materials other than fire brick 57
Carbon brick 57
Silica, magnesite, and beauxite brick 58
Types of fire brick used in hearths 59
Use of concave bottoms 60
Use of keyed bottoms 61
Use of conical jacket 61
Usual construction of hearth bottom 61
Large shapes laid in close contact with jacket 61
Small shapes in hearth bottom 63
Depth of jacket below hearth bottom 64
Use of subjackets 65
Use of cooling pipes inside the hearth 66
Leaving in the salamander on relining 66
Protection of furnace columns 68
Cooling the wall about the tapping hole 68
Importance of furnace design and construction 69
Need of attention to cooling-water supply 69
Care of the hearth ditch 70
Precautions at blowing in 71
Regulation of cooling water 71
Types of slags at blowing in 71
Regularity in flushing and casting essential 72
Practice when casting is delayed 73
Methods of opening hard holes 73
Precautions to prevent the iron rising too high 74
Attention to signs of a breakout 75
Procedure when a furnace is "losing" or "holding" its iron 75
Procedure when steam issues from about the furnace base 76
Repairing the jacket and wall after a breakout 77
Tapping-hole breakouts 78
Factors tending to destroy the iron notch 78
Method of stopping the hole at blowing in 79
Care of tapping hole when furnace is in blast 79
Use of cooling plates over the tapping hole 80
Typical tapping-hole breakouts 80
Example No. 1 80
Example No. 2 81
Example No. 3 81
Example No. 4 81
Example No. 5 82
Example No. 6 82
Example No. 7 82
Example No. 8 82
Other tapping-hole accidents 82

Part I. Blast-furnace breakouts—Continued. Page.

Hearth breakouts—Continued.

- Prevention of tapping-hole breakouts 83
  - Precautions in preparing and loading the clay 83
  - Omission of cooling plates about the notch 84
  - A good practice in stopping the hole 84
  - Cleaning the hole 84
  - Use of a long plug or stopping 84
  - Blowing on the hole 85
  - Especial care necessary when casting is delayed 86
  - Stopping the hole with the blast on 86
  - Leaving the gun in the hole after casting 87
  - Use of splasher plate or shield 87
  - Manipulation of gun 87
  - Precautions with short holes 88
  - Precautions in opening the hole 88
  - Depth and angle of tapping hole 90
- Cinder-notch breakouts 91
  - Packing for cinder-notch coolers 91
  - Breakout from defective packing 92
  - Breakouts from cutting of the intermediate cooler 92
  - Breakouts from changing the monkey with the blast on 92
  - Packing the intermediate and large coolers to prevent cutting 93
  - Arrangement of cooling system 93

Part II. Gas explosions at blast furnaces 94
- Purpose of investigation 94
- Theory of gas explosions 95
  - Pressure produced by explosion 98
  - Explosive limits of blast-furnace gas 98
- Explosions at hot-blast stoves 100
- Explosions of gas at blast-furnace boilers 102
- Explosions at gas mains and in furnace tops 103
  - Gas-seal explosions 103
  - Top and downcomer explosions 105
    - Explosions during emergency stops 105
    - Explosions at shutdowns for repairs 108
  - Practice during stops for repairs to top or mains 108
    - Procedure at plant 1 108
    - Procedure at plant 2 109
    - Procedure at plant 3 109
    - Procedure at plant 4 109
    - Procedure at plant 5 110
    - Summary of methods at the five plants 110
  - Importance of design of gas offtake and bleeder 110
  - Explosions at shutdowns for changing tuyères or cooling plates 111
    - Different practices in shutting down to change bronze 112
      - First method 112
      - Second method 112
      - Third method 113
    - Use of steam in gas mains 114
    - Relieving gas pressure in the mains 115
  - Explosions during shutdowns at casting time to stop tapping hole 116

Part II. Gas explosions at blast furnaces—Continued.

Explosions at gas mains and in furnace tops—Continued.

Page.
- Accidental stoppage of gas 117
- Care and operation of rotary gas washers 117
  - Typical accidents 117
  - Essential precautions in handling gas 119
- Method of handling gas at a large plant 120
  - Gas consumption at furnace plant 120
  - Description of gas-cleaning plant 120
  - Safety of the plant 122
  - Furnace operation as regards gas supply 122
  - General safety precautions 123
  - Signals and gages for Theisen washer 125
- Method used at another plant 126
- Automatic appliances to regulate the flow of gas 127

Blowing in 128
- Causes of explosions at blowing in 128
  - Spontaneous explosion of gas mixtures 128
  - Bringing gas down with bells open 129
  - Bringing gas direct to stoves or boilers containing a fire 129
  - Building fires in gas mains 130
  - Shutting off the blast 131
- Methods of blowing in practiced at 18 plants 131
  - Plant 1 131
  - Plant 2 131
  - Plant 3 132
  - Plant 4 132
  - Plant 5 133
  - Plant 6 134
  - Plant 7 134
  - Plant 8 135
  - Plant 9 136
  - Plant 10 136
  - Plant 11 138
  - Plant 12 138
  - Plant 13 139
  - Plant 14 140
  - Plant 15 142
  - Plant 16 142
  - Plant 17 143
  - Plant 18 145
- Method in use at another plant 147
- Precautions in blowing in 147

Blowing out 151
- Causes of explosions in blowing out 151
  - Danger from accumulations of water 152
  - Accident from drafting gas from downcomer 153
- Methods of blowing out practiced at the 18 plants 153
  - Plant 1 153
  - Plant 2 154
  - Plant 3 154
  - Plant 4 155

Part II. Gas explosions at blast furnaces—Continued. Page.

Blowing out—Continued.

Methods of blowing out practiced at the 18 plants—Continued.

Plant 5 156
Plant 6 156
Plant 7 156
Plant 8 157
Plant 9 157
Plant 10 158
Plant 11 159
Plant 12 159
Plant 13 160
Plant 14 160
Plant 15 161
Plant 16 162
Plant 17 162
Plant 18 163
Other methods of blowing out 163
Precautions in blowing out 164
Explosions in cold-blast mains 167
Typical examples 167
Preventive measures 169
Check valves 169
Other automatic valves 170
Valves in tuyère stocks 171
Explosions of air compressors at blast-furnace plants 171
Explosions of molten iron 172
Tuyère explosions 174
Blasting out hearth bottoms at relining 175
Hazards from the use of explosives 175
Customary practice in blasting salamanders 176
Preparation of holes 176
Charging the holes 177
Firing the charges 177
Precautions to be taken in blasting salamanders 177
Preparing the holes 177
Quarter drilling 178
Distributing holes over the face 178
Holes should be thoroughly cooled 178
Firing the blast 179
Connecting shots in series or in multiple 179
Precautions in connecting shots 180
Connecting legs to leading wires 181
Splices should not be opposite—Wrapping splices 181
Connecting leading wires to firing machines 181
All connections to be made from bore hole to firing machine 181
Precautions in storing and handling explosives 181
Storage of explosives 181
Thawing explosives 182
Handling detonators and explosives 182
Using detonators and explosives 182

Page.
Part III. Blast-furnace slips ... 183
  Purpose of investigation ... 183
  Occurrence of slips ... 184
    Indications of hanging ... 184
      Appearance of the gas ... 184
      Rate of charging ... 185
      Temperature and pressure of the gas in the top ... 186
      Blast pressure ... 186
      Relation of variations in blast pressure and rate of charging ... 187
    Conditions causing transition from "hanging" to "slipping" ... 187
    Theories of the causes of blast-furnace slips ... 188
      Causes of bottom hanging and slipping ... 188
      Causes of top hanging ... 190
        Carbon deposition ... 190
        Investigation by Metz ... 193
        Ejection of carbon dust ... 193
        Amount of carbon deposited ... 194
        Other theories of the cause of top hanging ... 194
      Probable cause of top hanging ... 196
      Cold working ... 196
    Hanging near the central part of the stack ... 197
    Character of blast-furnace slips ... 197
      Mechanism of top slips ... 197
      Mechanism of bottom slips ... 198
    Pressure generated by slips ... 199
      Methods of calculating pressure ... 199
    Furnace design for relieving pressure from slips ... 202
    Theories of the explosive character of slips ... 203
      Combustion of finely divided coke and ore ... 203
      Combustion of deposited carbon by carbon dioxide ... 204
      Admission of air to spaces above the hanging stock ... 205
      Admission of air to spaces beneath arched stock at bottom of furnace ... 206
      Skappel's theory ... 207
      Conclusions as to causes of slips ... 207
  Prevention of hanging and slipping ... 208
    Design of bell and relation to stock line ... 209
    Distribution of burden ... 211
      Revolving distributors ... 212
      Design and position of bell ... 212
    Lines of furnace lining ... 213
      Contour of upper part of furnace ... 213
      Contour of lining at the bosh ... 214
    Preservation of lining ... 215
      Preservation of the lining at the stock line ... 216
      Preservation of the lining above the bosh ... 217
      Thickness of lining, and water cooling ... 218
        Use of thick linings ... 218
        Use of thin linings ... 219
        Use of linings of medium thickness ... 219
        Placing of cooling plates ... 220

Part III. Blast-furnace slips—Continued. Page.
Prevention of hanging and slipping—Continued.
Inferior brick or bricklaying as a cause of failure 220
Gas offtakes and distribution of blast 221
Materials used for the burden 222
Flux 222
Coke 223
Ores 225
Effect of charging coarse ore 225
Effect of charging fine ore 225
Sintering and briquetting ores and flue dust 227
Furnace operation 227
Volume and pressure of the blast 227
Insufficient volume of blast 227
Excessive volume of blast 228
Irregular volume and pressure 228
Distribution of blast 230
Channeling from the blast 230
Blast pressure 231
Selection of tuyères 232
Changing of tuyères 232
Prevention of leaks in the water-cooling system 233
Effect of working at high temperatures 234
Composition of slag 235
Stocking and charging 236
Weighing and loading 236
Charging coke 237
Size of coke unit charged 238
Sequence of charging coke, ore, and flux 238
Inspection of the bells 241
Height of stock line 242
Use of stock-line indicators 243
Speed of opening bells 243
Importance of proper distribution 244
Use of coke blanks and "doses" 244
Methods of overcoming slipping and hanging 248
"Blowing down" 248
Use of auxiliary tuyères 248
Use of explosives to dislodge scaffolds 249
Other means of preventing slips 250
Manipulation of blast pressure and temperature 251
Procedure when cinder is at the tuyères 253
Varying the burden 253
Use of steam in blast 253
Examples of accidents caused by slips 254
Character of injuries 256
Severity of slips in present types of furnaces 256
Examples of expedients used to check slipping 257
Precautions and safeguards 259
The use of tight tops 259
Reinforcing the bosh and strengthening the top 260
Explosion doors 260

Part III. Blast-furnace slips—Continued. Page.
Precautions and safeguards—Continued.
Bleeder valves 260
Necessity of strong construction 261
Signals and warnings 261
Signals between cast house, blowing room, and skip operator's station 261
General methods of warning workmen 262
Protection from falling and flying material 263
Precautions to prevent accidents from slips 263
Precautions in going on furnace top 264
Precautions in working about the cast house 264
Precautions in working about dust catchers and stoves 264
Precautions in the boiler room 264
General precautions 265
Conclusion 266
Need of further investigations 266
Glossary 268
Publications on metallurgy 271
Index 273

ILLUSTRATIONS.

Plate I. Jacketed furnace with bottom of bosh resting on brickwork 14
II. Details of tuyère stock and blowpipe equipment 30
Figure 1. Section of hearth bosh, and foundation of typical modern blast furnace 4
2. Section of lower part of furnace 5
3. Section of hearth and bosh 6
4. Section of bosh and foundation 7
5. Erosion of bosh walls 8
6. Cast-iron box for cooling plate 9
7. Brickwork between plates 10
8. Diagrams showing construction of arch 11
9. Two methods of holding bosh bands in place 12
10. Jacketed bosh 15
11. Jacketed furnace with bosh jacket riveted to tuyère-breast jacket 16
12. Jacketed furnace in which expansion of bosh is provided for 17
13. Jacketed bosh built with cast-iron segments and circulating water pipe cast in 19
14. Jacketed bosh built with riveted steel plate lined with cast-iron staves, water pipe cast in 20
15. Bosh inclosed in steel jacket with holes for cooling plates 21
16. Sketch showing construction of tuyère-breast wall with cooling plates between cooler arched 26
17. Sectional view of tuyère-breast cooling plates 27

Page.
FIG. 18. Tuyère stock, showing lug for auxiliary bridle rod — 30
19. Eyesight with ½-inch aperture — 33
20. Eyesight with glass set back from tuyère cap — 34
21. Eyesight, keyed type — 35
22. Eyesight, stop-cock type — 36
23. Eyesight with counterweight — 37
24. Diagram showing how hearth walls are attacked — 38
25. Concave hearth bottom — 60
26. Method of laying blocks in hearth bottoms — 62
27. Another method of laying hearth bottoms — 63
28. A third method of laying hearth bottoms — 64
29. Hearth bottom having cast-iron jacket with cooling pipes cast in — 66
30. Section showing method of repairing hearth bottom — 68
31. Diagram showing hearth conditions of fast-driving furnace — 73
32. Method of closing tapping hole when furnace is blown in — 81
33. Cross bars for holding the tapping bar — 88
34. Sections showing method of using shield — 89
35. Section of tapping hole, showing plate over skimmer trough — 90
36. Diagrams showing relation between depth of hearth jacket and slope of tapping hole as regards safety — 91
37. Clamps for cinder cooler, intermediate, and monkey discharge pipes — 93

BLAST-FURNACE BREAKOUTS, EXPLOSIONS, AND SLIPS, AND METHODS OF PREVENTION.

By F. H. Willcox.

INTRODUCTION.

This publication is the third of a series of reports on hazards and the prevention of accidents at blast-furnace plants that is being published by the Bureau of Mines, Technical Paper 106[a] being the first and Technical Paper 136[b] the second of the series.

This bulletin, which is arranged in three parts, treats of the causes and prevention of blast-furnace breakouts, explosions, and slips. The first part discusses blast-furnace breakouts, the hazards therefrom, causes, and features of furnace design and methods of operation that tend to prevent breakouts or lessen their number, and points out safety measures to be adopted. The second part takes up the theory and causes of gas explosions at blast-furnace plants and reviews the better methods in use for handling gas. The third part analyzes the causes of hanging and scaffolding in blast furnaces, with the resulting slips that frequently cause injury to the plant and danger to the workmen, describes features of construction and furnace design and methods of operation employed to prevent hanging and slipping, and suggests precautions and safeguards.

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Thanks are due Mr. Frederick Crabtree, professor of metallurgy at Carnegie Technical Schools, Pittsburgh, Pa., for amendments and suggestions.

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[a] Willcox, F. H., Asphyxiation from blast-furnace gas: Tech. Paper 106, Bureau of Mines, 1915, 69 pp.

[b] Willcox, F. H., Safe practice at blast furnaces, a manual for foremen and workmen: Tech. Paper 136, Bureau of Mines, 1916, 73 pp.

PART I.—BLAST-FURNACE BREAKOUTS.

PURPOSE OF INVESTIGATION.

Breakouts are an infrequent and insidious hazard of furnace work. A study of their dangers, the technical aspect of their occurrence, and the methods in use for preventing them has been made by the writer. The results of this study are presented in the following pages.

DEFINITION OF THE TERM "BREAKOUT."

A "breakout" is the term employed to denote the conditions and results of the escape of gas and coke, or slag, or iron, from the bosh, tuyère breast, or hearth of a blast furnace. Breakouts may occur at any point below the fusion zone in the furnace, but the most frequent breakouts are of molten iron at a level below the surface of iron lying in the hearth, and are either through the hearth walls and jacket or into the hearth bottom and out under the hearth jacket.

EXPLOSIONS FROM MOLTEN IRON.

If this runaway fluid iron comes in contact with water or wet material an explosion occurs which may wreck the furnace or cast-house equipment. An explosion in one plant caused damage amounting to $80,000. Several furnaces have been put out of blast by such accidents, and in numerous instances men have been injured or killed by flames, flying metal, brickbats, and castings when these explosions take place.

The term "explosion," as applied in this sense, should not be confused with explosions of gas, which are discussed in Part II of this report.

HAZARDS FROM BREAKOUTS.

BREAKOUTS OF IRON.

At most plants in this country the danger of hearth breakouts has been reduced remarkably during the past 10 years, and at these plants, so the writer believes, is becoming almost negligible as a factor in the accident risk. The hearth has received much attention, and heavier, stronger, and more expensive construction has been developed until the two types of hearth now standard rarely permit explosive breakouts.

BREAKOUTS OF SLAG OR CINDER.

Slag breakouts occur from the top of the hearth jacket and about the cinder notch up to the level of the tuyéres. They are seldom dangerous to life or limb but may cause some damage to the brick lining and are a considerable nuisance and annoyance, because of the resulting delay for repairs and the time necessary to clean up the "mess" they cause.

BREAKOUTS OF THE BLAST, GAS, AND COKE.

Blast, gas, and coke breakouts, usually known as bosh breakouts, are almost a thing of the past; the last one in this country was in the spring of 1914. Their elimination may be attributed quite as much to improvements in practice, smoother work on increasingly inferior ores, and less violent "slips," as to strengthening of boshes. Although improvement in boshes is generally keeping pace with other furnace construction in augmenting security, nevertheless at many plants the bosh is not as strong, proportionately, as other parts of the furnace. With the present control over furnace operation, the bosh does not fail except at infrequent intervals. However, if plants with somewhat weak boshes should be subjected to periods of heavy driving for large production, with consequent severe slips, such as have happened in the past, it may be that some boshes although adequate for normal pressures and slips would give way. One such accident has happened.

BOSH BREAKOUTS.

CAUSES OF BOSH BREAKOUTS.

Bosh breakouts may be caused as follows: (1) By conditions inside the furnace, such as high pressure of blast, very heavy slips, or severe working on the hearth walls, all of which may lead to (2) breaking of the hearth bands, ejection of cooling plates, or parts of the brickwork between the bands and the plates, or (3) cracking and opening of bosh jackets. The second and third items may usually be traced back to deficient construction or questionable design.

POSITION OF BOSH IN FURNACE.

The bosh of the blast furnace shown in figure 1 is an inverted truncated cone, so designed as to bring its top just below the zone of fusion, the limits of which depend on the grade of ore used and the temperature and volume of the blast; the bottom is arbitrarily fixed at 12 to 24 inches above the center line of the tuyères.

Inside of this cone, which is usually about 22 feet in diameter at the top, 17 feet in diameter at the bottom, and 12 feet high, there is burned every minute about 750 pounds of carbon, which liberates approximately 3,000,000 heat units at a temperature of 2,750° to 3,200° F. and produces 62,000 cubic feet of gas. This gas, which is under a temperature of 3,000° F. and pressure of 15 pounds, is expanded to approximately 210,000 cubic feet, and rushes upward with a velocity of probably not less than 3,000 feet per minute. At the same time the coke in the bosh is sliding down and grinding the walls, which are incandescent but firm and solid, and down which globules of corrosive slag and iron continually run and drip. Under these conditions of high heat, strong pressure, and the scouring action of descending raw slag and iron, and ascending gas at high velocity, it is evident that the construction and maintenance of the bosh present difficult problems.

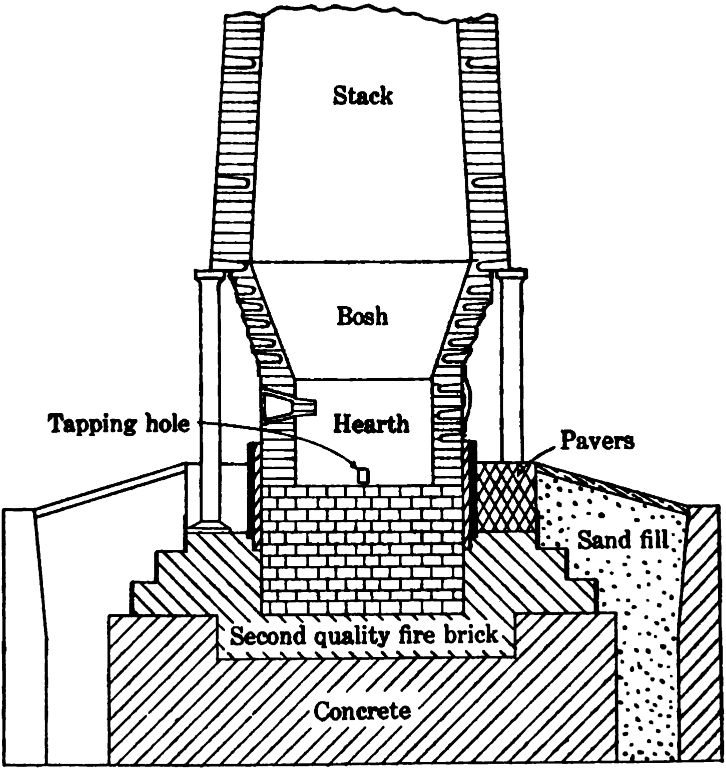


FIGURE 1.—Section of hearth bosh, and foundation of typical modern blast furnace.

DEVELOPMENT OF PRESENT TYPES OF BOSH.

The type of bosh now being built is a development of about 50 years' experience, its growth being markedly accelerated during the decade of 1900 to 1910. During this decade the increasing use of Mesabi iron ores resulted in heavy scaffolding and bridging in the furnace and in severe slips, which added to the normally heavy duty of the bosh the necessity of withstanding the impact of columns of stock, weighing hundreds of tons, which fell from a height varying from 10 feet to an estimated height of 30 feet. Several serious and fatal accidents were caused from the boshes bursting under the blow of a falling stock column and the compression of the gas beneath the stock. This subject is discussed at greater length under slips.

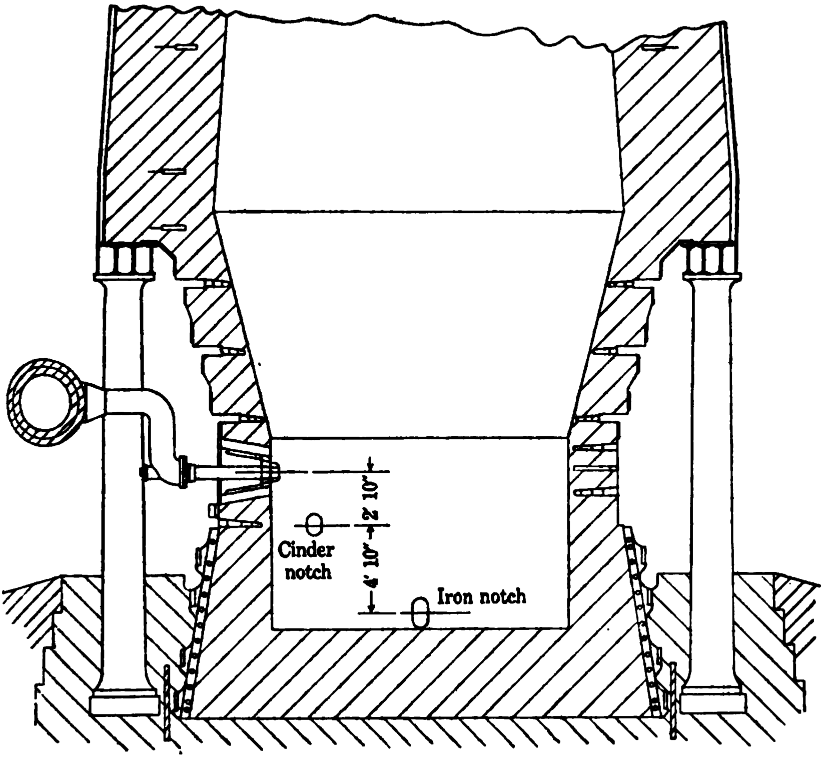


FIGURE 2.—Section of lower part of furnace.

The early blast furnaces were simply pyramids of stone with fire-brick linings. They had boshes, but these did not show on the outside, being surrounded by means of masonry. An opening was left on one side for the tapping hole. The use of tuyères in larger number introduced more openings or arches on the sides, until the stack was partly supported on brick or masonry pillars and the bosh assumed external form. About 1860 the stone or brick pillars were replaced by iron columns, but the fire-brick lining was still held in place by a massive ring of ordinary red brick, not water-cooled. This hearth fire brick, of course, was badly eroded and the bosh construction was made thinner, was banded with flat iron straps, and iron cooling plates were inserted. To describe in detail the development to the present form would require too much space. It was, in brief, a progressive evolution; changes in fuel and ore required more limy slag, a more limy slag required hotter combustion, and this in turn required a stronger and hotter blast, which resulted in larger production, and larger production required better construction. Weaknesses developed, accidents happened, deficiencies were corrected, and one improvement led to another until there are now two distinct types of bosh construction which, so far as may be conceived, are the last word in construction, design, strength, and security.

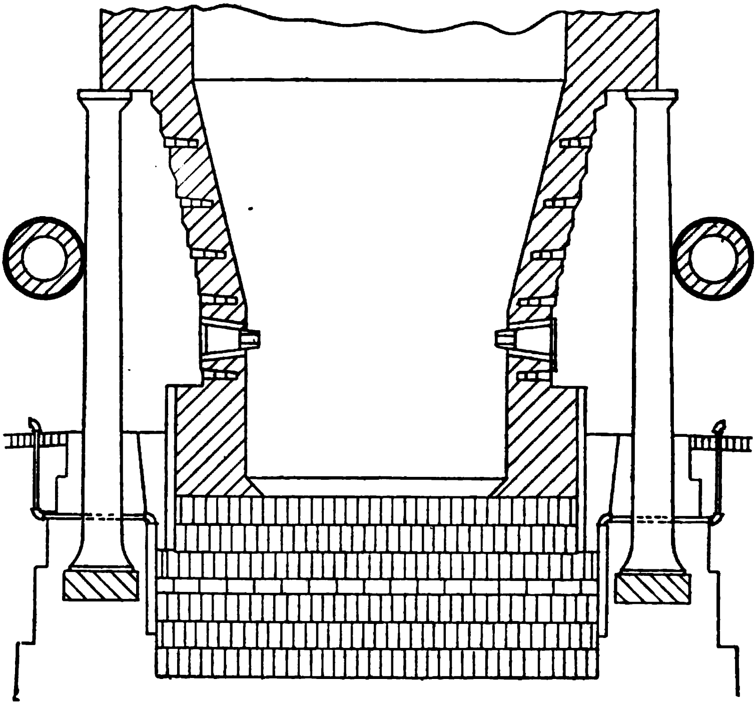


FIGURE 3.—Section of hearth and bosh.

BANDED BOSHES.

A type of construction largely used in plants dependent on Lake ores is illustrated in figures 2, 3, and 4. It consists of brickwork, in which is inserted five to eight horizontal superimposed rows of bronze cooling plates. These plates are, as a typical example, 4½ inches thick at the back by 3⅛ inches thick at the nose, which is rounded; are 2 feet 3 inches wide at the back and taper to 1 foot 11½ inches at the nose. The depths of the plates differ according to the manner in which they are placed. In most furnaces a flat brick arch is turned in the brickwork above the plate and the plate extends from the outside to the inside face of the lining or 1 inch back from the inside face. When these plates must be replaced they are likely to be exceedingly difficult to withdraw. They should be easily removable, for, in spite of the best water-cooling, plates will crack or burn. Also the new plate should fit tightly enough in the opening left from the old one that the blast or gas can not blow out, erode the brick seat, and let cinder run through, but the opening should be large enough to permit the new plate to extend inward the same depth as the old one withdrawn to prevent the wall from becoming dangerously thin. Putting in a new plate may be more difficult with a brick arch than with a plate box, because the brickwork may have wedged down and the old plate may be sprung.

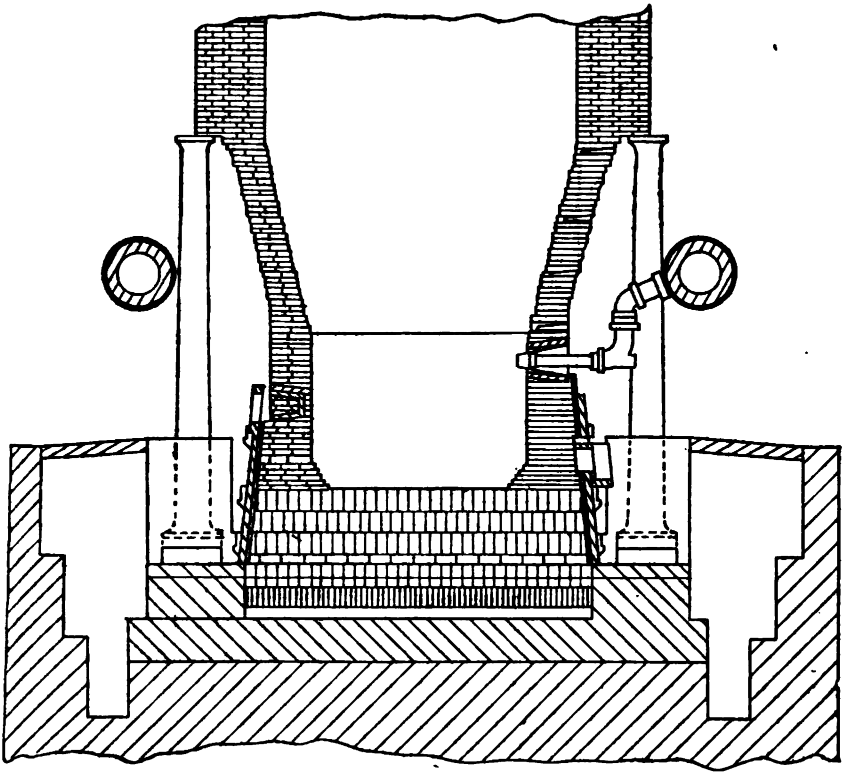


FIGURE 4.—Section of bosh and foundation.

Typical examples of erosion of bosh walls between the plates are shown in figure 5. Instances are known where the brickwork has disappeared almost to the band, being replaced with a mixture of dust, slag, carbon, and partly reduced ore. When the scouring action has thinned the walls too much these plates, being laid in the brickwork, are not held firmly, and in some instances have been tilted down by the weight and motion of the descending stock, breaking the water connections and weakening the adjoining construction.

USE OF PLATE BOXES.

To simplify the changing of plates, the use of cast-iron boxes is much in favor. These boxes are built in the brickwork one-half or one-third the thickness of the bosh wall, and the cooling plate inserted through the box, as illustrated in figure 6. Such boxes are recommended, by those who use them, as being free from any tilting of plates, admirably adapted to being tied in with the overlying and lower bosh bands to make a strong reinforcement immune to blowouts, and as reducing the weight of metal in the copper plate by one-fourth, and thus lessening the cost. The plate may be blocked in position in the box by a short lug, as has been done at some furnaces.

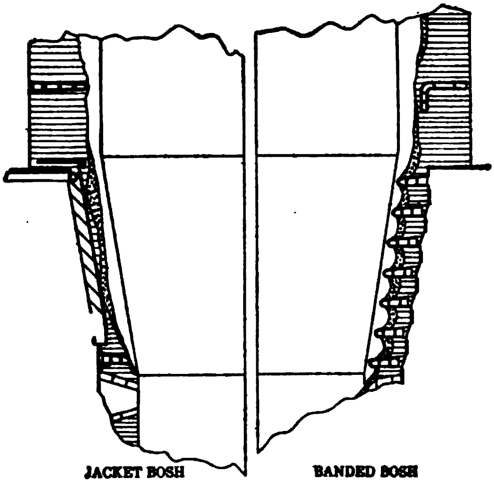


FIGURE 5.—Erosion of bosh walls.

There are 18 to 26 plates in each row, and they are usually separated by brickwork laid between the boxes, which are 13½ to 14½ inches apart, horizontally. In some plants the boxes are laid continuously about the entire row. The vertical distance between centers is 20½ to 31½ inches, of which 12 inches is usually covered with a steel band. This leaves a segment of brickwork, inclosed at the sides by the plates and at the top and the bottom by bands, ranging from 4½ to 13½ inches by 8½ to 19½ inches in size. This brickwork between the plates is usually not reinforced or backed on the outside with steel or iron plates. However, the practice at a few plants and the tendency at many others is to reinforce this small segment of brickwork with a short vertical buckstay extending from the lower to the upper band. A furnace thus equipped has no brickwork exposed on the outside, except the small areas between buckstays, plates, and bands. As the bosh brickwork is all held together by wide bands, with plate boxes held in by buckstays backed against the band, it is difficult to conceive that such boshes can blow or break out, and to date they have not.

BRICKWORK CONSTRUCTION BETWEEN PLATES.

The brickwork of the bosh averages, on modern furnaces, 31½ to 40½ inches thick, and is built in various shapes, as the construction is made complicated by the insertion of plate openings. Figure 7 shows a typical example of brickwork construction between plates.

The bosh bricks are almost exclusively 2½ inches thick and are laid flat; only a few plants have used 3-inch bricks on the bosh, although it is customary to use 3-inch in the inwall above the bosh. The bricks are laid in concentric circles with the joints broken or staggered vertically and radially; their dimensions closely conform to the circles with no cutting. The only unbroken joint is along the horizontal plane between each course. This construction is interrupted at each plate by an arch, as shown in figure 8.

In this figure the diagram at the left shows an end view of the arch. The view at the right shows a section on the vertical plane through the center of the arch.

The insertion of plates that vary in dimensions from the standard dimensions of the brick requires either considerable cutting of bricks or the manufacture of special shapes of brick, the dimensions of which are calculated from preliminary drawings showing the plate location.

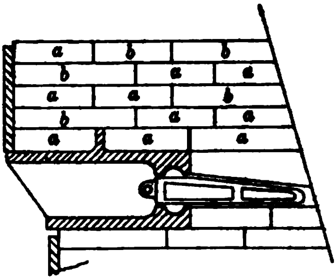


FIGURE 6.—Cast-iron box for cooling plate.

STRENGTH OF BOSH BRICKWORK AGAINST DISPLACEMENT.

Such a construction is very safe from blowouts if it holds together. The brickwork itself possesses no resistance to being pushed out other than the resistance due to its inertia, and to the compression of the superincumbent brickwork of the stack, yet it must withstand the pressure of the blast, stock, and slag, and the strain due to expanding of the bosh from heating. Under these conditions it is easily conceivable that sections of brickwork may be ejected from the bosh, and it is not surprising that such failure does occur, more especially as the thickness of the lining may be reduced from 36 inches to 18, and even less. This possibility is generally recognized, and the bosh brickwork, as has been previously stated, is usually reinforced between the plate rows with a metal band or, where plate boxes are not used, the brickwork between each two rows of plates is reinforced by two bands which, although they may not cover the brickwork because of the necessity of "stepping" the exterior face, should not allow more than one course of brick to be unbound. (See fig. 8.)

PLACING OF PLATES.

As a rule, in most furnaces the same shape and size of plates is used in each row, the number of plates being increased as the circumference of the rows up the bosh increases. There may be 20 plates in the bottom row and 26 in the top row, the horizontal distance between centers of adjacent plates being kept as nearly equal as possible. The plates in the different rows are not vertically above each other, but are alternated so that the brick between each pair of plates is covered and shielded to some extent by the plate above. This alternation is more exactly accomplished by using different widths of plates, and a few plants have as many different plate patterns as there are rows of cooling plates in the furnace. This arrangement can not be said to give more efficient protection than the use of plates of one pattern which do not permit such exact alternative spacing.

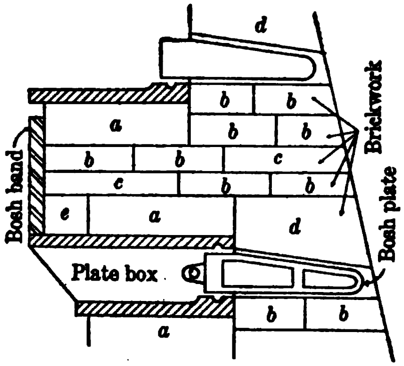


FIGURE 7.—Brickwork between plates; *a* to *e*, various shapes of brick.

COOLING EFFECT OF PLATES.

The cooling water is always fed in at the bottom of the plate, so as to displace mud deposited or carried in the water, and discharged from the top to prevent air bubbles accumulating near the roof of the plate, which would allow the plate to crack or burn or, possibly, explode. It is solely by the cooling effect of these plates that the bosh is preserved. A bosh, although built strong enough to withstand mechanical pressure and abrasion at low temperatures, might not last an hour if it was not cooled with water. This protection is essentially effected by keeping the temperature of the brickwork about the plates below that at which slag can combine with the brickwork. In fact, the temperature is usually kept lower than is necessary, and at times cooling is carried to excess, so that instead of an equilibrium between cooling effect and slagging action being obtained, scaffolding takes place—that is, slag, or pasty, partly reduced ore, or carbon dust freezes on the side of the bosh and impedes the descent of the stock from above. Overcooling of the bosh is serious and dangerous because it may lead to heavy slips; also the furnace may be put out of commission for days when the scaffold is dislodged.

COOLING-WATER SUPPLY.

The number of plates that can be used is not arbitrary, or even standardized. For instance, three furnaces in the same district, with closely identical production and dimensions, are equipped as follows: Furnace A, 5 rows, 91 plates; furnace B, 8 rows, 178 plates; furnace C, 6 rows, 132 plates. Furnace C works smoothly and regularly; furnace B is characterized by frequent and continual slips; furnace A works much more smoothly than B, but loses more plates.

The amount of water used is about the same, regardless of the number of plates in the bosh. When 90 plates are used, the bottom two rows of plates have an individual feed and discharge for each plate, whereas with 180 plates, the bottom two rows have two plates connected in series. The plates in the higher rows are always in horizontal series of two, three, and four each; the top rows, being under less severe temperature conditions, have the larger number of plates in series.

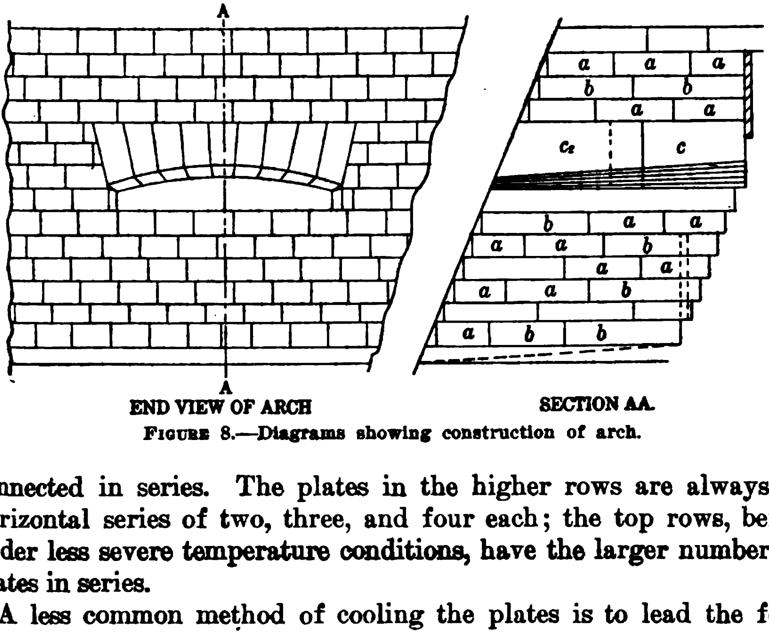


FIGURE 8.—Diagrams showing construction of arch.

A less common method of cooling the plates is to lead the feed into a plate on the bottom row; the discharge forms the feed for a plate immediately above in the second row, and so on up to the top row, five to eight plates being thus connected in series. The plate on the bottom row, at the hottest part of the combustion zone, is supplied with the coldest water; whereas the top-row plate, where overcooling is most to be feared, is supplied with water at a comparatively high temperature, but rarely more than 110° F. However, this arrangement uses less water, the piping is less in the way, and it appears to have been as efficient in cooling and preserving the bosh walls as the other method.

The temperature rise in the cooling water is very inconsistent; it varies, even with very cold water, from 4° F. to about 50° F. Furnace men from abroad have invariably commented on the large volume of cooling water used at American plants, it apparently being much in excess of that used in foreign practice. It goes without saying, however, that American managers err, if anything, on the side of overcooling, which, if not markedly excessive, is toward the safety of the men and plant, although resulting in some theoretical or actual increase in coke consumption. That it is somewhat hazardous to experiment with diminution of cooling effect is indicated by an experiment in which the water cooling in two alternate top rows was eliminated. This led to failure and breakout of the bosh, but, fortunately, no lives were lost.

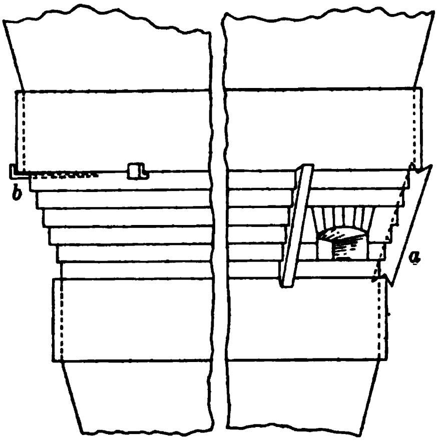


FIGURE 9.—Two methods of holding bosh bands in place. *a*, Iron strap laid in brickwork; *b*, lug laid in brickwork. The best method is to rest the bands on the plate boxes and not use straps or lugs.

CONSTRUCTION OF BOSH BANDS.

On furnaces having plate cooling there are usually between each row two bands about 5 inches by 1 inch each, or one band 12 or 14 inches by three-fourths inch up to 1⅛ inches, to give circumferential strength. With six rows of plates there are, then, seven circles of single bands or ten double and two single bands about the bosh. The most recent bands are 1½ inches thick and a few are 1¾ inches thick; they are of sufficient width to cover all the brickwork between the plate boxes, except for one-half inch vertically to provide for any slight expansion. These bands are strong enough, as a rule, to resist a bursting pressure of 160 to 350 pounds per square inch of bosh area. They are butt-strap riveted in two to four segments, thirty ¾-inch to 1⅛-inch rivets generally being used. The bands are usually so arranged that no two riveted parts are vertically in line, so that if one segment of the bosh is under heavy stress the load is distributed equally over the butt riveted and the other sections. The brickwork is always laid tightly against the bands, with no packing space, as is sometimes permissible in the stack and hearth, because lateral displacement of brick in the bosh can not be permitted.

Some means of holding the bands in position is necessary. In one accident the bosh bands slipped down after a hearth breakout, the bosh walls were immediately pushed out, and three men were overwhelmed in an outburst of flame and incandescent coke. The bands are best held by resting them on top of the plate boxes. This arrangement, which is shown in figure 4, is absolutely secure. Another method is the use of a distance piece between the plates, extending from the lower to the upper band, as shown at *a*, figure 9. A third means is to lay an iron strap in the brickwork so that the strap projects and affords a base for the band, as shown at *b*, figure 9. This construction is the least satisfactory of all, though perhaps most widely used, as the strap may become loose.

JACKETED BOSHES.

There are two stock objections to the plate-and-band construction: (1) The cost is excessive; (2) when the brickwork between the cooling plates has eroded back 12 or 18 inches, as inevitably happens, it forms a series of ledges on which slag and other material can lodge. When the furnace is working irregularly under conditions of heavy driving, and a large scaffold forms and then the furnace slips, this mass of relatively cold material slides down in front of the tuyères and closes them up, chills the furnace, and perhaps freezes the tapping hole or cinder notch. This has led in some districts to the use of the jacketed bosh.

In this construction the bosh is entirely inclosed by a riveted steel-plate jacket. The lining is usually made very thin, 9 to 13 inches, although a lining 24 inches thick is occasionally put in, and the exterior surface is cooled with a film of water, sprayed at the top, which in descending covers the plate work of the bosh and carries the heat away rapidly enough to cool the brick lining of the hearth below the slag-forming point. In this way the cutting action of the slag is prevented.

DURABILITY OF BRICK LINING.

It is invariably found that in the course of a few days or weeks the original 18, 13, or 9 inches of brickwork is reduced to 2 to 6 inches thick, and it may completely disappear. However, owing to the sloping surface of the bosh and the chilling effect of the cooling water, any slag coming in contact with the thinly protected or naked plate work cements itself together with other material, largely carbon, on the exposed surface and builds a skin of insulating refractory material which, as long as it remains in position, is just as effective a protection as the original brick lining.

JACKETS COMPARED WITH PLATE-AND-BAND CONSTRUCTION.

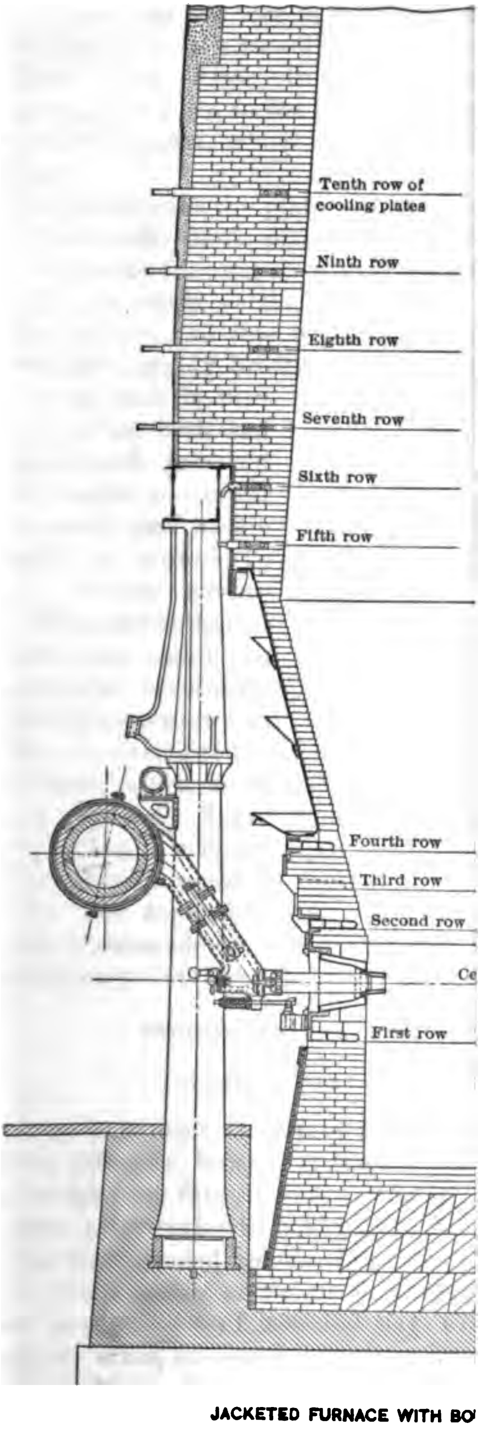
Theoretically, such a jacket has much to recommend it. It can be made as strong as desired, cooling should be effective over the entire surface, and the interior surface should be smooth instead of corrugated. Yet, so far as known, not a single furnace in the Pittsburgh, Youngstown, Buffalo, Cleveland, or Chicago districts, nor in contiguous territory, is jacketed. In northern New York, central and eastern Pennsylvania, southern Ohio, and the South jacketed furnaces are used in considerable numbers, but even in these districts the plate-and-band construction seems to be largely replacing the jacket. This tendency is not, however, based on considerations of safety so much as on advantages in operation, a discussion of which is not germane to this paper. Indeed, a superficial review of the history of the jacketed bosh would indicate that with regard to safety it is superior to the banded construction. The hearth and stack of the blast furnace were formerly constructed of fire brick and banded with steel strips, and historically the banded bosh might be considered a remnant of early building, which at other parts of the furnace less subject to heat, pressure, and erosion has been replaced with jacket construction, dictated by requirements of strength and resistance to bursting and outbreaks of blast and stock.

However, in spite of the theoretical advantages of smooth slope, strength, cooling, and tightness, furnace men going from plants smelting Lake ores to those using magnetites, hard ores, cinders, and local ores usually construct the bosh, at the first relining, according to their former experience—that is, with plate-and-band construction—while those who have tried the jacketed bosh on Lake ores express themselves as perfectly satisfied with the plate-and-band bosh and have no desire to try the jacketed bosh again.

JACKET CONSTRUCTION.

Bosh jackets are constructed of half-inch to 1¼-inch steel plate, riveted vertically on butt straps. Riveting and calking the seams is not unusually difficult or different from other riveting work, except that it is necessary to have all seams absolutely tight, the riveting first class, and the rivets fill the holes completely so that no opportunity may exist for gas to blow out. The interior and exterior conditions of service when the furnace is in blast are too rigorous for any possibility of outbreaks to be permitted, which may be obviated by special care in construction.

The bosh jacket is usually provided with riveted flanges at the top or is flanged to provide means of making connection at the mantle plate. Sometimes this flanged top is not riveted to the mantle, but is anchored or imbedded in the brickwork above the mantle, or it is riveted to a segment of the stack jacket. The stack jacket is, however, riveted or otherwise attached to the mantle plate and extends vertically downward to the junction point of jacket and bosh. (See fig. 10 and Pl. I.)



JACKETED FURNACE WITH BO

In the same way the bottom may be flanged or angles riveted on for attaching a trough to collect the cooling water running down the surface of the jacket. In some cases the bosh jacket is simply extended farther down to the hearth jacket, the lower part forming a tuyère-breast jacket, which is cooled by the water running off the bosh jacket.

As contrasted with the plate-and-band bosh, in which the weight of the whole bosh—brick, plates, bands, and boxes—with part of the weight of the stack lining rests on the hearth walls, in jacket boshes practically the whole weight of the jacket and brick lining is hung from the mantle, and this is one source of weakness. Contrasted with the thick, sturdy brickwork of the banded bosh, the lining in the jacketed bosh is very thin, and as the bosh and stack brickwork gets hot the compression may crush the brickwork and buckle the jacket, a possible though remote contingency. When the bottom of the jacket rests solidly on a structure of brickwork, as in the furnace shown in Plate I, or is riveted to the tuyère-breast jacket, as in the type shown in figure 11, such accidents may occur. Usually allowance is made for this expansion, as in the furnace shown in figure 12, in which the bosh jacket in expanding may slide over the water-cooled tuyère-breast jacket.

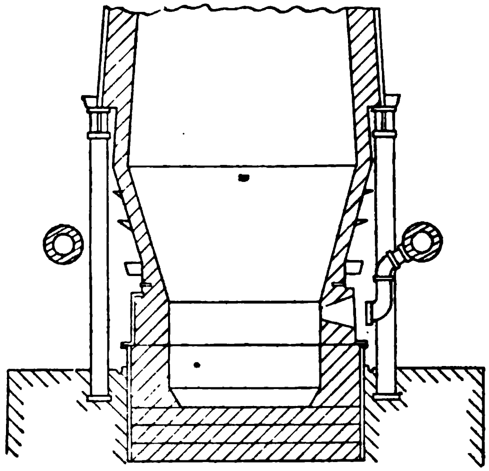


FIGURE 10.—Jacketed bosh.

FACTORS TENDING TO DESTROY JACKET.

POSITIVE WATER COOLING ESSENTIAL.

Buckling is evident at certain furnaces, and introduces the main operating difficulty, because if the surface of the jacket is not perfectly smooth, the film of water essential for cooling and preservation of the brickwork will not cling to it. In a short time the small space not water-cooled becomes hot and then white hot as the brickwork or conglomerate on the inside melts off. The jacket is quickly exposed to the attack of iron and slag, which may melt an opening through it. This, of course, lets out a jet of incandescent coke, gas, and slag and iron globules, which rapidly enlarges the opening and may result in a serious breakout. The formation of surface in-

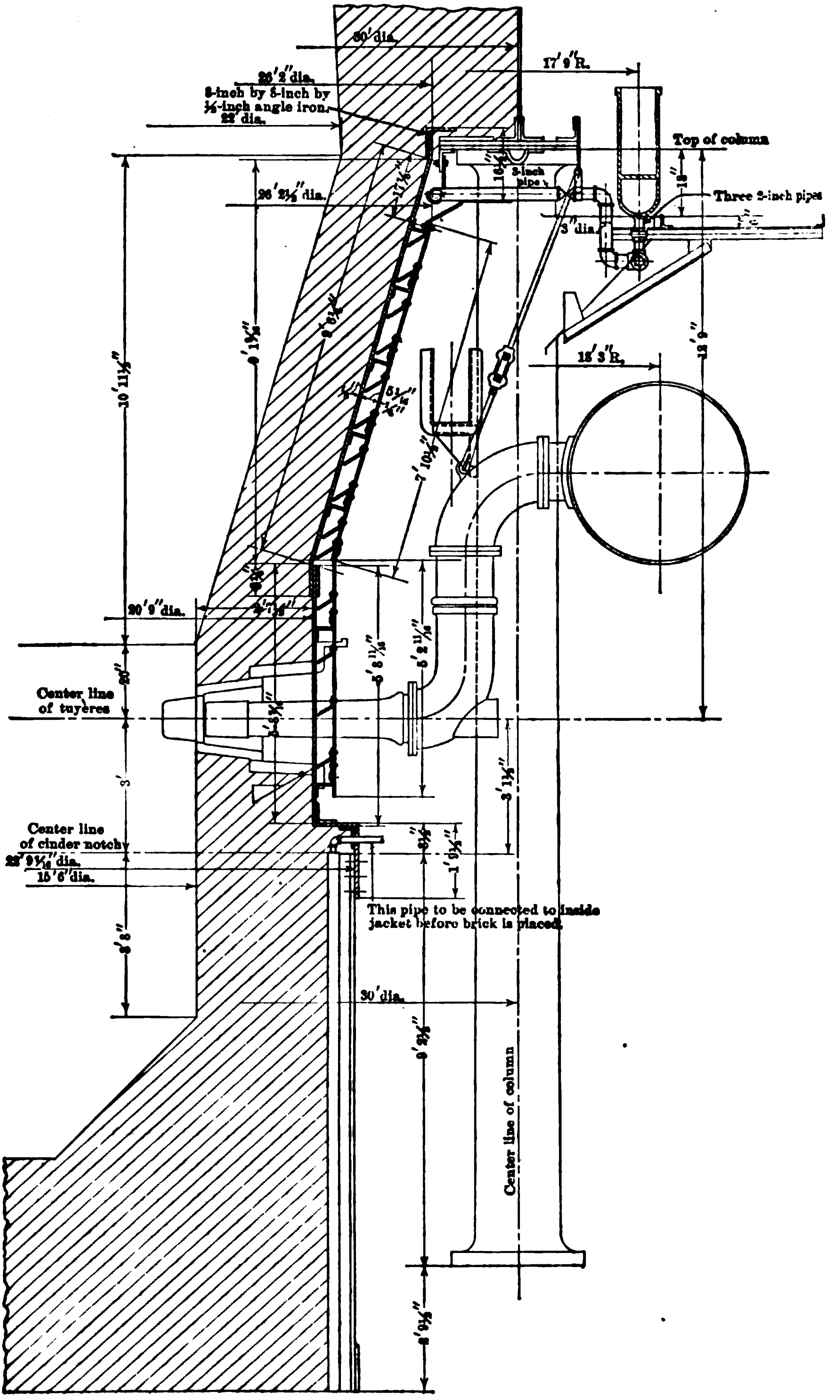


FIGURE 11.—Jacketed furnace with bosh jacket riveted to tuyère-breast jacket.

equality during operation is discouraging, for it makes useless the care and money spent during construction to insure a perfectly smooth surface.

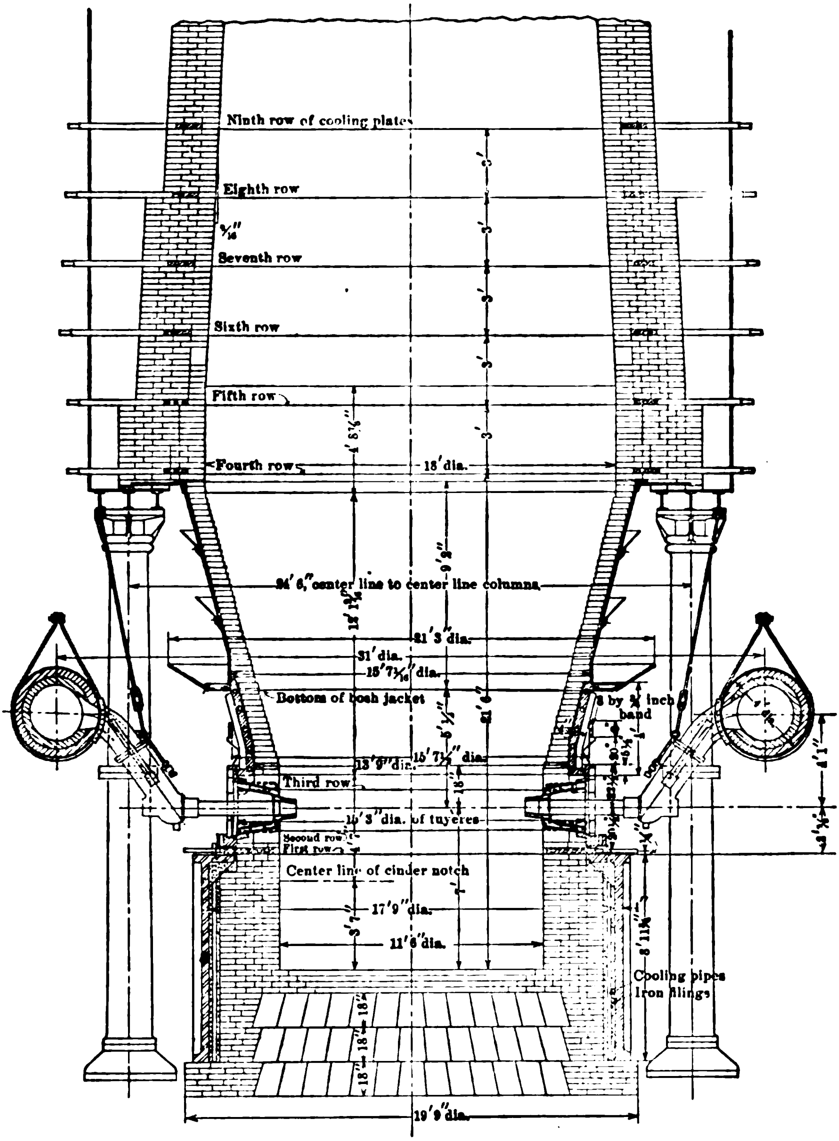


FIGURE 12.—Jacketed furnace in which expansion of bosh is provided for. The bosh jacket in expanding may slide over the water-cooled tuyère-breast jacket.

The rivets are countersunk flush with the surface so as not to present any interruption to the flow of water. From 2 to 10 annular funnels are placed about the jacket to catch the water thrown off and return it to the face of the jacket, or a spiral or helical trough is riveted onto the jacket with tap bolts. These spiral troughs are held away from the jacket by distance pieces (iron straps), part of the water running down over the surface while the rest runs down the spiral trough. This is probably the most efficient means developed for obtaining a positive contact of water with the shell.

FORMATION OF SKULLS.

Even with positive water cooling assured, other factors remain which may destroy the jacket. The skull that forms on the interior of the jacket may peel off and build up intermittently and thus cause warping, cracking, and opening of the jacket, as it is constantly subjected to extreme fluctuations of temperature. Sometimes when the skull slips or melts off, the gas may blow out between the jacket seams and unless immediately stopped may erode a dangerous hole. If these apertures or holes formed in other ways get beyond repair by calking, it is necessary to stop the furnace and put on a patch. Such a patch is difficult to cool and is of questionable strength and tightness, as it must be held externally by tap bolts.

EROSIVE ACTION OF CHARGE.

The occasional tendency of the furnace charge to work on the walls, or the necessity of so running the furnace at times that this condition can not be avoided, is a severe ordeal for jacketed boshes. As an example may be cited a furnace in Alabama, where this tendency proved too severe and the bosh was burned out. In many instances a similar accident has been averted only by the prompt use of longer tuyères. Long tuyères, although effectual in averting the difficulties and possible dangers of jacket construction by reason of excessive slagging of the walls or local cracks developing from intense heat, have the disadvantage of reducing the useful hearth area and changing the descent of the charge so that the burden moves too slow or "goes to sleep" on the walls.

USE OF AUXILIARY SPRAYS.

At plants having jacket boshes it is not unusual to see men engaged in putting on auxiliary sprays to take care of hot spots, deficient regular sprays, corrugations in the jacket, or gas blowing from rivet holes or riveted seams. However, at one plant the jacket bosh is entirely covered by an auxiliary shield to confine splashes of water, although this shield effectually prevents emergency first aid to the bosh, or noting signs of its failure. Although such "running in the dark" seems objectionable, when the difficulties at some other plants are considered, it has nevertheless been found practicable at this plant. However, the fire-brick lining is here approximately twice as thick as in the usual practice.

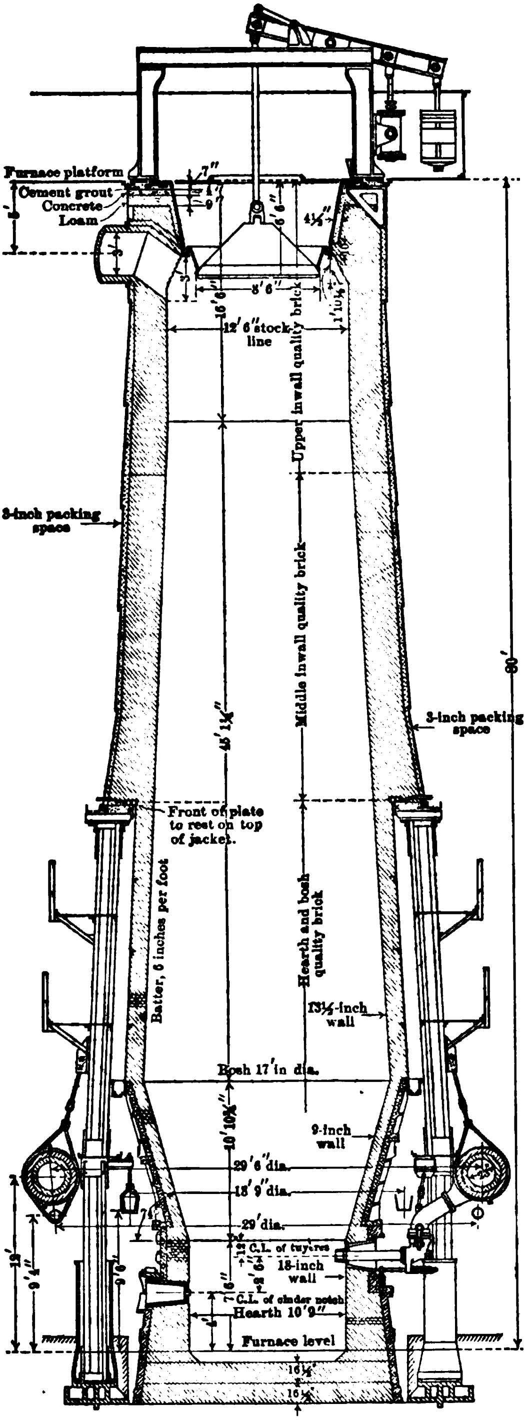


FIGURE 13.—Jacketed bosh built with cast-iron segments and circulating water pipe cast in.

OTHER TYPES OF BOSH CONSTRUCTION.

Several types of bosh construction have been developed to avoid the corrugations of the banded bosh and the warping or local failure of the jacketed bosh. One of these boshes consists of built-up sections of cast-iron segments with the water-circulating pipe cast in (see fig. 13). Another type consists of a riveted steel plate jacket, inside of and next to which are laid cast-iron staves with the water-circulating pipe cast in, which extends from the top to the bottom of the bosh (see fig. 14). These two types may be said to leave something to be desired in that any clogging of the water supply that might cause failure of the sections through overheating is a serious matter. Another type is the Roberts Farrell bosh (fig. 15), in which the entire bosh is inclosed by a steel jacket. Holes are cut in the jacket and through them plates are inserted in the brick lining. Provision is made for anchoring the plates to prevent tilting or ejection. This type of bosh, in modified form, is meeting with considerable favor, as it offers obvious advantages as regards security, structural strength, and operating convenience.

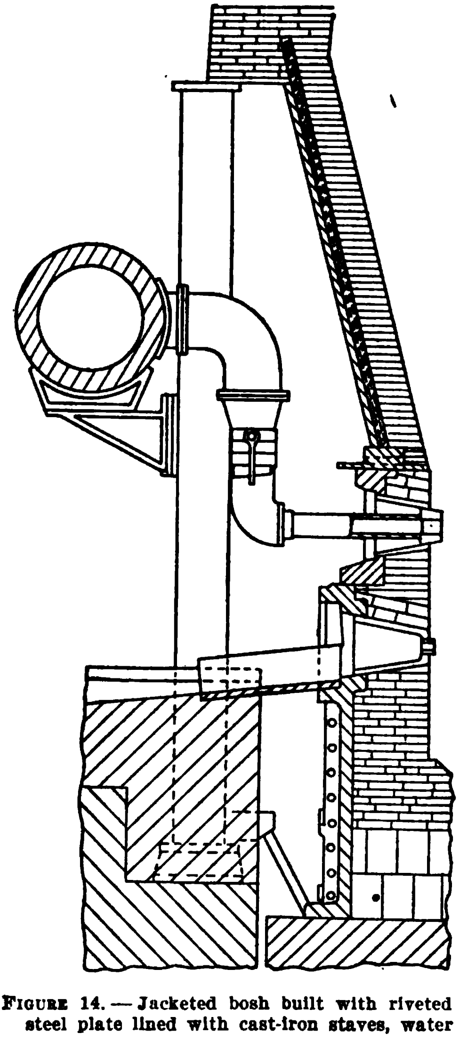


FIGURE 14.—Jacketed bosh built with riveted steel plate lined with cast-iron staves, water pipe cast in.

CAUSES OF BOSH BREAKOUTS.

Before describing bosh breakouts, it will be useful to summarize their causes. The causes of breakouts in banded boshes are as follows: (1) Excessive blast pressure, owing to scaffolding in the furnace; (2) excessive load or pressure caused by columns of stock falling or slipping in the furnace; (3) weak bands; (4) bands not held in position; (5) ejection of plates or brickwork between bands; (6) collapse of bosh by excessive corrosion of the brickwork; and (7) improper design, as where the top line of a weakly banded bosh extends below the mantle.

The causes of breakouts in jacketed boshes are: (1, 2, and 3) As in banded boshes; (4) formation and enlargement of holes through (*a*) rivet holes or riveted seams, or from (*b*) local failure of water cooling, or from (*c*) severe corrosion of walls and segregated melting or combustion adjacent to the walls.

EXAMPLES OF BOSH FAILURES.

BURSTING OF BOSH BANDS.

At one plant a plate-and-band furnace had been working irregularly for four days, the pressure varying 18 to 25 pounds per square inch after casting. About 40 minutes after each cast it was necessary to reduce the pressure to make the furnace move, and thereafter at about 40-minute intervals between the casts, which were at four-hour intervals. The furnace frequently failed to slip at the first check, and it was necessary to put the wind on again for 15 or 20 minutes and then take another check. Various expedients were tried to correct the difficulty, but the furnace persisted in slipping heavily, the tests showing slips of 8 to 30 feet. The furnace had been checked shortly before casting, when the charge moved slightly. The crew went on the drill, got the iron out, and after about one-third of the cast was run, the blowing engines were slackened, thus reducing the pressure from 19 to about 11 pounds. Then the furnace slipped and the bosh burst at the top, directly beneath the mantle. An outburst of flame, incandescent coke, fine carbon, ore, and in fact the entire content of the stack, was ejected as if from a gun, and engulfed a number of men working about the furnace or passing through the cast house. This furnace had 7½-inch by 1¼-inch bands about the bosh, which were broken both at the riveted joints and also between the riveted joints. The strength of the bosh was supposed to be approximately 10 times that of any pressure to which it might be subjected.

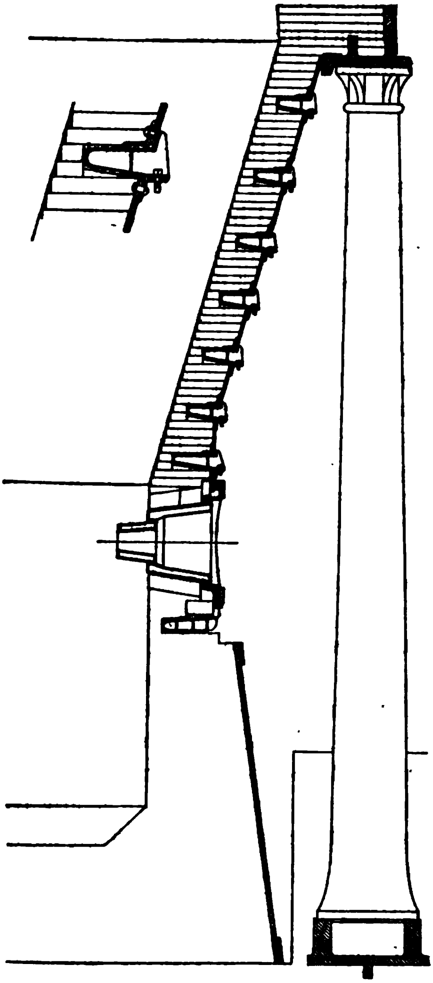


FIGURE 15.—Bosh inclosed in steel jacket with holes for cooling plates.

ESCAPE OF IRON INTO COOLING-WATER DITCH.

Another accident happened at a small plant having a banded bosh furnace, and was presumably caused by a hearth breakout. The iron escaped underneath the jacket and came up in the cooling-water ditch about the hearth jacket. An explosion followed that shook the entire plant and jarred the bands about the bosh so that the bands above the second, third, and fourth rows of plates slipped down. The brickwork, which obviously was under compression from within, was pushed out at one side of the front of the furnace and engulfed two men.

FALLING SCAFFOLD.

A plate-and-band furnace had been working on abnormally high pressures, with a scaffold in it. The scaffold fell and the gas ignited on the bosh and burned fiercely, indicating that a large mass of material had slid off and had thinned the walls. A large quantity of excess coke and sand, which had been charged previously to the slip, had about arrived at the bosh, and as it melted began to exude through the joints of the brickwork. Several water hose were being played on various parts of the bosh when at one side the brickwork between two adjacent plates and bands of one row blew out. The bricks and following flame and coke fatally burned a man who was standing in its path.

FAILURE FOLLOWING HOT SPOTS ON JACKET.

At a furnace built with a jacketed bosh, trouble had been caused by hot spots on the jacket, owing to the tendency of the furnace to burn on the walls, there being presumably a core of cold stock in the center. The bosh blew out at the rear, the furnace contents running out like water. Considerable damage was done to the plant but no men were injured. The failure was probably due to melting of the plate work and not to weak construction, because the furnace, aside from making raw cinder and off-grade iron, was not working abnormally or on high pressure, or making severe slips.

FAILURE FROM BURSTING OF RIVETED SEAM.

At another furnace which had a jacketed bosh, the jacket burst vertically along a riveted seam because of a heavy slip, and the furnace contents caught and killed a foreman and four workmen.

FAILURE FROM A HEAVY SLIP.

At an old stack, originally built with a high bosh, the top of the bosh was lowered on a relining 6 feet 9 inches below the mantle plate, the brickwork being inclosed by a vertical steel jacket of ½-inch riveted plate hung from the mantle. This jacket was reinforced on the outside with four 6-inch by 1-inch bands and had three rows of cooling plates inserted. Below the jacket the bosh was of substantial banded construction. The upper part—that is, the banded steel jacket with 2 feet of fire-brick lining—burst in consequence of a slip, filling the cast house with a mass of white-hot coke and flame. The furnace had not been checked, and the pressure chart, which stood at 14 pounds, did not indicate any increase in pressure at the time the bosh burst. The failure probably was due to the impact of a falling column of stock on the contents of the bosh.

FUTURE HAZARD FROM BOSH BREAKOUTS.

In the trade journals from 1885 up to 1907 frequent mention was made of bosh breakouts; since 1907 they have steadily decreased. Rarely are any details given in these notices, and they are mentioned here only because of the significance of their decreasing number.

At plants which are in nearly continuous operation, with relining and reconstruction of the bosh at three to five year periods, the strength of the bosh has generally been brought to a point where the factor of safety is large and so far as managers and engineers can design with former failures in mind, or anticipate future requirements in view of the trend of raw materials and practice, such plants are free from any dread or possibility of bosh failures.

At plants that operate chiefly in times of good market conditions some boshes are liable to be deficient in strength and armoring, and with heavy driving bosh breakouts may be expected.

TUYÈRE-BREAST BREAKOUTS.

CONSTRUCTION OF TUYÈRE BREAST.

The tuyère breast is comprised in that part of the hearth which extends from the bottom of the bosh to the top of the hearth jacket. The brickwork at this point is 18 inches to 2 feet 9 inches thick and is built of standard 9-inch brick inclosing the tuyère-cooler arches and cooling-plate openings. The tuyère breast in a modern 500-ton furnace is about 5 feet 6 inches high and the outside surface, with a 2-foot 6-inch wall inclosing a 17-foot hearth, 22 feet in outside diameter, is about 390 square feet, which is diminished by 12 cooler openings 30 inches in diameter and usually by 36 cooler-plate openings 5 inches by 24 inches in size, making a total exposed surface of approximately 330 square feet.

FACTORS TENDING TO DESTROY THE BRICKWORK.

SOFTENING OF THE BRICKWORK.

The most energetic combustion of coke and the strongest inthrust of the blast are directly within this part of the furnace. The temperature at the nose of the tuyère may reach 3,300° F., and may be as high for a considerable space adjacent, because the combustion of 750 pounds of coke can not be instantaneous at the 12 or 16 small spaces in front of and above the tuyères. As the nose of the tuyère is but 6 to 15 inches beyond the face of the brickwork the combustion temperature must reach nearly maximum intensity at the face of the brick. This temperature is sufficient to melt any but the best hearth brick and to soften the best brick so that it becomes plastic or will flow under compression.

EROSION BY THE BLAST.

To the effect of this high temperature must be added the effect of the blast. In an average furnace 40,000 cubic feet of air, expanded to 60,000 cubic feet at 1,000° F. and at 15 pounds blast pressure is forced in each minute through twelve 7-inch tuyéres at a velocity of 310 feet per second. As the blast emerges from the nose of the tuyère with this high velocity it impinges on the mass of coke in the hearth. As this coke is held down by the weight of the furnace burden, the only way of escape open to the gaseous products of combustion is through the interstices between the pieces of coke. In spite of the great weight of the ore, limestone, and coke the force of the blast makes the coke in front of the tuyères and presumably that in the interior of the hearth above the tuyères and along the walls dance briskly. On seeing this movement at the tuyères one may imagine that the blast is thrusting the coke, which is still strong and harsh, against the hearth walls, and this is found to be the case on stopping the furnace. In some instances the hearth lining above, below, and between the tuyères has disappeared for a depth of 6 to 18 inches and may even be cut to so thin a shell that the furnace has to be blown out.

SLAGGING OF THE BRICKWORK.

In the lower part of the tuyère breast the accumulated slag lying in the hearth attacks the brickwork. In normal work this slag rarely comes within 1 foot of the tuyères, although it occasionally perplexes furnace men by persisting in lying in front of the tuyères and running back in them, when no cinder can be drawn from the cinder notch below. In this zone of slag accumulation there is, of course, slagging of the brickwork and thinning of the walls.

PRESSURE WITHIN TUYÈRE BREAST.

The chief factor in corrosion of the tuyère-breast lining, however, is held to be erosion by the cutting action and back thrust of the blast and the particles carried in it at the high temperature and pressure of the hearth.

If it be assumed that the hearth has a diameter of 16 feet and inclosing walls 31½ inches thick, and is 4½ feet high, then the inside surface is 226 feet square and the exterior surface is 301 feet square. The interior surface is reduced by 12 tuyère openings to 185 square feet and the outside surface to 242 square feet. With a blast pressure of 15 pounds at the tuyères there is a drop in pressure inside the furnace which amounts to approximately 2 pounds, so that the pressure inside the hearth may be taken as 13 pounds. Under the most unfavorable conditions of hanging the pressure might mount as high as 30 pounds. The total lateral pressure which tends to produce rupture on a circular area 1 inch high about the tuyère breast may then be: $P=d\times p$, where d=diameter and p=pressure. $P=16\times 12\times 30=5{,}760$ pounds pressure, which is the total force on a longitudinal plane against the 1-inch high circular area about the tuyère breast.

This pressure, exerted on a wall built up of small bricks, 2½ inches thick by 4½ inches wide by 9 inches or 13½ inches long, laid without cement other than a thin seal of fire clay between the joints, may conceivably displace and force out segments of brickwork, although in fact such failure does not often occur. The writer has seen furnaces at which the brickwork at the tuyère breast had started and was held from bulging out farther and causing a breakout by being braced against the column with wooden blocking.

USE OF COOLING PLATES IN TUYÈRE BREAST.

The danger of breakouts between the tuyère coolers, when the brickwork gets thin from melting, slagging, and mechanical abrasion from the back thrust of the gas, is usually provided against by the insertion of bronze cooling plates. These plates are usually built solidly into the brickwork from the outside face to within 9 inches to 4½ inches of the inside face. Occasionally only one plate placed vertically is used; more often two plates are laid horizontally one above the other about 15 to 18 inches apart, center to center; infrequently three plates 12½ inches apart, center to center, are used. The plates are roughly of the same type as those used in the bosh, being tapered horizontally and vertically. Although the wearing action is most severe, such plates have proved entirely satisfactory. A few managers go a step further and provide water-cooled plate boxes in which to place the plates, but many furnace men do not consider it necessary or advantageous to do so, as a water-cooled plate cools the brickwork just as effectually and also prevents thinning and breakouts of the tuyère-breast wall.

Figures 16 and 17 show clearly how the insertion of cooling plates between the cooler arches reduces the noncooled part of the tuyère breast and lowers the temperature of the brick below the slagging point before it has been reduced to an unsafe thickness.

PROTECTION OF BRICKWORK ABOVE THE TUYÈRES.

That part of the brick lining above the line of the tuyères is similarly cooled by radiation of heat to the lower row of bosh plates above and to the water-cooled tuyère coolers and the tuyère-breast plates between the coolers. The vertical distance between the tuyère cooler and bosh plate is, on an average, about 16 inches on the inside face; the distance between the tuyère-breast plate and bosh plate is 15 or more inches. This obviously leaves only a small volume of brickwork unprotected.

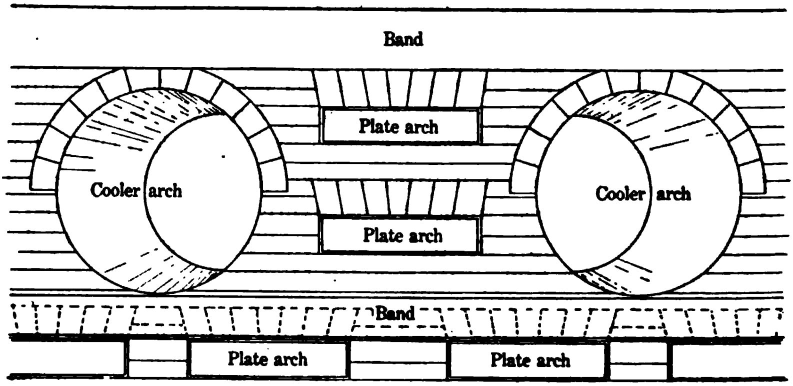


FIGURE 16.—Sketch showing construction of tuyère-breast wall with cooling plates between cooler arches.

PROTECTION OF BRICKWORK BELOW THE TUYÈRES.

That part of the brick lining below the tuyère coolers is under a rather severe load, as it is subject to the slagging action of cinder, which accumulates and rises along the hearth walls and is then drawn off, and also to the back thrust of the blast. To prevent severe erosion this part of the hearth walls is almost always copiously cooled. As a general rule two plates per tuyère are inserted at this point, or 24 plates for a furnace with 12 tuyères. On a 500-ton furnace the plates are usually placed 32 to 36 inches below the center line of the tuyère, being spaced horizontally about 3 feet center to center. As the plate opening is about 2 feet wide, there is only 1 foot of brickwork horizontally between centers. The plates are set back about 9 inches from the inside face of the lining, the idea being that this depth will be sufficient to obtain an equilibrium between the cooling effect on the brickwork and the attack of blast and slag, so that a sufficient thickness of brick will be left in front of and above the plates. However, such a result is almost never attained. Examination usually shows that at one or more periods when corrosive conditions have been unduly severe the brickwork has disappeared and been replaced with graphitic carbon, slag, and other substances, so that although the plate may have been naked intermittently, it may be assumed to be insulated normally by furnace deposits. It is rather essential that some covering about the plate be preserved, as sometimes labor conditions, slowness in handling ladles in the yard, or hard tapping holes delay the cast so that iron accumulates in the hearth and reaches the level of the cooling plates. Unless the plates are insulated from the molten iron by a scab of slag or graphitic carbon they are likely to be cut and explode when the water reaches the fluid iron.

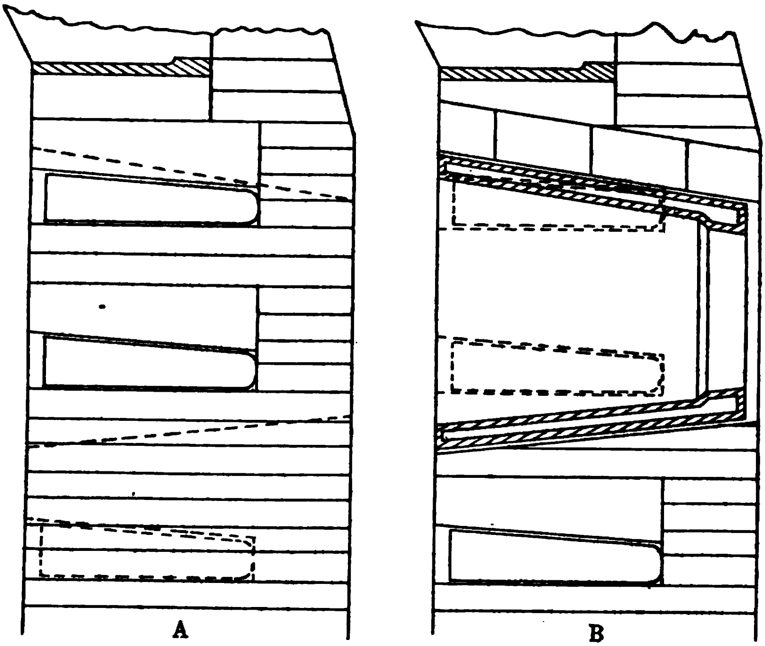


FIGURE 17.—Sectional view of tuyère-breast cooling plates.

Such plates are usually cooled by connecting them below and between the cooler arches in series, the inlet being at the first lower plate, thence to the second, and then through the two upper plates between the coolers.

OMISSION OF PLATES BENEATH THE TUYÈRE COOLERS.

A few plants do not insert the plates below the coolers, but bring the water-cooled hearth jacket up to the bottom of the cooler arch, thus relying on exterior cooling. Though the wall becomes much thinner before a cooling effect sufficient to arrest erosion is established, officials of these plants claim immunity from explosions of plates or breakouts at this point.

The objection to this construction is that the brickwork beneath the tuyère coolers becomes so thin that only an insecure foundation is left to support the coolers, and they may drop down or failure may develop beneath them. This is a valid objection, for the weakest place in a furnace equipped with cooling plates is immediately about the cinder-cooler opening, where at times the plates must be left out because of lack of room. Cinder breakouts occur at this point, between the top of the hearth jacket and cooler arches, not infrequently at some plants. There is, of course, the additional effect of increased erosion from the blast when the furnace is "blowing on the monkey" (flushing cinder when the blast blows through with the slag).

USE OF PLATE BOXES.

At a few plants boxes similar to the plate boxes in the bosh have been used at this point, but they are not satisfactory because they require cooling with sprays on the inside to prevent cinder breakouts. Evidently there must be cooling from the outside face. In Southern practice, when these plate boxes are used with outside cooling also, they are cast with hollow walls in which water is circulated. This arrangement has proved satisfactory, but probably is not more so than a simple bronze plate.

REINFORCEMENT OF BRICKWORK.

Although there are a few plants at which the entire tuyère breast is inclosed in cast steel or iron segments, cooled by water circulating through pipes cast in them, the tendency seems to be to inclose the tuyère-breast brickwork in a jacket of steel plate, holes being cut in it for cooler and plate openings. This jacket extends from the hearth jacket up to the bottom bosh-band and makes a breakout of brickwork literally inconceivable, provided it is water-cooled by plates.

Where the brickwork is held in only by the lower bosh-band and a band between the cooler arches and bottom row of bosh plates, there is nothing to prevent displacement of the segment of brick between the coolers and bands except the binding effect from the weight of the bosh above and the wedging effect of the cooler arches. The pressure is applied on the under or concave side of the arch, which obviously offers no resistance other than its own weight and inertia. Such construction is not safe.

The reason that unreinforced brickwork on the tuyère breast is unsafe is apparent from the previous statement (p. 25), in which it was shown that the pressure applied to the inside of the brick wall amounted to, in an extreme case, 5,760 pounds, which tended to move the walls outward and to produce rupture. If the resistance of the built-up brick wall to this rupture be ignored, and its resistance may in fact be slight, it follows that a reinforcing jacket about the tuyère breast is necessary. These are almost universally used nowadays on large furnaces, and when built up of steel plates are ½ to ⅞ inch thick. The factor of safety in blast-furnace construction is very large at this point, as may be seen from the following figures:

$$T=\frac{p \times d \times f}{2 \times S}$$

T=thickness of plate.
p=pressure in pounds per square inch.
d=diameter in inches.
f=factor of safety.
S=tensile strength of plate.

$$T=\frac{30 \times 192 \times 10}{2 \times 55{,}000}=0.52 \text{ inch.}$$

When instead of a tuyère-breast jacket only two 12-inch bands are used, one immediately below the bosh and just above the cooler arches, with the other just below the cooler arches and a short distance above the hearth jacket, then if the tuyère breast be 4½ feet high, it is evident that the bands will cover but four-ninths of the total area and should therefore be made correspondingly heavier than the tuyère-breast jacket. For example, if a factor of safety of 10 requires a thickness of tuyère-breast jacket of ½ inch, then if the duty of the jacket is to be carried by two bands like the ones mentioned above, they should be 1⅛ inches thick by 12 inches wide. As a rule these bands are 1 inch to 1¾ inches thick, and the lower band is often made the heavier of the two. The reason for this is that the lower one carries most of the load unless buckstays are placed about each cooler arch and held by the bands to prevent bulging of the brickwork. Without such buckstays in addition to the two bands, it is apparent that each cooler inserted in its opening, which has the shape of a truncated cone, forms a pier that receives the thrust of adjacent brickwork resulting from its expansion and its tendency to be forced by interior pressure. The cooler is primarily held in place by the pressure of the blowpipe against the tuyère stock. The stock is held by the bridle rod, which is attached to the bridle post, and the post is fastened to the lower band, which thus prevents displacement of the brickwork about the cooler. This is obviously a roundabout way of transmitting the thrust of the cooler to the band, and as a rule the cooler is anchored by inserting a forged steel piece between the cooler and the bridle post. When resistance to internal pressure is provided by vertical buckstays between the bands, the cooler is anchored to the buckstays, which is preferable in that it distributes the load over both bands. Another practice is to provide a second auxiliary bridle rod between the tuyère stock and the upper band. (See fig. 18 and Pl. II.)

PACKING OF TUYÈRE COOLERS.

In addition to the possibility of breakouts from the brickwork being displaced, cut through by the blast, or slagged away by cinder,

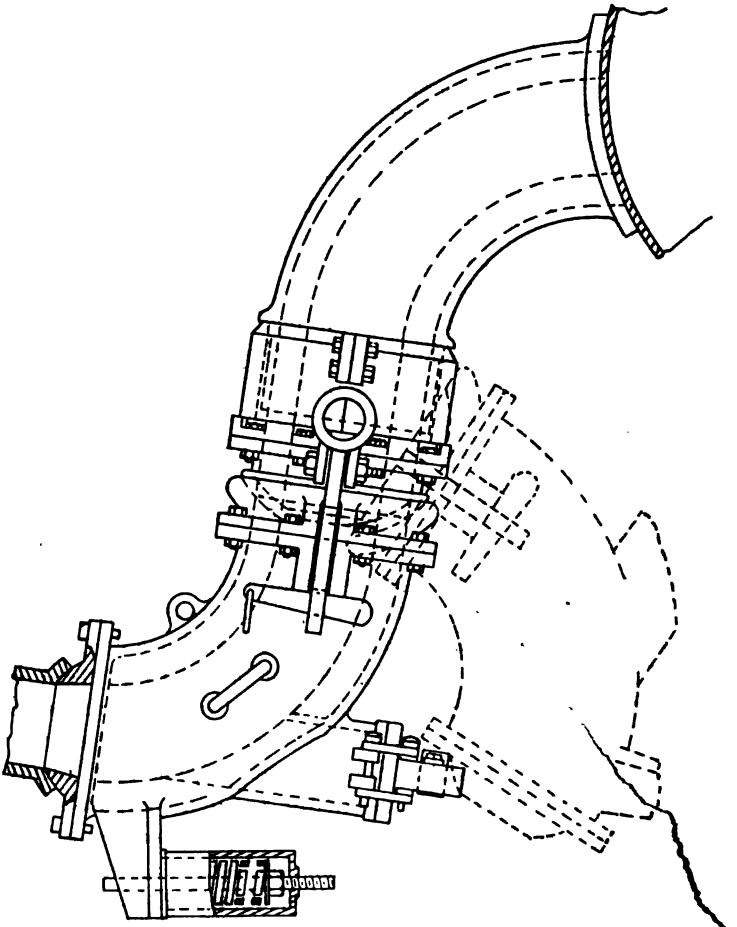
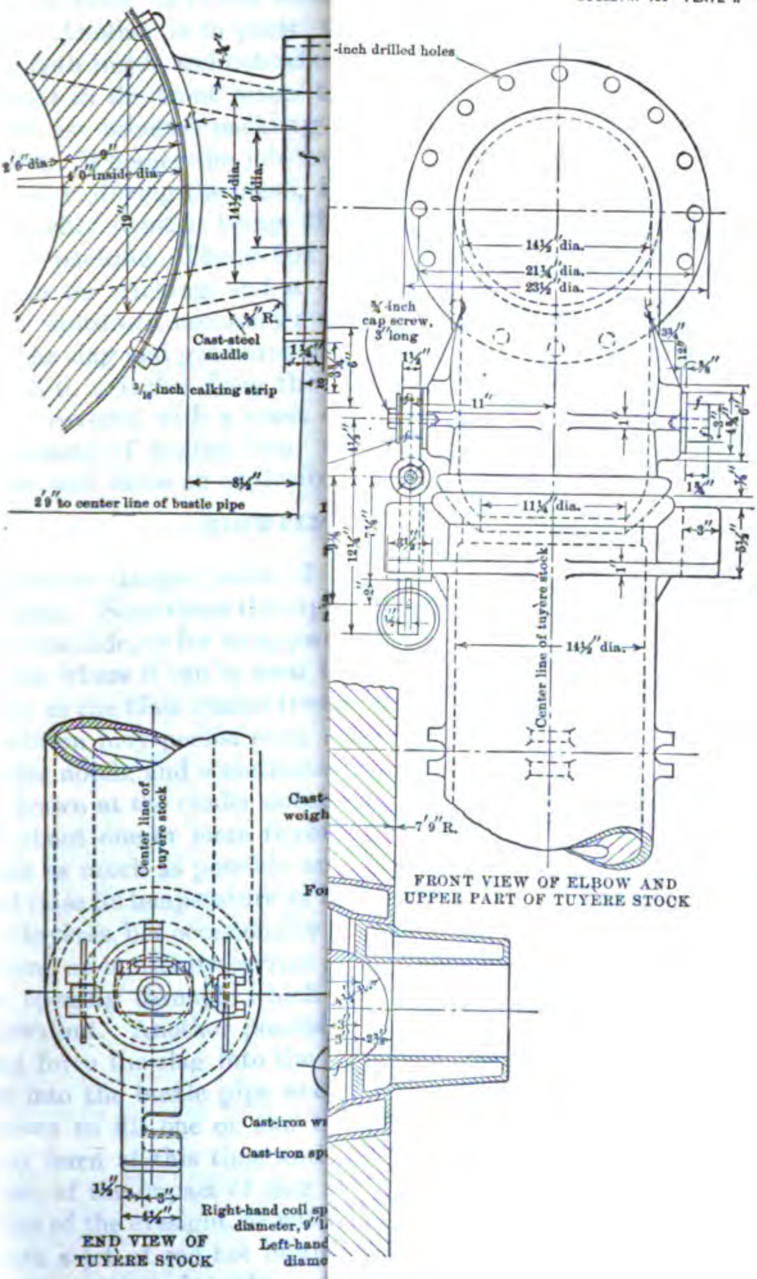


FIGURE 18.—Tuyère stock, showing lug for auxiliary bridle rod.

there is also a possibility of cinder breakouts between the brickwork and tuyère coolers, which is only minimized by good packing. The coolers are never made to fit snugly in the cooler arches, because it is impossible to get the necessary continuous contact and because the cooler arch may not give the cooler a true position if it must fit in it snugly between the brick and cooler surface by this means, so a packing space of 1 to 1½ inches is left. The coolers are set before



FRONT VIEW OF ELBOW AND UPPER PART OF TUYERE STOCK

END VIEW OF TUYERE STOCK

starting the furnace and are usually packed much more tightly than is possible when they must be replaced later on. One method, not much in favor, is to run neat cement in about the cooler and brickwork. Another is to pack the space with clay, tightly rammed in from both inside and outside the furnace; sometimes asbestos rope is rammed in the front space to make an extra tight packing. A few plants use asbestos packing throughout. The plates are similarly packed. Whether the job is done on relining, or, as is occasionally necessary, during the blast, it is recognized that the packing must be thoroughly done, it being the poorest economy to save time by improper packing. This is for two reasons; the blast will find the weak spot in the packing, and a cinder breakout will follow, making another shutdown necessary; or the weak spot will yield from a slip and the slag and gas burst out. The cooler nose is generally set back ½ inch to 2 inches from the inside face of the lining, so that it becomes covered with a crust of graphite and slag and is not exposed to pockets of molten iron, which will cut it, come in contact with water, and cause an explosion.

BLOWPIPES AND EYESIGHTS.

Another danger point of a slag breakout is in the blowpipe and eyesight. Sometimes through a furnace working cold on the bottom, or to one side, or for no apparent reason, cinder will lay in front of the tuyère where it can be seen through the eyesight, tossing and rolling away as the blast rushes from the tuyère nose into the furnace. This condition may persist even after repeated draining of slag from the cinder notch, and sometimes only gas and a few globules of slag can be drawn at the cinder notch while a large volume of slag is banking up about one or more tuyères. The usual remedy is to reduce the blast as much as possible and still keep the slag from running back and raise its temperature as high as possible. This sometimes "dries" the tuyères, but occasionally the slag suddenly runs into the blowpipe, where, as it usually carries metallic iron, it cuts through and makes an opening through which a large volume of slag, coke, and gas blows out. Another possibility is that the furnace burden will slip and force the slag into the blowpipe. Sometimes the slag is driven up into the bustle pipe over into the hot-blast main, and has been known to fill one or two hot-blast valve chambers. The blowpipe may burn at this time and cause a similar or worse "mess." The force of the impact of slag against the pipe and stock may burst the glass of the eyesight, or with old types of eyesights, throw them open, when a jet of red-hot liquid slag will spurt out, as if from a high-pressure pump, into the cast house.

There is no adequate safeguard to prevent slag coming into the blowpipes, although, of course, many flap valves and relief valves have been patented which have never been long used because they are without exception intricate or difficult to keep in working order. The blowpipes are made as heavy and with as thick walls as possible compatible with the necessity of taking them down and putting them up by hand almost daily. For this reason the pipe walls should not be thicker than seven-eighths inch, which is sufficient to withstand a very severe attack, and are usually one-half inch thick. They are usually made of the best grade of Bessemer iron, as a poor grade will bend and cut more easily. A few plants use blowpipes made of lapwelded steel pipes. Such pipes may be cheaper under local conditions, but are not as resistant to attack by slag and iron.

TYPES OF EYESIGHTS.

The eyesight has received much attention, and several safe and excellent types are now in use. It is to be kept in mind that when the furnace is being taken off at cast, or checked, it is usually necessary for some one to watch the tuyères to see that cinder does not enter them. If the slag is suddenly dashed back, the glass of the eyesight may be shattered or the eyesight may be pushed from its seat, and the observer be caught and burned.

Obviously, the elementary precaution is to have a glass eyesight, but there are plants at which the keeper, blower, or helper opens a blind sight and, with the blast blowing in his face, attempts to peer into the tuyère. Because of the difficulty, and the danger of flying dust getting into his eyes, it is apparent that the watcher may devote minimum attention to the working of the furnace at the tuyères, thus increasing the possibility of a blowpipe burning.

The size of the aperture for the glass need not be large; three-eighths inch is as near standard as any other figure, although many are twice as large. It may be as small as one-eighth inch with certain types of eyesights, and the smaller the aperture the less chance of the glass flying out if a splash reaches it or if it should crack from any other cause. Many furnaces use an eyesight (see fig. 19) one-eighth inch in diameter, and it affords a good view of the entire surface of the tuyère. The farther back from the tuyère cap the eyesight is, the larger the eyesight opening may be without danger of breakage from coke or slag. An example of such an eyesight is shown in figure 20.

Both these types are also safe from being thrown open. A third type is shown in figure 21, and a fourth in figure 22. A type still in use at many plants in western Pennsylvania is shown in figure 23. This type, being held down only by the counterweight *a*, can be thrown open on severe slips to allow cinder, coke, or gas to blow out; so that it is disliked in other districts.

EXAMPLES OF TUYÈRE-BREAST BREAKOUTS.

In the following paragraphs are given instances of tuyère-breast breakouts.

1. At a plant where the iron was cast in beds, both beds were full, owing to trouble with the iron carriers, and the furnace had gone beyond the usual casting time. Cinder had been flushed several times, and at the last flush the iron had risen so high that it came over the "monkey" (or cinder notch) and burned it. As it was thought that the bed would soon be cleared, the blast was kept on until cinder showed at the tuyères. Under these conditions the blast could not be taken off without the blowpipes becoming filled with cinder, nor could cinder be drawn off at the cinder notch for fear of the cinder-notch cooler exploding. About two hours after casting time one bed was cleared and the crew went on the drill, when the furnace wall collapsed near one of the tuyères next to the tapping hole, throwing out the brickwork together with molten iron and slag. Several men were injured. The furnace did not slip, had been working regularly, and the accident was evidently due to failure of a presumably thin wall under too heavy a load. The tuyère breast was held in by bands above and below the coolers and had no cooling plates in the brickwork.

2. The furnace at another plant had been working badly and scouring the walls, while cinder and iron kept coming into the blowpipes. The iron finally ran out between the cooler and brickwork, where it cut the bronze and exploded on contact with the cooling water. The cooler was blown out with great violence; slag, coke, and gas shot from the furnace and killed two men. This accident may be regarded as purely accidental and a hazard of the work.

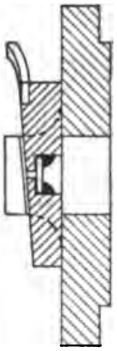


FIGURE 19.—Eyesight with ⅛-inch aperture.

3. A furnace, which was hanging, slipped after about two hours; a section of brickwork blew out between the cooler arches, bursting the band. Incandescent coke and cinder blew out above the tuyère and burned a helper fatally.

4. In another instance a blowpipe was loose and the hot-blast man started to tighten the nut on the bridle rod. The bridle post was a round pin which, under the added strain, evidently turned in its socket, allowing the bridle rod to slip off. The blowpipe dropped immediately and the man was caught in a sheet of flame from the tuyère.

5. At another plant the furnace walls beneath the tuyères were eaten away by the slag and blast. and failed at a place near the

cinder cooler. The cinder with melted iron came through, caused some damage by cutting the iron hearth jacket, and, cutting a large hole, flowed out over the cast-house pavement. No men were injured. After the mess had been cleared away, fire clay and crushed brick were rammed into the hole, and a 9-inch wall was built on the face and wedged in tightly. A small water spray was put on and the furnace started. No more trouble was experienced. This incident is typical of the usual slag breakout, and many other examples could be given.

6. A furnace was being checked under difficulties as it was "sloppy" at several tuyères and the cinder tended to run into the blowpipes. The foreman had checked the furnace once and was stepping from eyesight to eyesight to watch the tuyères; the helper was also similarly engaged. The helper suddenly shouted, indicating that cinder was coming back, and jumped aside. The foreman sprang to the whistle switch to signal for the blast to be put on, and while standing at the switch for the brief instant necessary, the blowpipe burned through and he was struck on the back with a shower of coke and slag and burned severely.

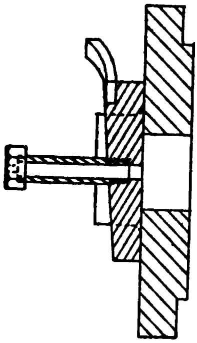


FIGURE 20.—Eyesight with glass set back from tuyère cap.

7. At another furnace there had been difficulty in taking the furnace off at cast. Cinder came into the blowpipes, and the snort valve could not be opened. As a fierce flame showed at the tapping hole, preparations were made for a quick stop, the blast was thrown off, and the men swung the nose of the mud gun into the hole. At the same instant, cinder and iron came into the blowpipe, which was above them, burned through, and caught two men, causing severe burns on their faces, arms, and bodies.

8. In another instance iron accumulated in the hearth of a furnace and rose up in contact with the cooling plates beneath the tuyères. An explosion, caused by a leaky plate or by the molten iron cutting the plate, threw the plate about 200 feet from the furnace. A shower of molten iron followed, but fortunately no men were injured.

9. A similar explosion blew out the side of a furnace above the hearth jacket.

10. A breakout of gas, coke, and cinder from a tuyère opening was caused by a lug, cast on the gooseneck breaking off. This allowed the stock hanger to fall; then the stock swung aside, the blowpipe dropped, and the furnace contents blew out. No one was injured. A similar accident occurred when a bridle rod burst and allowed the blowpipe to drop. In this case a workman was killed. Both acci-

dents happened without warning and in absence of any such work being done as tightening the stock hangers or bridle spring.

11. The hazard from occasional duties about the furnace is shown by an accident in which several men were severely burned. The furnace had been blown out and after the last cast the crew were engaged in taking down the blowpipes, two gangs being at work. Without warning, a mass of material, presumably a scaffold which had become loosened by the heat and water, incident to blowing out, fell into the hearth and blew a cloud of incandescent carbon dust and gas from the tuyère openings. A similar accident which occurred at another plant cost several lives.

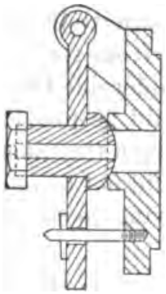


FIGURE 21.—Eyesight, keyed type.

PREVENTION OF TUYÈRE-BREAST OR BOSH BREAKOUTS.

IMPORTANCE OF CORRECT CONSTRUCTION.

Safe methods of practice are of little avail in preventing tuyère-breast or bosh breakouts, if by faulty design or construction, weakly built, insufficiently reinforced, or improperly cooled segments of brickwork have been incorporated in this part of the furnace. Modifying methods of practice is of little benefit because of the suddenness with which such accidents occur. Possibly 95 parts of prevention lie in construction and 5 parts in experience, resourcefulness, and arrangement of the cast house for accessibility of signals and possibilities of escape.

COOLING.

The bosh plates should be flushed out regularly to prevent loss of cooling efficiency from sediment depositing in them. Hot spots on the bosh jacket should be remedied by putting on additional sprays. The use of longer tuyères is effective in correcting any tendency to local heating on the side of a bosh jacket. A longer tuyère also diminishes the tendency to cutting about the bosh plates, between the tuyère arches, on the tuyère breast walls, or near the cinder cooler. At best longer tuyères, tuyères with a smaller nose, or plugged tuyères are but temporary expedients and if they must be persistently used to prevent fast or abnormal working on the walls and severe erosion, their use is likely to produce off-grade iron and dangerous conditions of scaffolding, hanging, and slipping.

REGULARITY OF FURNACE OPERATION.

This brings the discussion into the realm of stock distribution, location of gas offtakes, blast distribution in the hearth, burdening and fluxing, lines of the furnace, use of high heat, regular blast, and the various and many factors contributing to uniformity in practice, freedom from slipping and scaffolding, and long life of the lining. These factors are not without relation to bosh breakouts, but must be passed by with the statement that careful operation is a requisite for prevention.

ARRANGEMENT OF CAST HOUSE AND SIGNALS.

The arrangement of the cast house should provide for security in case of bosh breakouts. Means of exit should be provided about the furnace itself, on at least two sides, and in addition there should be an exit from the foot of the cast house. If the cast-house floor is at a considerable height from the yard level, steps or, still better, inclines, should be provided with strong railings. There should be a shield between the snort-valve lever or wheel and the furnace, and the wheel should not be in direct line with a tuyère opening. In addition to the blowing-room signal at the snort-valve wheel an emergency signal should be placed at some other point about the cast house, preferably in some unexposed place outside—for instance, near and behind the No. 1 stove. When this is inconvenient, the emergency signal can be placed in the cast house on the opposite side of the furnace from the one in common use or immediately outside a doorway. A noteworthy safety device is an auxiliary arrangement by which the snort valve on the cold-blast main may be opened from beneath, independently of the cast-house lever or wheel.

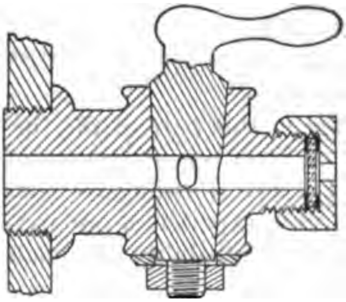


FIGURE 22.—Eyesight, stop-cock type.

OTHER PRECAUTIONS IN CAST HOUSE.

With these arrangements made the last precaution is to insist that members of the force not belonging to the cast-house crew must not make a habit of walking through the cast house in proximity to the furnace. When the furnace is working irregularly, slipping heavily, or being checked after hanging, men should be warned to keep away from the front, from underneath blowpipes, and to use every precaution and be alert when watching the tuyères.

HEARTH BREAKOUTS.

During the past few years serious breakouts occur more frequently at the hearth than at the bosh and tuyère breast. In fact, this has always been so, but with small amounts of iron in the hearth the breakouts were not necessarily serious, especially as the blast pressure was not high. With increasing tonnage and fast driving, they assumed serious proportions, sometimes wrecking the furnace, occasionally costing lives, and almost always causing bad messes, delays, and inconvenience.

BREAKOUTS THROUGH THE HEARTH JACKET.

The destructive agencies within the hearth walls are as follows: (1) Erosion of the hearth walls by the blast, especially over the tapping hole; (2) disintegration of the brickwork by the chemical action of liquid slag and iron; (3) mechanical action of fluid iron in penetrating the joints of the brickwork. The result of these attacks is indicated in figure 24.

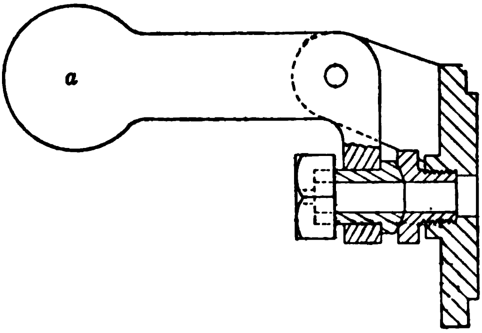


FIGURE 23.—Eyesight with counterweight.

It is evident that the bringing down and accumulation of molten iron and slag in a zone of intensive combustion, pressure, and heat require exceptional construction. Improvement in strength, insulation, and cooling has developed this part of furnace design to a point where plants at which one or more breakouts always occurred during a blast, have reduced the danger of breakouts to an almost negligible factor.

THINNING OF THE HEARTH WALLS.

One point should be kept in mind in regard to hearth breakouts, and that is—regardless of the thickness of hearth walls or the cooling intensity on the exterior, all furnaces become worn away on the inside in much the manner shown in figure 24, so that the wall becomes relatively thin and retains its form only by virtue of the cooling and reinforcement of a metal jacket. This condition is found in furnaces blown out even after a short run. The study of the cause of this condition and of means of holding the hearth walls after it has developed provide a basis for understanding the occurrence of breakouts.

ABRASION BY THE BLAST.

The thinning of hearth walls is partly attributed to the erosive action of currents of gas, caused from combustion and the blast, when the cinder and iron is low after a cast. Erosion is effected as in the bosh and tuyère-breast walls, and is especially active when at casting or flushing the gas sweeps down along the wall and blows out of the tapping hole or cinder notch. Although the temperature may be theoretically too low to melt fire brick, it is high enough to soften it so that it may be abraded by coke ground against it by the back thrust of the blast.

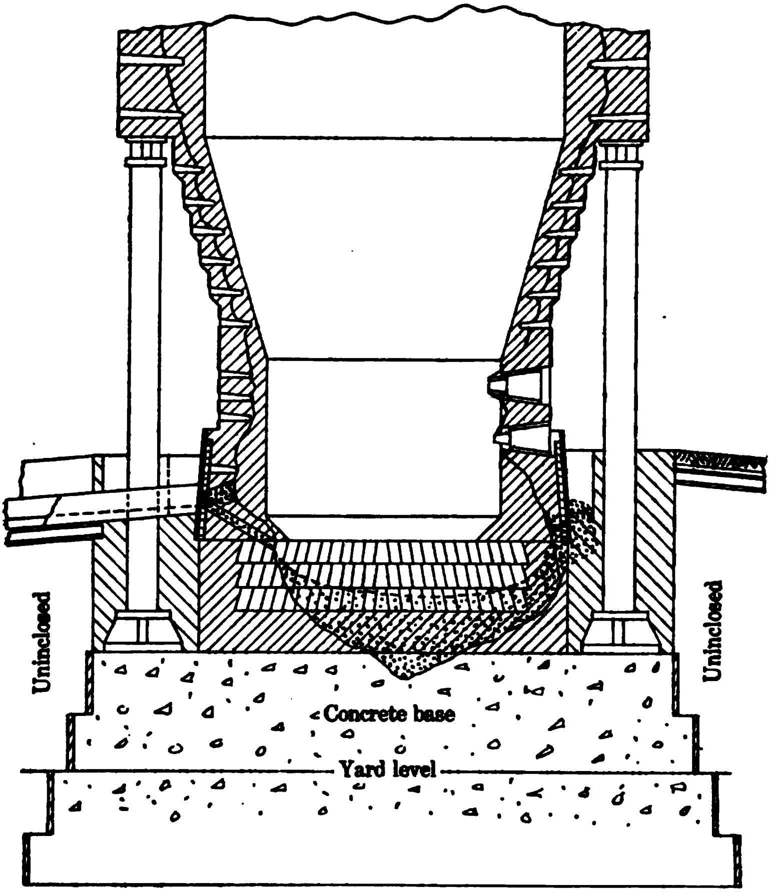


FIGURE 24.—Diagram showing how hearth walls are attacked.

ATTACK OF SLAG.

Added to this erosion is the attack of slag. While the composition of blast-furnace slag is frequently acid, that is, rich in silica and alumina, nevertheless even when viscous with silica, it will combine with avidity with more silica. This is because slags of even pronounced acidity, as regards furnace requirements, are basic from a mineralogist's standpoint. Of greater significance, however, is the physical state of the slag and its effect on the facility with which the slag will attack brickwork. A thinly liquid slag will penetrate the interstices and joints of the brickwork more easily than will viscous aluminous or mucilaginous limy slags. Such slags tend to adhere to the walls and to freeze in cavities, whereas a fluid slag is more likely to drain from the walls at each cast and flush, and on rising it again attacks the exposed surface of the constantly enlarging cavities in the brickwork. All refractory materials tried are destroyed by the physical-chemical activities of heat, erosion, and slag, and with a sharp slag and highly heated furnace, corrosion will bring about an unperceived destruction of hearth walls in a very short run, the hearth walls sometimes losing 3 feet of thickness.

This action continues to the point where the chilling effect of the water-cooled hearth jacket is sufficient to prevent softening or slagging of the brick. Erosion is also retarded by the separation of graphite from the iron, but this factor probably does not assume much importance until the hearth wall has been cut back to a point where the cutting action of the blast is less vigorous than in the plane 12 to 18 inches back from the nose of the tuyère and where the chilling effect becomes more intense. This is indicated by the contour of the hearth walls of furnaces blown out after a short run. In one case the hearth walls of a furnace in blast only six weeks showed a depth of erosion of 22 inches on a 31½-inch wall at the deepest point. This is as much as is found in some furnaces in blast for a number of years. After the destruction of the hearth wall has been arrested by the cooling jacket, the graphitic coating deposited over the brickwork aids greatly in preserving the remaining ring of brickwork, as the graphite is not affected by nonferruginous slag.

The graphite coating is rarely homogeneous. On examination it is found to be composed of flakes, layers, and stratifications, between which are veins of slag and iron. These layers are evidently built up as the furnace alternately scours and incrusts the wall with varying conditions of graphitic or white iron, limy or raw cinder, hot or cold working. Various analyses have been made of this material; three typical analyses are given below:

Typical analyses of graphitic coating on hearth walls.

Constituent.	Per cent.	Per cent.	Per cent.
Graphite..........	25.3	48.4	70.0
Iron...............	70.0	18.3	12.0
Silicon............	1.9	.1	3.0
Titanium.........	1.9	.1	....
Slag...............	....	27.1	....

CRACKING OF THE WALLS.

The protected brickwork may be destroyed unless held solidly in a tight, jointless ring, with no opportunity to develop cleavage cracks through displacement by expansion. Fire brick expands approximately 0.5 per cent in volume on being heated to incandescence and before it vitrifies. Consequently if the hearth walls are not held equally rigid about their circumference, that segment or wedge opposed by the least resistance will move out bodily, forming a crack from the inside to the outside face.

It is considered that loosely held brickwork, because of the possibility of cracks developing in it, gives the greatest danger of breakouts. When such a crack forms, the molten iron penetrates it instantly, and if not checked by the hearth jacket may come through. There may be 6 feet of fluid iron above the tapping hole and 2 feet of slag above the iron. The pressure per square inch of 1 foot of iron is 3 pounds, and 1 foot of slag, 1 pound, so that in this case the pressure is 20 pounds. If to this a blast pressure of 25 pounds be added, a total pressure of 45 pounds per square inch is exerted on the molten iron at the bottom of the hearth.

DEVELOPMENT OF BREAKOUTS THROUGH THE JACKET.

As a rule, breakouts through the hearth jacket occur suddenly. If cleavage cracks form slowly with the iron following them, the water on the hearth jacket or in the circulating coils becomes steamy several hours before the crack reaches the jacket, and, by cooling the jacket vigorously with a large volume of circulating water, the fluid iron may be chilled and the crack stopped. Undue thinning of the walls is similarly shown up and combated. However, if the crack opens suddenly, as is frequently the case, a jet of fluid iron is forced through the joint and penetrates without warning to the hearth jacket. Unless this invading molten mass is instantly solidified by the cooling effect of the hearth jacket it melts a hole in the jacket and the entire fluid contents of the hearth above this point are ejected with considerable velocity. The opening enlarges very fast, sometimes until it is large enough to put a barrel through.

If the liquid iron comes in contact only with dry masonry, sand, or filling, no explosion takes place. If it runs into small amounts of water, violent explosions and heavy concussions result. Frequently at a single breakout a series of sharp detonations occur in rapid sequence, the sound of which is not comparable to that of any explosive, and is never forgotten. When the iron runs onto damp material, similar explosions may take place or steam may be rapidly generated and heave the pavements and floor plates. If there is enough water to chill the iron, there may be no explosion.

The cause and nature of explosions from iron is fully discussed on pages 172 to 174.

THICKNESS OF THE HEARTH WALLS.

The combating of breakouts through the hearth walls above the tapping hole has developed standard forms of construction. Although a few plants persist in building the wall of the hearth very thick, in some cases as much as 5 feet, it is more generally thought that safety does not lie in thick walls but that they rather increase the danger.[a] Many believe that the lining of refractory brick in the hearth walls should not be more than 36 inches thick. After such a wall has been eroded to a thickness where cooling is effective, the contour of the furnace interior as finally established brings the face of the lining far enough from the nose of the tuyère so that the cutting effect of the blast is not severe. Few plants go as far as to reduce the initial thickness of the brick to that observed in heavy walls after thinning is effected. Making the walls only 18 or 22½ inches thick insures efficiency in water cooling from the start, but it is urged that the cutting action of the blast on such thin walls may extend to nearly the same depth as with a 3-foot wall, resulting in an extremely thin sheet of brick between the hearth jacket and hearth contents. This is perhaps hazardous in that with a change in the composition and character of slag and iron, or blast pressure and penetration, a different shape of face and different rate of erosion of the hearth walls is introduced; a more serious result is that a wide joint at the foot of the jacket lining, between the hearth walls and bottom is lacking or may fail partly so that the anchoring effect of a substantial hearth wall over the hearth bottom brick is lacking, which makes it possible that the bottom will float up.

Although average hearth-wall thickness can not be calculated, the thickness generally lies between 31½ and 42 inches. The wall is built of standard hearth and bosh fire brick, 13½-inch and 9-inch keys, and straights being employed. These bricks are laid in a thin grouting of fire clay of the same material and are rubbed together and beaten in place with wooden mallets to make the closest possible joints, all joints being lapped on adjoining courses.

USE OF PACKING BETWEEN HEARTH WALL AND JACKET.

Practice varies in regard to the packing space between the hearth wall and hearth jacket; some plants use packing, others do not. The use of packing is based on the presumption that cracks are bound to develop in the brickwork, and that a ring of some material, independent of the brick, must be placed between the brick and jacket so that molten iron following a crack can not penetrate to the jacket. Also, it is claimed that packing allows for the inevitable adjustment of hearth walls that results from expansion stresses. The two claims are somewhat contradictory in that the packing is supposed to combat the effects of the cracking which it permits.

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[a] Konig, R., Uber Mittel zur Verhütung von Roheisendurchbrüchen bei Hochöfen: Stahl und Eisen, 1918, Jahrg. 38, pp. 485–486.

At some plants no packing is used and the brickwork is laid tightly against the jacket, the jacket being made extra strong with a view to prevent expansion and cracking. This method has been tried at some plants with the result that the hearth jacket cracked, and at these plants the sentiment is largely in favor of the use of packing. With walls more than 3 feet 6 inches thick, the expansion strain is apt to crack even the strongest jacket, as it is sufficient to overcome the tensile strength of even prohibitively heavy jackets. The packing space when used is one-half inch to 4 inches wide and is packed with stamped fire clay, a mixture of fire clay and loam, ganister, ganister and boiled tar, iron borings, granulated slag, or various combinations of the above materials. The packing is stamped in very firmly, but it still possesses some yielding power, for the expansion force of the brickwork exceeds the resistance to compression of the packing material. another method of packing is to lay strips of soft pine between the jacket and brick. A few plants pour neat cement in the packing space, which makes an extremely rigid construction, but one apt to crack when it becomes heated and stresses set up in it. Probably if the packing is put in very tightly and offers sufficient resistance so that the expansion of the brick will close every joint, and is not more than 2 inches thick, it offers the greatest security, both against breakouts and the possibility of cracking the jacket.

TYPES OF JACKETS.

As regards the jacket itself, a wide variety of types are in use. The two most in favor will be described in detail.

THE ROLLED STEEL-PLATE JACKET.

The first consists of a rolled-steel jacket, usually butt-strap riveted both externally and internally. The jacket on a 500-ton furnace is usually 1 to 1½ inches thick and extends approximately 5½ feet above and 4½ feet below the bottom of the hearth, and is frequently reinforced by steel bands 12 inches by 1 inch riveted in at the top and bottom.

Inside of this steel jacket are placed vertically 12 to 15 cast-iron plates 3 to 5 inches thick with 1½-inch pipe cast in for circulation of cooling water. These cooling plates are continuous about the entire circuit of the hearth jacket and usually extend to the bottom of the outside steel jacket. They are not tied together, the strength being furnished by the exterior steel jacket; neither are the edges machined to make a close fit. A packing space is essential on account of the inequality of surface furnished by the rivet heads on the steel jacket and the space between the butt straps. This is filled as described above, the brick in this case being laid solidly against the plates. This type of jacket is usually vertical, that is, cylindrical.

JACKETS WITH COOLING PIPE CAST IN.

The other construction consists of 10 to 16 cast-steel or, more usually, cast-iron segments 5 to 6 inches thick and 8 to 10 feet high, forming a truncated cone perhaps 22 feet in diameter at the top and 25 feet in diameter at the bottom. These segments are held together by six sets of forged-steel links per segment. The links are made of 2-inch square bars and are shrunk onto lugs extending 2⅛ inches from the face of the segment. In addition to these links three to five steel bands 1¼ inches thick by 10 to 12 inches wide are placed about the jacket and tightened by driving steel wedges between the bands and a set of lugs on the faces of the segments. The segments are water-cooled by 1½-inch extra heavy pipe, which is bent and cast in, with feed and discharge ends at the top, and their edges are machined to make a tight joint when the links are shrunk on.

OTHER TYPES OF JACKETS.

The two types mentioned above are most generally in favor, but a great variety of other types are in use. Of these the one probably in most common use is a steel-plate jacket which is generally of truncated-cone shape. It is cooled by water which runs down in a thin film over the surface from a spray at the top of the jacket, the water collecting in a ditch about its base. Somewhat infrequently this steel jacket is protected by pipes, spaced 6 inches center to center, about the inside of the hearth jacket. These are set in clay or iron borings packed between the brick and jacket, and extend down to the bottom of the jacket. They are usually closed at the bottom and water circulation maintained by means of a ¾-inch pipe which is inserted in the 2-inch pipe and the water delivered to it. The overflow is sometimes down the exterior of the jacket, or a number of pipes may be connected in series. At some plants the bottom of two adjacent pipes are welded to make a coil or U tube, and a number of these connected in series, the inside ¾-inch pipe being omitted.

Another variation is to use a cast-steel jacket of 12 segments which are bolted together by flanges and banded and are cooled by exterior sprays or by pipes inside. In connection with these five types of jackets as usually constructed, one may also find combinations of types, such as water-cooled plates behind water-cooled cast-iron jackets, or cooling pipes between steel jackets and water-cooled plates.

COOLING THE JACKET.

It should be borne in mind that the cooling effect of the jacket extends only slightly into the brick on account of the low heat conductivity of the brickwork and does not hinder the molten iron from working back to where the cooling is effected. From this point a sudden crack or erosion of the wall will admit the iron to the jacket. This has happened so frequently that repeated experience has demonstrated that a simple rolled-steel jacket does not suffice to hold the iron in. A ½-inch jacket is often useless in this emergency, and a 1-inch jacket occasionally so. Although the jacket is covered with a film of water that cools it sufficiently under normal conditions, the mass of cold metal is not sufficient to freeze the fluid iron by dispersing its heat, and thus the advantage of the cooling water is not utilized. Practice shows that breakouts with this type of jacket are frequent. The breakout assumes more serious consequence because the jacket is surrounded by a ditch which contains a few inches to two or three feet of water. If only a small amount of this iron comes in contact with a large volume of water there is no explosion, but when the iron continues to escape in increasing quantities explosions usually occur. This danger has been so repeatedly demonstrated that most plants have eliminated the ditch and abandoned the idea of surface cooling.

A few plants still retain surface cooling, but reinforce its effect by cooling pipes behind the jacket. This idea is discounted by many operators because they feel that it has been demonstrated that material for prevention of breakouts must be able to solidify suddenly penetrating iron by quick absorption of a large amount of heat, and they do not think that this requirement is met by circulating water in the interior of refractory material which has a very low heat conductivity. Fire brick of hearth and bosh quality has a heat conductivity of 1.82 (kilogram-calories per square meter per hour). The requirement is better met by a material having a high heat conductivity, and this demand is satisfied by iron or steel with a heat conductivity of 800. Also the plate must be thick enough so that in addition to its high heat conductivity and the quick dissipation of heat to cooling water it offers a large mass of cold material to absorb the heat and chill the fluid iron. This is the case with a 6-inch cast-iron jacket, cooled by water circulating in pipes, or by a 4-inch cast-steel jacket with spray cooling. In the South, the cast-iron jacket with pipe cast in is sometimes designed with a corrugated surface and is cooled with a spray in addition to the circulating water in the pipes. This is probably the most intensive cooling possible. . . . . .

Practically all plants having cast-iron jackets with pipe cast in, or steel jackets with water-cooled staves behind, have eliminated surface cooling and have no ditch whatever. Though some cooling efficiency is lost by discontinuing the spray or water-filled ditch, a gain is effected in that if the iron does come through the joints it comes in contact only with warm, dry brickwork, and there can be no explosion, and in addition there is no open space into which it can run quickly and cause a shutdown by the cutting of a large hole. If it cuts a cooling pipe inside the casting, the possibility of great danger is removed because the resulting explosion has little chance of escape. The water surface exposed inside a 1½-inch pipe is small, and as a rule the fluid iron simply runs into and fills up the coil, where it solidifies.

To eliminate the danger of iron coming through the joints between water-cooled castings, one firm uses two rows behind a rolled-steel jacket, the joints being lapped. In order to eliminate the danger of losing the cooling effect of an entire section by intrusion of iron, the pipes may be bent and cast in in U shape, the feeds and discharges of several being connected in series so that if one is lost the others in the casting may still be kept in commission. To take care of water from temporary discharges of tuyères, plates, and other cooling parts, there is left a shallow ditch 6 inches deep by 12 inches wide, but even this is sometimes eliminated. In some plants a circle of 6-inch pipe is laid about the hearth with funnels at each column, or there may be a circle drain pipe with funnels at columns laid in the brickwork about the jacket.

BREAKOUTS BELOW THE TAPPING HOLE.

DANGER IN USING SPRAY-COOLED, STEEL-PLATE JACKETS.

Where rolled-plate jackets are provided for holding in the hearth brickwork the spray cooling does not extend down to the base of the jacket. Commonly it extends only to the level of the tapping hole or, perhaps, up to 2 feet below the hole, which is approximately the level of the hearth bottom. It is thought dangerous to have the bottom of the ditch about the jacket lower than the bottom of the hearth because of the probability of very severe breakouts, and the jacket is bricked up solidly from this point down, completely around the furnace. The jacket usually extends 2 feet 6 inches to 4 feet 6 inches below the bottom of the hearth and rests on second-quality fire brick, ordinary river brick, or even concrete, the lower segment of hearth being filled with fire brick of hearth and bosh quality.

FORMATION OF "SALAMANDERS."

Under these conditions, it is apparent that this mass of brickwork without water-cooling comprises a highly heated reservoir, which continually attains higher and higher temperatures and affords ideal conditions for melting and penetration of the bottom brickwork by iron. This is found to be the case. Upon blowing a furnace out, the brickwork of the hearth bottom is seen to be replaced by a mass of pig iron, or a mixture of pig iron, slag, and graphite, varying in depth from 4 feet to, in one instance, 26 feet below the tapping hole. Comparatively shallow depths are usually liquid, while deep pockets are usually solid or, unless drained, solidify after the furnace is blown out. These masses of material have long been known as "salamanders."

CAUSES DESTROYING HEARTH BOTTOM AND WALLS.

The penetration of this fluid mass into the foundations of the furnace gives rise to breakouts beneath the tapping hole. The cause is due to a variety of conditions. The erosive action of the blast may be eliminated, and also corrosion by slag, for except prior to the first cast and at infrequent intervals during the first part of the run, the hearth bottom is covered with liquid pig iron, which would prevent any slag coming in contact with the brickwork or gas circulating adjacent to it, except for brief periods immediately after casting. As the depth becomes greater, even this occasional contact is eliminated.

SLAGGING ACTION OF PIG IRON.

There remain therefore the chemical action of pig iron and the mechanical effect due to the weight. Stead[a] has called attention to the fact that manganese in iron attacks fire brick. He found that ferromanganese containing 80 per cent manganese attacked and slagged away portions of a fire-clay crucible.

The action is as follows:

$$4Mn+Al_2O_3.2SiO_2=4MnO+2Si+Al_2O_3$$

The silicon goes into the pig iron, whereas the manganese oxide reacts with additional silica to form manganese silicate, according to the following reaction:

$$4MnO+2Al_2O_3.2SiO_2=4MnSiO_3+2Al_2O_3$$

Both reactions are endothermic and would require abstraction of heat from the fluid manganiferous iron. If it be assumed that such a condition is possible, 1 pound of manganese would require 1.6 pounds of silica, and if the silica content of the fire brick was 50 per cent, 3.2 pounds of fire brick. If 0.01 per cent maganese, of the 0.75 to 2 per cent of manganese, in every ton of iron, thus entered into reaction there would be dissolved by a furnace making 500 tons of pig iron a day, or 150,000 tons a year, 717 cubic feet of brickwork, as follows: 150,000×0.0001=15 tons, or 33,600 pounds of manganese, and this quantity of manganese would dissolve 33,600×3.2=107,520 pounds of fire brick, or $\frac{107,520}{150}=717$ cubic feet of fire brick. Therefore on a 16-foot hearth a layer of brickwork $\frac{717}{201}$ or 3.5 feet in depth would be disintegrated and replaced by pig iron in the course of a year. Such action is conceivable only as long as the pig iron continues to possess sufficient heat to accelerate the reactions noted above. To what depth this action might extend is again problematical, but considering that the iron must be fluid at all places where it effects penetration, and this may be more than 20 feet below the tapping hole, the above action is not unworthy of notice and consideration.

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[a] Stead, A., Blast-furnace bears and what they teach us. Proc. Cleveland Inst. Engineers, vol. 6, 1913–14, p. 169.

BUOYANT EFFECT OF SUPERINCUMBENT IRON.

The other and most accepted explanation of the formation of salamanders is that the individual shapes comprising the hearth bottom are buoyed up by the weight of the supernatent iron, and being insecurely bound in, float up and dissolve in the slag. A cubic foot of brickwork weighs approximately 150 pounds, as compared with a corresponding weight of 450 pounds per cubic foot of pig iron. It is to be noted that it is not at all unusual for a furnace with a 16-foot hearth to make six casts per day of 100 tons each, and this presupposes a height of molten metal in the hearth of 2 to 4 feet, according to the extent of erosion and widening of hearth walls. With two feet of slag on top of the iron and 15 pounds blast pressure in the interior of the furnace the force exerted over the surface of an area 12 inches square would be equivalent to a hydrostatic pressure of 2,800 foot-pounds. Unless every joint between the shapes making up the hearth bottom is absolutely flush and tight along all vertical and horizontal faces, the fluid iron under such high pressure will penetrate the joints.

When one or a group of these shapes has been isolated from the adjacent brick by a skin of fluid iron, the mass must, by reason of its lighter density, float up in the overlying molten iron.

SEPARATION OF GRAPHITE.

A third influence which disrupts the hearth walls and bottom is the separation of graphite from the molten iron. This action has been mentioned as tending to preserve the brickwork, and such is the case after the cooling effect has checked further destruction of the brick. From what is noted in the furnace after it is blown out, the action that takes place is somewhat as follows: When the pig iron accumulates in the hearth and cools off very slowly under the influence of variations in the burden, increased moisture in the blast, and disturbance of equilibrium between the heat furnished and that lost by water-cooling and radiation through retarded rate of driving or local chilling effects, graphite crystals separate out.

When this separation takes place in joints or cracks in the hearth brickwork into which the iron has penetrated, the brickwork is wedged asunder by the volume increase from graphite separation. Consequently more iron intrudes and the process may repeat itself, and though taking place on an exceedingly small scale, the cumulative effect may entirely destroy considerable amounts of brickwork. This intrusion of graphite-coated iron veins into the joints of brickwork, as yet undestroyed, can be noted in both the walls and hearth bottom of furnaces where the bottom is being torn out for renewal. A similar condition is usually found next the hearth walls, where graphite flakes, iron, brick fragments, and occasionally slag, intimately mixed together, may be seen with intrusions of graphite and iron in the joints. This condition, which presents a view of brickwork destruction arrested while in progress, probably results from sudden chilling of the pool of iron, through exterior causes, while disintegration was in full play. Such conditions are usually found in the salamander accumulated under the tapping-hole level.

RATE OF LOWERING OF HEARTH BOTTOM.

The eating away of the hearth bottom is usually slow. Upwards of six weeks may elapse after the blowing-in of the furnace before it becomes evident, from the way the furnace casts iron, from the increasing angle of the tapping hole, and from the volume of iron per cast becoming greater than the designed capacity of the hearth will contain, that the bottom is becoming lower. One furnace blown out after six weeks showed a lowering of the hearth bottom of about 20 inches. Years ago, when the distance between the center line of the tuyères and the hearth bottom was half the distance now general, the position of the tuyères is said to have had considerable effect on the hearth, and it was believed that if the distance was too small there was little means of preventing the danger of breakouts. To what extent this relation was blamed for weaknesses in the hearth jacket and brick, now corrected, is beside the mark. With a distance of 8 feet 6 inches or more between the center line of the tuyères and the hearth bottom in large, modern furnaces dictated by the necessities of tonnage output, no significance whatever is attached to it.

In many cases furnace men report that after the furnace bottom has reached a depth of 4 to 6 feet below the tap hole, little further burrowing of the iron takes place, but that the furnace bottom works up and down, varying with the working of the furnace. When a furnace is working hot in the bottom and fast, the hearth deepens, and when it is working cold or slow, the hearth builds up. The latter condition should, in fact, promote building up, because the equilibrium between the heat supplied to the furnace bottom and the amount given off to the surroundings is changed in favor of the latter. If the hearth bottom is laid snugly and on heating up expands slightly so that it is tied in, this same heat equilibrium should arrest the downward growth of the salamander, both by retarding any slagging action and also by the decreased fluidity due to chilling of the iron at depth.

This balance or alternation between growth or deepening may remain steady for months or during the entire blast. If the brickwork is poorly laid the deepening may progress throughout the entire run. Infrequently the furnace may "lose its iron," which indicates that the iron is deepening at an extremely rapid rate or is escaping through some crevice in the bottom into the soil, sewers, or filling about the furnace foundation. In such cases there is nothing obtained at cast except a heavy run of slag and a quantity of iron short of the tonnage corresponding to the number of charges. Such losses are also usually indicated by the pavements getting very hot, steam coming up about the columns, or gas burning with a blue flame about the brickwork inclosing the hearth. The abnormality may persist for several days, though if it is serious the furnace may be temporarily shut down or driven slowly with a very limy slag to stop the loss of iron.

EXAMPLE OF A FURNACE "LOSING ITS IRON."

A case in point may be of interest. The furnace had been in blast for nearly three years and had been working with a very deep tapping hole, frequently at cast time giving a heavy run of cinder before running the iron. The furnace was at this time making about 440 tons of iron per day, though its capacity with a good lining was about 530 tons. The fact that the furnace was losing iron became apparent on a Tuesday, when the 3 a. m. cast gave only two ladles of iron. The 7 a. m. cast was three ladles. The furnace was taking the customary number of charges and should have averaged five and one-half ladles per cast. At the end of the 7 a. m. cast a 24-foot pricking rod was pushed completely into the tapping hole without finding bottom. Considering the oblique position of the rod, the furnace bottom must have been at least 12 feet below the bottom of the tapping hole. The blast was reduced from 42,000 feet to 37,000 feet per minute, 600 pounds of limestone added to the burden during the day, and 6 holes drilled through the pavement outside the brickwork about the hearth jacket into the filling about the foundation and water turned in. At this furnace the practice was as follows:

Practice in operation of furnace.

Day of week.	Day or night.	Number of charges.	Time of cast.	Iron (tons).
Monday	Night	46	3 a. m.	75.4
			7 a. m.	68.5
			11 a. m.	75.0
Do	Day	44	3 p. m.	70.2
			7 p. m.	68.2
			11 p. m.	51.7
Total		90		409.0
Tuesday	Night	47	3 a. m.	24.6
			7 a. m.	35.1
			11 a. m.	23.4
Do	Day	39	3 p. m.	(a)
			7 p. m.	18.3
			11 p. m.	48.6
Total		86		150.0
Wednesday	Night	43	3 a. m.	70.8
			7 a. m.	61.1
			11 a. m.	67.5
Do	Day	38	3 p. m.	59.7
			7 p. m.	66.2
			11 p. m.	67.8
Total		81		393.1

(a) Cinder.

Thursday morning the blast was increased again to the normal amount, as the furnace from all indications had effected its own repairs and had stopped losing iron. The water about the hearth bottom was left on to cool the surroundings and thoroughly chill the iron vein leading from the hearth. The salamander in the hearth was not removed at relining, nor was the brickwork removed lower than a couple of feet beneath the bottom of the hearth jacket, so the depth to which the salamander had burrowed or the place where the iron made its escape is not known, though it is certain that it took place at least 12 feet below the tapping hole and 8 feet below the bottom of the hearth jacket. About 330 tons of iron was lost, calculated from the charges, during the above period.

DEEP SALAMANDERS A SOURCE OF DANGER.

Whenever a furnace shows indications of such a condition there is danger of an explosion if the iron comes in contact with moisture. Even if the hearth alone becomes unusually deep the danger of a breakout is increased, for as the iron in the hearth becomes lower than the brickwork inclosed by water-cooled jackets it begins to eat outward. The deeper the hearth becomes, the farther downward the hearth walls extend and the greater is the lateral pressure, owing to the increasing head of liquid iron. Once displacement is fairly started by reason of bricks insecurely tied in, uneven shape, thick joints, inadequate anchoring by the hearth walls, or shrinkage cracks, the iron, as the hearth becomes deeper, automatically acquires an increasingly uncontrollable and destructive power and tends to penetrate downward and laterally into the brickwork. The danger which threatens if this deep pool of molten iron suddenly breaks through is evident. The iron, in fact, sometimes works out sideways several feet, and sometimes under and beyond the column base plates, the brick foundation being replaced by a mass of iron. There is always considerable speculation when this state of affairs is revealed in removing a salamander, as it is evident that at some time the weight of the stack under one or more columns must have been carried by a pool of fluid iron. Often lumps of iron weighing many tons are found extruded outside the hearth jacket and below the surface, where they have replaced the fire brick. Just how they melt or displace the brick is also a cause for speculation, especially as the iron, if it does not come in contact with water, shows no sign of intrusion. To date the efforts of furnace men and engineers to prevent the formation of salamanders have failed. Except on special grades of iron and on a few cold-blast furnaces a salamander always forms to a greater or less depth and presents the annoying condition arising from a pool of iron that can not be removed through the tapping hole. So long as salamanders form there will always be uncertainty concerning the depth to which the iron is penetrating, the distance to which it is working sideways, and the possibility of outbreaks beneath the tapping-hole level.

TYPICAL EXAMPLES OF HEARTH BREAKOUTS.

EXAMPLE NO. 1.

At one furnace a serious explosion was caused by the iron breaking out from the hearth foundations. The iron came out about 20 feet below the tapping hole and on the yard level in such great volume that it overflowed the hot-metal tracks and ran along until it came to the manhole of a waste-water sewer. It ran into this, and on coming in contact with the water caused an explosion which threw brickbats, chunks of concrete, and earth over a wide area and fatally injured six men who were standing nearby watching the stream of iron. These men had been ordered away, but in the short time between the breakout and the explosion there was no opportunity to compel them to leave, and their ignoring of the warning was the primary factor in their death. This hearth foundation was not inclosed, as is usually the case, the idea being that if it was exposed to the air sufficient cooling would take place to chill any fluid iron which might penetrate near the exterior of the brickwork, and thus prevent a breakout. The foundations were strongly banded with 15 inch by 1¼ inch bands to prevent displacement.

EXAMPLE NO. 2.

In another instance the furnace had a rolled-steel jacket, three-fourths inch thick, cooled by a spray and ditch in which about 2 feet of water was maintained. The bottom of the ditch was 12 inches beneath the bottom of the tapping hole. The iron came through the jacket about 8 feet from the iron notch, on the side away from the cinder notch and beneath the bottom of the ditch. It ran beneath the bottom of the ditch, not causing any disturbance, and came through the 4 feet of brickwork surrounding the jacket, where it came in contact with wet sand. The explosion that followed threw brick and flame through the cast house and caused the death of the foreman and one cast-house man.

EXAMPLE NO. 3.

In a third instance the furnace jacket was of rolled steel, behind which vertical cooling pipes were inserted at 8-inch centers. Each pipe had an individual feed, the overflow running down the face of the jacket and collecting in a ditch. A 3-inch packing space was filled with a mixture of granulated slag and loam, in which the cooling pipes were set. Without warning, iron came through the jacket into the ditch, which had only about 6 inches of water in it, where it caused an explosion. The force of the explosion stripped the water connection from the bosh and seriously damaged the stack. One man was killed and another seriously burned. The iron lying in the ditch was not removed, but was chilled and left in the bottom, where it covered the hole cut in the jacket by the outbreak. No further trouble was experienced at this place.

About 36 hours later, however, the furnace broke out again on the same side and lost a cast, the iron running out beneath the pavement and into the cinder pit about 100 feet away, where it emerged beneath the surface of the water and chilled, forming a large skull. Aside from violent boiling of the water, considerable flame from gas generated by the contact of iron and water, and throwing about of granulated cinder left in the pit, no damage was caused.

EXAMPLE NO. 4.

The furnace had a cast-iron hearth jacket 4 inches thick, with cooling pipe cast in. An expansion space was provided between the jacket and brickwork in which were placed packing strips of pine. A ditch about 18 inches deep carried discharge water from the columns and also from the "monkey" and intermediate cooler. The iron came out beneath the ditch and worked its way up into the bottom of the ditch, where it exploded. The supposition is that the brickwork suddenly developed a crack, as no preliminary heating of the cooling water was noticed. The explosion is stated to have occurred shortly after the outbreak was discovered. Some men had run up and were in the vicinity when the explosion occurred. One was thrown out of the cast house and severely injured, another received burns of the back and other injuries which proved fatal, and a third minor burns. After the breakout stopped, the brickwork was dug away and a hole found cut through the segment of the jacket between the cooling-pipe coils, the coil being cut by the widening of the hole and rendered useless. The hole was only partly filled with iron as the tapping hole had been opened and the hearth drained immediately after the breakout. The hole was rammed full of crushed fire brick and fire clay, and then the entire face of this segment of the hearth jacket was reinforced by pouring concrete in front of it, forming a block which overlapped a few inches on the adjoining segments. No further trouble was experienced.

EXAMPLE NO. 5.

At another furnace the hearth foundations were not inclosed. The jacket was of cast iron with cooling pipes cast into it and extended 3 feet 6 inches beneath the tapping hole. The iron broke out beneath the jacket and came out about 6 feet above the yard level in large volume, and a huge flame continued to blow out of the opening. Considerable damage was done to the plant. At the next relining the hearth jacket was extended down 2 feet farther and the entire space between the hearth jacket and foundations, and the outside of the cast house about the furnace, filled in with concrete. A space was left about the hearth jacket which was filled in solidly with paving brick set in cement mortar to allow removal of the jacket when necessary on relining and digging out the salamander. No ditch about the jacket was allowed. No further trouble with breakouts has been experienced.

EXAMPLE NO. 6.

Another furnace of the same type as above had uninclosed foundations and a cast-iron jacket with cooling pipes cast in. A breakout occurred in the same manner, but was accompanied by a series of violent explosions as the iron flowed out onto the yard and into a sewer and also onto wet ground. The hearth was completely emptied, and after the iron had chilled, the hole was plugged with clay and fire brick, a wall wedged in on the face, and a water spray put on. This construction held until the next relining, when the opportunity was utilized to dig out the salamander and put in a subjacket beneath the original jacket. This subjacket is water-cooled with water circulating pipe cast in, and brings the depth of water-cooling 2 feet lower. The furnace is now running with the base uninclosed as before, but with no further trouble.

EXAMPLE NO. 7.

The furnace had a rolled-steel jacket 1 inch thick. At one period, cooling was effected by cooled cast-iron plates set vertically behind the steel jacket. The edges were not machined, and there was a packing space of 1 inch filled with a mixture of iron borings and loam. The iron came out between the plates, between the cinder and iron notches, and below the ditch, in which 2 feet of water was maintained. An explosion followed which tore out some of the brickwork and threw about the cover plates over the ditch. Three men were injured and slight damage done. After this experience the furnace was provided with two rows of cooling plates behind the hearth jacket, the edges being lapped. No further trouble has been experienced.

EXAMPLE NO. 8.

Another furnace had a rolled-steel jacket one-half inch thick, with packing space of 3 inches. There was 42 inches of brickwork in the hearth walls, and the jacket was spray cooled, the water being collected in a ditch 30 inches deep, in which about 20 inches of water was maintained. The jacket was 7 feet 6 inches deep, and below the ditch a wall of paving brick was built around the jacket, which extended down to the bottom of the jacket. This wall retained the heat and permitted the jacket to melt away beneath the ditch, an area extending nearly halfway around and from 12 inches to 24 inches high being melted. Eventually the iron broke through, not being retarded by the cooling water in the ditch, and a series of breakouts followed. The iron came up in the ditch and ran along the bottom of the ditch beneath the water until the hearth drained and the iron was chilled. No explosions resulted. This iron was not removed. and finally, in consequence of a number of breakouts, the bottom of the ditch was entirely covered with chilled iron to depths varying from a few inches to about 2 feet. The spraying of the jacket was continued and the water level in the ditch raised from time to time. As this auxiliary iron jacket was built up the breakouts gradually decreased in frequency until finally the furnace ran about three years without a breakout. This experience illustrates the efficiency of a large mass of cold iron to check the melting and failure of fire brick close to the jacket, or to solidify and check, by virtue of its high heat conductivity, any intrusion of iron in contact with it.

EXAMPLE NO. 9.

Another furnace had a cast-steel jacket which was spray cooled, the water being collected in a ditch, the bottom of which was 1 foot above the base of the jacket. The ditch had slope enough to carry off the water quickly and to prevent its becoming more than 5 or 6 inches deep, the outlet being at the rear of the furnace opposite the tapping hole. The brickwork beneath this jacket was also inclosed in a rolled-steel jacket to prevent cracking by expansion. This subjacket was not cooled, but was solidly bricked in with miscellaneous paving brick, second-quality firebrick, and other brick left over from various jobs. The furnace gave warning of a breakout by vigorous steaming about the base in front of the tapping hole for several hours. During the night gas that burned with a blue flame began to issue from the pavement in such quantities that the blower took the blast off and ordered the crew out of the cast house. The volume of gas generated increased, and about midnight a violent eruption took place halfway down the cast house. The floor of the cast house was of cast-iron plates, laid on sand filling held in by concrete retaining walls. The eruption, which continued vigorously for a half hour, hurled the plates and the cast-iron runners about, upset one retaining wall, and bulged the opposite wall. A column of burning gas arose through the cast-house roof. When the eruption quieted down, there was a hole in the cast-house floor about 12 feet deep and over 20 feet across. No explosion took place. The filling about the furnace foundations was saturated with water while repairs were being made, these taking several days. The furnace was started up and run slowly and on a limy burden for two days, and as there was no further evidence of a breakout, it was quickly put up to normal output. About 18 months later the furnace "lost its iron" for several days in succession, but without any damage or alarming signs of breakout other than moderate steaming about the columns and pavement.

EXAMPLE NO. 10.

This furnace had a rolled-steel jacket, one-half inch thick, inclosing very thick hearth walls, so that the hearth jacket was about 15 inches outside the tuyère-breast walls. One-inch holes were drilled in the brickwork between the jacket and tuyère breast, down to the tapping-hole level. A ½-inch pipe was inserted in these holes and led the cooling water to the bottom, whence it rose up in the hole and overflowed down the face of the jacket. These holes were drilled at a slight angle, and at the top were about 4 inches from the hearth jacket. Without warning a severe explosion occurred inside the jacket, cracking it and tearing it open to the top. It is supposed that a crack or joint opened which admitted molten iron to the water-saturated brickwork adjacent to the jacket.

EXAMPLE NO. 11.

A furnace had been in blast for several years, and at the time of the following accident was on its third lining. At neither of the previous relinings had the salamander been removed, it being simply split off about the sides down to the bottom of the hearth jacket, 3 feet below the tapping hole, to permit a new hearth wall to be built in. The furnace began to work sloppy at the tuyères, and conditions became so bad that it could not be dried at the tuyères at the end of the cast. As two tuyères had been cut by iron lying at their noses, the blast was thrown off and the cinder and iron permitted to come back into the blowpipes, which were then taken down. While one gang was cleaning them and other gangs were changing the two burned tuyères and sledging cinder and iron out of the tuyères, an explosion inside the furnace threw coke and flame from the tuyère openings, fatally burning several men. After this all men were withdrawn from the cast house, the approaches roped off, and watchmen stationed. At irregular intervals of one-half hour these interior explosions continued, throwing material from the tuyère openings, but gradually diminishing in frequency and violence. The furnace was allowed to stand 24 hours after the explosions had stopped, the mess was then cleaned up, the furnace gotten in shape, and gradually worked up to regular output. These explosions were not caused by gas nor by water leaking from plates or other cooling equipment, as none of the cooling pipe cast in the health jacket was cut. It is thought that the iron ate its way down into the foundations, forming a large body of molten metal which finally penetrated into damp subsoil. The furnace was built on originally wet soil, covered with refuse and cinder, which was excavated to put in the concrete foundations. A similar explosion occurred in England in 1909 in which the molten iron penetrated the concrete base and came in contact with water.

EXAMPLE NO. 12.

The iron came through a steel-plate, spray-cooled, hearth jacket at the bottom of the ditch and between the cinder notch and tapping hole, and an explosion followed. A shower of molten iron and cinder was thrown down the cast house and about the furnace, burning two men fatally and doing much damage to the stack.

Many descriptions of breakouts could be given, but it is thought that the preceding discussion of causes of failure of the jacket, hearth wall, and bottom brickwork, with the specific illustrations cited, gives a comprehensive review of the factors connected with the process of hearth destruction.

PREVENTION OF BREAKOUTS THROUGH THE HEARTH WALL OR JACKET.

USE OF MATERIALS OTHER THAN FIRE BRICK.

Thus far no material ever used in hearth bottoms has resisted the destructive agencies inside the hearth.

CARBON BRICK.

The presence of graphitic coatings on the hearth walls and in crevices in the salamander, which are seen when the furnace has been blown out, has presented the obvious suggestion of a lining of carbon or coke brick in the bosh, hearth, and bottom. Theoretically, such bricks should be highly satisfactory, because they are infusible at the highest temperatures reached inside the furnace and resist the corroding action of slags, either basic or acid. Also, they are not acted on by molten iron, unless the iron is unsaturated with carbon, in which case the iron would probably satisfy its requirements for carbon at the expense of the carbon brick. In addition, these bricks have a high heat conductivity—7, as compared to 1.3 for hearth-quality fire brick—which seemingly is of advantage, because the cooling effect of the jacket water would penetrate farther into the hearth wall and preserve a thicker wall, while conduction of heat from the bottom brickwork in the hearth would be more rapid and presumably check the lowering of the hearth bottom sooner than is the case with fire brick. Another property of less benefit is that such brick expand much less than fire brick at high temperature.

So far as is known, these bricks are not now, nor have been for many years, used in this country. Many works abroad and three works in America have introduced them, thinking that a panacea for the preservation of the hearth had been found and that the costly work and expensive delay in removing salamanders would be largely eliminated. It was found, however, that the carbon brick were failures when used about the tuyère breast and the tapping hole; that the hearth bottom tended to float up and disappear; and that if even the smallest gas flame issued from the joints between the brick, a hole formed in a few minutes, which steadily grew larger and had to be stopped with fire clay, so that a prejudice against their use developed in this country. They are, however, used abroad, but do not appear to possess any advantage over good fire brick. The results of a symposium prepared in Germany showed that of a total of 10 opinions, 3 were favorable, 5 unfavorable, and 2 noncommittal.[a] The following description of these brick is taken from this article:

> Only coke of best quality, very hard and pure, can be used for this brick. The finished brick must not contain more than 12 per cent of ash. The coke is ground very fine in suitable mills, mixed with 20 to 25 per cent of the best specially prepared, hot steel-mill tar, with which it is thoroughly kneaded, and the mass is then carefully pressed into suitable molds. The brick when formed is enveloped in coke dust to make it air-tight and set into plates provided with a spring and nut and fired at temperatures of 2,375° to 2,550° F. The finished product must be ringing hard and dense, with a specific gravity of 1.2 to 1.35. The size should be as large as possible, to insure better adhesion of the masonry and lessen the number of the joints. They are no longer difficult to make, now being as heavy as 880 to 1,100 pounds apiece (?); heavier than this they are unwieldy and suffer greatly in transportation. The surfaces and edges must be carefully worked over on laying. The sole brick especially, should be put together on a perfectly level cement plate beforehand, in order to avoid outward joints during the work. The mortar consists of a mixture of coke dust containing up to 20 per cent of powdered fat clay, finely ground and mixed into a syrupy mush with pure water. A mixture of tar and finely ground coke dust may also be used as a mortar, which is applied warm. In some mills ordinary chamotte mortar also gives satisfaction. It is essential that the mortar be even and thin, and that the bricks be carefully ground to fit closely, so as to make the joints as small as possible. Contraction and expansion from cooling and heating when the furnace shuts down, especially at first, will cause changes in volume even in this carbon brick, and in insuring the permanence of the brickwork absolutely accurate work is essential. It is therefore a better plan to pay better wages to these workmen than to ordinary masons, and to employ them not by contract but on a per diem wage under steady supervision, and to promise them a bonus if the masonry proves durable after a specified time of operation.

SILICA, MAGNESITE, AND BAUXITE BRICK.

Besides carbon brick, various other materials which would supposedly be more inert than fire brick to the chemical actions taking place, such as silica brick, magnesite brick, and bauxite or alumina brick, have been proposed for hearth brickwork. In a practical way these proposals have not met with any enthusiasm, as furnace men feel that hearth failure is almost entirely due to mechanical causes. All efforts in this country have been in the direction of better quality and uniformity of shape of fire brick, rigid construction, efficiency in water cooling, and elimination of danger elements in advance of impending or possible breakouts.

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[a] Geiger, C., Ueber die Verwendung von Kohlenstoffsteinen im Hochofenbetrieb: Stahl und Eisen, Jahrg. 32, 1912, pp. 1685–1690.

TYPES OF FIRE BRICK USED IN HEARTHS.

The brick used in the hearth and bosh are made of a fire-clay mixture containing about 85 per cent flint clay and 15 per cent plastic clay. The flint clay is ground coarse or fine, as dictated by experience with different types, dissatisfaction with previous linings, or expectation of improvement. Until very recently blast-furnace operators and fire-brick manufacturers did not make much progress in combining knowledge of brickmaking with knowledge of furnace operation, and consequently furnace men were disposed to progress in a circle, constantly wanting a type of fire brick which they had not tried, but which some one else had tried, perhaps with entirely different conditions, and found more satisfactory. Within the past year, by means of tests ingeniously designed to conform to conditions in the furnace, it is becoming known exactly what effect abrasion, heat, and other factors have upon different types of fire-clay brick, and with the cooperation of fire-brick manufacturers progress is being made in the manufacture of brick that will stand up better. The furnace man is interested in results, and is beginning to care little about methods of grinding, weathering, bonding, or burning, as long as the brick does the work required.

Under these circumstances no definite statements can be made in regard to the best type of brick for the hearth. However, only enough plastic clay is used to bind together the more refractory clay. Refractoriness is the prime requisite in hearth and bosh fire brick, and as the more coarsely ground material is as a rule the more refractory, the flint clay, called the grog, is usually ground as coarse as is compatible with strength and resistance to spalling. It is burned at 2,800° to 3,000° F. in order to decrease shrinkage after it is placed in the furnace and subjected to high temperatures, because fire brick undergoes a progressive shrinkage the hotter it is heated, and if it is underburned in the kiln, shrinkage cracks develop later in the blast furnace.

The brick for the bosh and hearth walls consist of 13½-inch and 9-inch straights and keys with a few special shapes for plate and cooler arches. The brick for the hearth occasionally consists of the same shapes, but usually are larger than these. The largest shapes are 18 by 12 by 8 inches; other shapes are 18 by 9 by 4½ inches, 18 by 9 by 6 inches, and 13½ by 9 by 4½ inches. These blocks are machine-made under heavy pressure and must be true in shape and form. They are culled carefully and any soft-burned or warped blocks are rejected. Little or no attention is paid to porosity, conductivity, or density. Only rarely are they analyzed. The following information may be of interest in indicating the composition of the brick:

Composition of four samples of hearth fire brick.

Sample.	SiO_2.	Al_2O_3.	FeO.	CaO.	MgO.	TiO_2.	Alkalies.	Ignition loss.
A	52.80	42.50	1.17	0.50	0.53	2.10	0.53	....
B	56.65	40.51	1.77	.80	.63	....	....	0.2
C	55.01	38.92	3.19	.58	.90	2.21	1.16	.66
D	50.57	42.73	3.87	.25	.34	....	2.17	.0

The average melting point of eight different makes was 3,218° to 3,290° F.

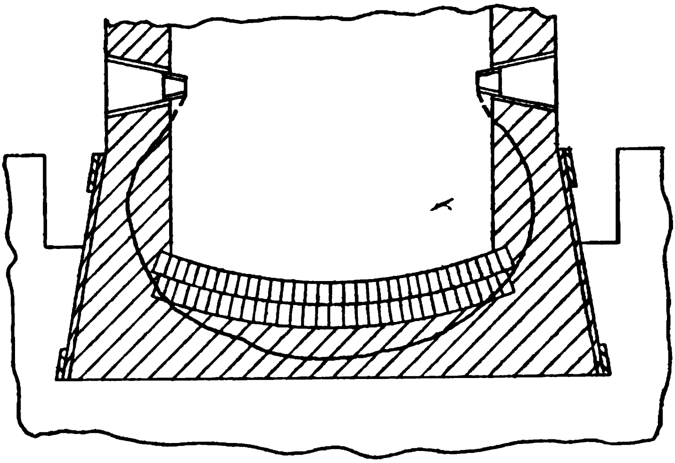

FIGURE 25.—Concave hearth bottom.

USE OF CONCAVE BOTTOMS.

Hearth bottoms were at one time built concave (fig. 25) on the theory that the destruction of the bottom resulted from the displacement of the brick from the pressure and greater density of the molten iron, so that the brick, not being tied in, float up. The layers, usually two, are arranged in concentric circles with a key brick at the center, and the outside circle is anchored down by the hearth walls.

These bottoms did not prove sufficiently superior in permanence to the less expensive shapes and it is not believed that any of these hearth bottoms are now being put in. The weak point was in the periphery, for as the hearth walls were eroded by various agencies the outside circle of blocks began to rise as the overlying brickwork disappeared, and once started the whole course was left without binding and floated up very quickly. The second course soon followed, leaving below the underlying brickwork of small shapes to bear the brunt of the work. In 1914 a few furnaces lined with this type of bottom were in blast, but they were relined with other styles of hearth brick.

USE OF KEYED BOTTOMS.

Figure 12 shows a modification of this style of construction, which is less expensive. Although the keying effect is of course not as strong on a perfectly flat as on a concave arch, it still possesses considerable resistance to the lifting effect of the iron. So far as known, these circles did not extend to the hearth jacket, but were anchored down by the hearth walls, as shown. It is here that their weakness lies; in addition, any defects in workmanship, in manufacture, or in laying, which open a way for the intrusion of iron, would be as fatal to the stability of the bottom as in brickwork not keyed in. Nevertheless, this method of constructing the hearth bottom is followed by many plants, as it is felt that the keying in delays or retards to a profitable extent the disappearance of the hearth bottom.

USE OF CONICAL JACKET.

In both of the above types of hearth bottom, bricks are laid with the vertical joints staggered, as shown in figure 25. This retards the intrusion of iron and promotes the permanence of the bottom. The erosion line shown in figure 25 brings out the significance of the use of a conical hearth jacket. The use of conical jackets was probably introduced to permit thicker brickwork at the juncture of the walls and hearth bottom, in order that the keys could be extended farther out, so that after erosion of the walls had been arrested a substantial ring of brickwork would remain to lap over the outside circle of keys and keep them from rising. A secondary advantage is that the conical shape wedges the whole mass of bottom brickwork in and prevents it from rising. This feature has, however, proved of detriment in some cases. A furnace was seen where the conical jacket had been forced up bodily on one side, away from the iron notch more than an inch, so that it sheared the nipples off the cooling-plate connections before depressions could be cut in the top of the jacket beneath the nipples. The expansion of the brickwork or salamander or the formation of graphite must have exerted sufficient force to overcome the weight of the jacket and its frictional resistance to the pressure on its interior. A possible result of such displacement would be cracking of the brickwork and penetration of iron through these cracks to the jacket. This result, however, did not materialize.

USUAL CONSTRUCTION OF HEARTH BOTTOM.

USE OF CONICAL JACKET.

The method of building the hearth bottom most generally followed is to construct it of large shapes, 18 by 12 by 8 inches, 18 by 9 by 4½ inches, 13½ by 9 by 4½ inches being used. The two most favored styles of laying the blocks are shown in figures 26 and 27; another method not uncommon is shown in figure 28. The blocks are laid completely across the hearth, being in contact with the jacket entirely around the outer circumference of the brickwork, and the hearth wall rests on the blocks instead of 9-inch and 13½-inch brick being laid between the special bottom shapes and the jacket, as in figures 12 and 14.

This construction is much stronger and more rigid than the keyed bottom, because the pressure developed by the expansion from heating after the furnace is in blast exerts its thrust across the hearth diameter, with no opportunity of the individual shapes slipping. The keying effect of the previously described bottoms is, of course, absent, but an equivalent tying-in effect builds up as the brickwork assumes its working heat, on account of the expansion stress against the hearth jacket.

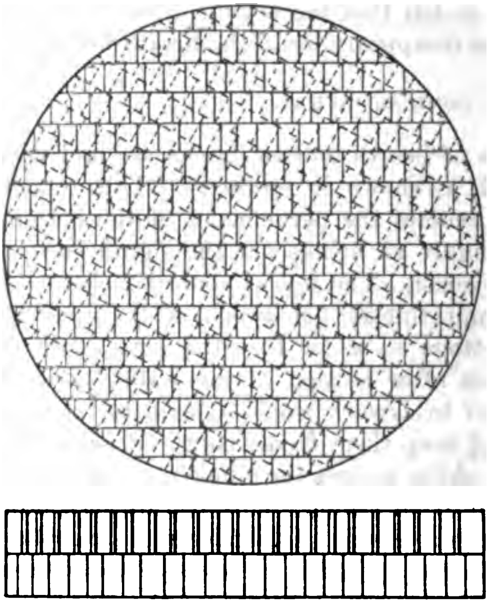

FIGURE 26.—Method of laying blocks in hearth bottoms.

These bottoms have proved as long-lived as the keyed bottoms when it has been possible to lay the brick tightly against the hearth jacket and on a 4-year blast have in a number of instances held the depth of salamander to less than 6 feet. A few plants that lay emphasis on removing the salamander find these types of bottom sufficiently resistant, so that they make a practice of removing the salamander only on alternate relinings.

As a general rule, the smaller size blocks are preferred, as they run truer to shape and permit thinner joints. The brick must be burned sharply to eliminate shrinkage cracks. It is of primary importance that the brick laying be absolutely accurate, with as small joints as possible, and if possible, tight against the jacket in order to allow the expansion completely to close every joint. Each course is preferably laid at an acute angle to the previous course, though it is often laid at right angles. This breaks all joints between the courses and tends to prevent the iron from working down between the courses and joints.

SMALL SHAPES IN HEARTH BOTTOM.

Hearth bottoms are sometimes laid with standard 13½ and 9 inch keys and straights. At the bottom of a 25-foot (inside diameter) hearth jacket, the first course is started by laying a circle of 13½-inch keys and straights next the hearth jacket, and continued by working into the center until 13½ keys complete the circle. The space inside of the circle of 13½ keys is completed by laying 13½ and 9 inch straights in straight courses. The next course on top of this is started by laying one circle of 9-inch keys and straights next to the hearth jacket, followed by circles of 13½-inch straights and keys until 13½ keys complete the circle. Straights are then laid across to complete the course, being laid in opposite direction to the course below. In this way alternate courses are laid until the hearth is completed.

In this manner all the joints are broken, the work of constructing the hearth is made easier, as cutting of fire brick is reduced to a minimum, and the construction sets together well when it is heated and expands. Much less advantageous is the method of laying 13½ and 9-inch straights in straight rows across the hearth, bricks in alternate courses being laid edgewise and flatwise.

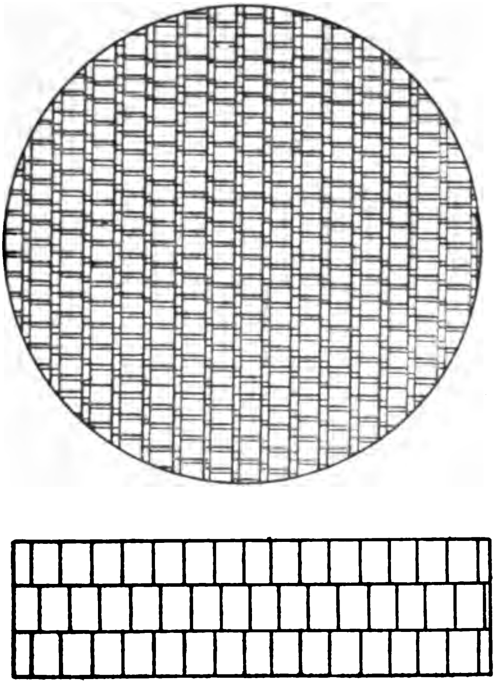

FIGURE 27.—Another method of laying hearth bottoms.

The most commonly advanced reason for the use of smaller shapes, of standard size, is that they are cheaper and truer in form than the larger sizes, having warped less in cooling off in the kiln, so that a minimum of clay is necessary in filling the joints when laying the brick. It is obvious, however, that the number of joints is greatly increased, and in direct proportion, the probability of iron penetrating the joints with inevitable disruption of the hearth bottom is increased.

DEPTH OF JACKET BELOW HEARTH BOTTOM.

The repeated experience of finding a salamander that extends to a considerable depth and has worked out under the jacket, even with the most painstaking care in selecting and laying hearth brickwork, has led furnace managers to extend the hearth jacket downward. In present practice the hearth jacket extends, on an average, 4 feet below the hearth bottom, and most of the jackets recently built extend 4 feet 6 inches and 5 feet below the hearth bottom. The writer knows of one which extends 6 feet, and another 8 feet below the bottom of the hearth; in each case, water-cooled plates set behind the hearth jacket are used to cool the bottom jacket. These figures, however, are extreme, as it is felt by the great majority of furnace men that 4 feet 6 inches provides a large factor of safety, provided that the hearth quality brickwork is carried down beneath the jacket for another 4 feet 6 inches and is bound in rigidly with steel bands or by a steel subjacket.

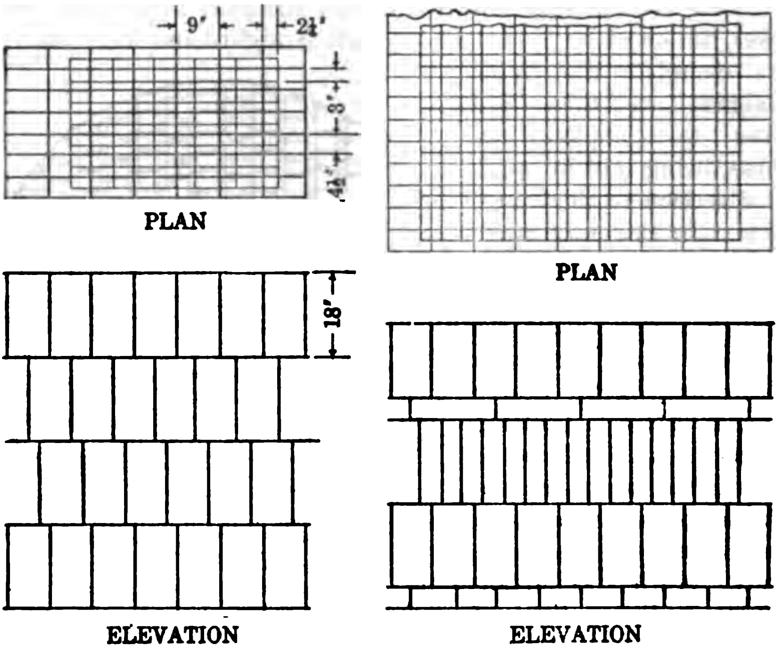

FIGURE 28.—A third method of laying hearth bottoms.

At this depth, 4 feet 6 inches below the original bottom of the hearth and, as an average figure, 12 feet to 14 feet 6 inches beneath the center line of the tuyères, it is only infrequently the case that the heat is sufficient to keep iron fluid so that by either mechanical or chemical action it can effect further disintegration of the brick. This hypothesis can hold only if the brickwork beneath the jacket is so strongly held in that expansion cracks can not form. If the brickwork is thus held and expansion closes the joints, the chance of a breakout beneath the hearth jacket may be disregarded. It is felt that water-cooling, if desired at this depth, is preferably effected by independent means, rather than by prolonging the depth of the interior water-cooled hearth segment, or plate, inside a steel-plate jacket. There are two reasons for this preference. Uneven expansion strains at the different depths below the tapping hole can be more adequately taken up by two superimposed jackets than by one of equivalent length; and stoppage of the water supply by plugging of a cast-in cooling pipe in a subjacket, or of a pipe embedded in the brickwork about the hearth bottom beneath the jacket, is not as liable to be immediately serious as a similar plugging of the pipe at the bottom of a deep hearth-jacket section or plate, as this involves the failure of cooling the hearth walls in a zone where continued and positive cooling is essential.

USE OF SUBJACKETS.

As mentioned above, subjackets are sometimes used to reinforce hearth bottoms. The subjackets used are of four types. One type consists of a steel-plate subjacket, 1½ inches thick and 4 feet high, riveted. These jackets may be set directly beneath the hearth jacket or outside, with the top 2 to 6 inches above the base line of the hearth jacket.

A second type is a cast-steel jacket, flanged and bolted. This is set similarly to the steel-plate jacket, or outside the line of the columns.

The third is a cast-iron jacket, made up of segments, with cooling pipe cast in. This is usually set with the top a little above the base line of the hearth jacket, and is a little larger in diameter than the jacket. Feed and discharge pipes are led down to it along the columns, as shown in figure 29. The disadvantage with these water-cooled subjackets is that the water connections sometimes break off by slight movements of surrounding brickwork or of the jacket, owing to expansion adjustments. Their advantage lies in the binding effect of the strong jacket on the inclosed brickwork, as the segments are held together by shrunk-on links.

In the fourth type the subjacket is replaced by massive steel bands. There may be as many as four bands, 12 to 15 inches wide and up to 2 inches thick. A bottom built from large shapes with a band inclosing each course is very strong.

All the types mentioned have given good results; the cast-steel type is least in favor. With the exception of the water-cooled subjacket, difficulty is found with any of them when used beneath a steel-plate jacket that is spray cooled and when the bottom of the water ditch about the jacket is 2 feet or more above its base line. It has repeatedly happened in such cases that the surrounding brickwork has become so hot that the subjacket is melted or cut and the iron has come through. It is clearly established that positive water-cooling must extend down to the bottom of the hearth jacket if the subjacket is to be of value. So many subjackets have been found cut from top to bottom that several plants cool the subjacket or bands by means of 2-inch extra-heavy pipes, set vertically about the outside of the jacket or band, on 10-inch centers. These pipes are welded or otherwise connected at the bottom to form a gooseneck circuit.

USE OF COOLING PIPES INSIDE THE HEARTH.

Probably the most courageous method of combating the penetration of iron into the bottom and the formation of a salamander is the placing of jamb pipes inside the hearth brickwork and extending from the top of the hearth jacket, between the hearth and tuyère jacket, diagonally downward to beneath the bottom of the hearth. Until this construction has been thoroughly tried out few furnace men will admit that any good can come of it.

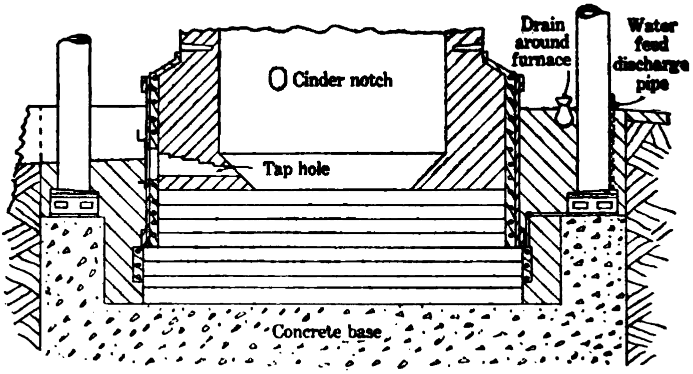

FIGURE 29.—Hearth bottom having cast-iron jacket with cooling pipes cast in.

LEAVING IN THE SALAMANDER ON RELINING.

On the basis of inquiries made, it seems that about 35 per cent of the furnaces in this country make the salamander the hearth bottom. The men in charge of these furnaces have come to the conclusion that regardless of the kind of brick or construction employed the bottom will disappear to a certain depth, and that if the iron should by any chance go any deeper no harm is done. The view has also been advanced that the iron eats to a certain depth at the first blast, then solidifies into a permanent block, which undergoes no change during the remaining life of the furnace. This solid mass of cast iron, it is claimed, is more effective than the best brickwork in preventing breakouts, because it is without cracks or joints and, from its high heat conductivity, will dissipate heat quickly and freeze any iron that may be lowering the bottom. In support of this view instances have been cited where, on blowing the furnace out and tearing out the brickwork between the jacket and salamander, a distinct and separate layer or skull of iron could be seen on top of the old salamander from a previous blast. This skull represented the new salamander, and goes to support the theory that once the lower part of the salamander formed on the first blast is solidified in any way the hearth temperature never after becomes high enough to remelt it.

Against this view may be cited the breakout described under example No. 10 (pp. 55, 56), which indicates unmistakably that the salamander had remelted and continued to work down, and also the breakout described under example No. 11, in which case the salamander had not been removed. Other objections are that on blowing in with an iron bottom, the bottom tends to work high for some time, inducing cold, high-sulphur iron, and causing difficulties with the tapping hole, that is, hard holes which may lead to hazardous conditions. However, if enough of the salamander is removed so that about 18 inches of brickwork can be laid on top of it, this danger disappears. Another disadvantage of having the salamander in is that the furnace is prone to work on a deep bottom in the course of a few months.

More serious is the trouble caused by the expansion of the salamander. When the salamander is left in, the hearth jacket is usually re-covered, the brick dug away, and enough of the salamander wedged or shot away to permit laying 30 to 40 inches of brickwork between the jacket and salamander on relining. If the salamander extends outward when uncovered in removing the brickwork, it is necessary to work to the bottom of the hearth jacket in splitting off the side, in order to put in an underpinning of brick beneath the base of the jacket and obtain the desired thickness of new wall. In some instances the brick foundation for the jacket, perhaps 9 or 13½ inches thick, is laid directly on top of the chilled salamander, which had worked to an unknown distance laterally and downward, as shown in figure 30. If no expansion space is left between the jacket and brick, or brick and salamander, it may easily happen, and has happened, that the salamander heats and expands after the furnace is blown in, so that the hearth jacket is cracked and crevices form in the hearth brickwork, which may result in a breakout. Column base plates have been cracked and columns displaced so that they get out of plumb. Such methods of rebuilding the hearth can not be said to be approved of by the majority of furnace men. Of statements made, about 70 per cent recommend removing the salamander at each or alternate relinings.

PROTECTION OF FURNACE COLUMNS.

The columns about the furnace are usually incased in fire brick or concrete from the base plate up to the cast-house pavement (see fig. 29). Often this reinforcement in the vicinity of the tapping hole and cinder notch is extended about 2 feet above the pavement. On most of the recently built furnaces the base plates of the columns rest on a concrete column or ring built about the hearth jacket and having its top 2 to 3 feet above the bottom of the jacket. With this construction any iron escaping through or beneath the jacket will be less likely to come in contact with and cut the column, thus undermining and weakening the foundations of the furnace. In the older furnace plants the columns stand on the concrete or brick foundation on which the hearth brick rests; their base may be at the same level or lower than the base of the hearth jacket. Several instances have occurred where molten iron penetrated the columns, which are usually hollow with 3-inch walls, and rose up inside the columns. The brick and concrete foundations inclosing the columns and base plates are usually held in against expansion of the brickwork and of the hearth by very heavy steel bands.

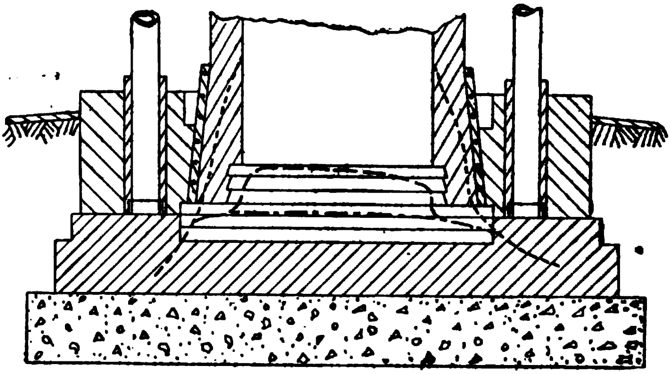

FIGURE 30.—Section showing method of repairing hearth bottom.

COOLING THE WALL ABOUT THE TAPPING HOLE.

With furnaces having steel-plate hearth jackets cooled by sprays, the weakest point is usually just above and below the tapping hole. A few have Z-bars riveted on over the iron notch which divert the spray water to one side. More often this part is not cooled, reliance being placed in a cooling plate about 3 feet above the tapping hole, set back 12 inches from the inside face of the lining, and the use of considerable amounts of clay in stopping the hole. This clay flattens against the wall and tends to build up and constantly renew the brick eroded by the blast. Frequently, beneath the tapping hole a brick pier is built against the jacket on which the skimmer trough is placed. This induces melting of the jacket and breakouts at this point. To take care of this difficulty in jackets relying on spray cooling, a small auxiliary trough is riveted or cast on; the skimmer trough is set in this and packed with loam to make a tight fit. The ditch is extended completely about the jacket and a spray introduced beneath the trough. A very serious accident was caused by the lack of such a trough; the iron at cast ate down between the jacket and skimmer trough, dropped into the water in the ditch, and caused an explosion.

IMPORTANCE OF FURNACE DESIGN AND CONSTRUCTION.

It may be thought that too much attention has been devoted in previous pages to details of construction. In contrast with nearly every other class of furnace accidents, breakouts are primarily dependent upon construction. If the bosh, tuyère breast, hearth walls and jacket, or hearth bottom lack strength or rigidity, or are insufficiently cooled, changes in practice or methods of work to avert breakouts are of little avail, and inasmuch as these accidents usually occur with appalling suddenness and frequently without warning, proper design of this part of the furnace is practically the only means of prevention.

NEED OF ATTENTION TO COOLING-WATER SUPPLY.

Several precautions and emergency measures may be taken to minimize the danger incident to breakouts. The bosh-cooling plates and also the plates about the tuyère breast should be flushed out regularly once a week with water under high pressure to prevent excessive accumulation of sediment. If this precaution is neglected failure of the plates is much more likely. In the lower part of the furnace, where the plates may come in contact with molten iron in the hearth, an explosion may result if the iron touches a plate insufficiently cooled because of water circulation being hindered by deposited mud.

The feed water for the cooling equipment should go through a double basket strainer with holes not larger than one-quarter inch. These baskets should be changed and cleaned at the beginning of each shift or oftener, and in times of flood water when leaves, chips, and other débris may be drawn in at the pumps, the strainer should be inspected more frequently. In addition to provision for high-pressure water, there should be a steam line into the cast house to blow obstructions out of cooling pipes cast into the hearth-jacket segments or plates. Steam at 130 to 150 pounds pressure will frequently open a plugged or slowly dribbling pipe that does not respond to high-water pressure.

The discharge pipes from hearth-cooling equipment should be carefully watched for signs of slackening flow or steamy water. As soon as such are seen the high-pressure water should be promptly turned on and kept on for several hours, or until the service water does not become unusually hot when turned on. An abnormally high temperature of the water is not necessarily, but is likely to be, an indication that the wall is becoming thin near that part of the jacket or that a crack is slowly opening. A larger volume of water circulated will help to stop the intrusion of molten iron by cooling the wall in that vicinity much faster. With spray-cooled plate or cast jackets, an additional short spray over any part that is becoming steamy is for the same reason an effective safeguard against failure of the jacket at that point.

CARE OF THE HEARTH DITCH.

If there is a ditch about the jacket, it should be kept clean. All débris, such as coke, slag, and dirt, should be cleaned out regularly and not permitted to accumulate. Before the furnace is started up, the bottom and sides of the ditch should be grouted with a neat Portland cement mortar so as to be absolutely tight and prevent water from seeping down about the hearth foundations. Whenever the furnace is shut down one or two days for repairs, as is the case once or twice a year, the spray should be shut off, the ditch cleaned out, and another coat of cement applied to close up any cracks developed by the expansion of the hearth and other brickwork.

The outlet of the ditch should not be raised to raise the level of the water. Cooling is more effectively done by a spray giving a rapid flow of water down the entire surface to the bottom than by a body of water flowing with relative slowness about the ditch. Another reason for not keeping the water in the ditch at a high level is that if the surface is higher than the bottom of the skimmer trough the water may, as has happened, work its way between the bottom of the trough and the brickwork of the pier upon which the trough rests, seep up between two sections into the trough, and cause an explosion. The need of such precaution is obvious when the ditch extends completely around the jacket, but is not so self-evident when the trough rests upon brickwork in front of the hearth jacket below the tapping hole. When that section of the spray-cooled jackets above the tapping hole becomes hot and water cooling is not feasible, it can be effectively cooled by directing a blast of cold air on its surface from one or more "rose-head" nozzles.

PRECAUTIONS AT BLOWING IN.

REGULATION OF COOLING WATER.

In blowing in it is customary to delay turning the cooling water on or into the hearth jacket until the first flush of cinder has appeared, in order to avoid any possible chilling of the hearth. After the first or second flush the water is started slowly and brought up to full head in the course of 48 hours, depending on the rate of blowing in and of obtaining the first cast. It is imperative that the water system be given a thorough try out before blowing in, for in event of any defect appearing immediately after the furnace is started there is good cause for apprehension, as water-cooling must be effective from the start in order to establish the working thickness of the walls.

Cooling-water discharges from the columns, if the columns require cooling, should be in plain sight of the furnace crew, and not led into the ditch beneath the ditch plates. Cooling should not be employed until the columns become quite warm to the touch, and even then they are probably as safe without as with interior cooling, for if iron should penetrate into a column filled with water, in all probability the resulting explosion will wreck the column. Water-cooling of columns was originally employed at furnaces where the column base was beneath the bottom of the hearth jacket and was useful in cooling the surrounding brickwork.

TYPES OF SLAGS AT BLOWING IN.

In blowing in, it is customary so to burden the furnace that the slag will be extremely limy, as a limy slag, particularly with the extremely hot iron made during the first 8 or 10 casts, is conducive to a heavy separation of graphite which may, to some extent, protect the hearth walls and bottom. Typical analyses of blowing-in slags are given herewith:

Results of analyses of limy slags from three plants.

Constituent.	Content in slag from—					
	Plant A.[a]		Plant B.[a]		Plant C.[a]	
	Theoretical.	Actual.	Theoretical.	Actual.	Theoretical.	Actual.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
SiO_2	30.53	34.34	24.35	29.10	28.98	30.23
Al_2O_3	11.76	16.42	19.39	19.88	16.58	16.04
CaO	40.91	....	50.11	46.40	47.58	47.33
MgO	12.23	....	4.15	2.56	2.53	2.07
Mn	1.54	....	.39	.15	.33	.81
S	3.03	3.14	1.55	1.71	2.19	1.45
Fe	....	.55	....	Trace.	....	.21

[a] Analyses by plant chemist.

The slag obtained at plant B gave trouble for several days, as in spite of high-silicon iron, high sulphur persisted and was not lowered until 500 pounds of gravel was added to the burden to decrease the alumina content. The theoretical analyses from plants B and C show needlessly limy or aluminous slag. The slag at plant A, although limy, was much more fluid on account of the high magnesia content. The slag at plant C was so limy that it slaked in the runners.

As is evident, liming the charge at blowing-in can be overdone very easily, and it is becoming the tendency to make the blowing slag as nearly normal as practicable with regard to the limitations of the constituents of the blowing-in burden, as the high silicon content of the iron causes graphite to separate regardless of the character of the slag. Examples of blowing-in slags from two plants, as analyzed by the plant chemist, which gave richly graphitic hot iron without being abnormally basic, are:

Results of analyses of slag from two plants.

Constituent.		Slag from plant D.	Slag from plant E.
SiO_2	per cent	32.80	35.39
Al_2O_3	do	17.20	15.03
CaO	do	45.30	44.05
MgO	do	1.36	3.28
Mn	do	.41	----
S	do	2.44	1.68
Fe	do	.22	----

All these slags were described as hot and gray. A few furnace managers fill the hearth below the tapping hole with clean, large lumps of converter cinder. This cinder carries from 8 to 15 per cent of manganese oxide, which is supposed to have a reducing action and impart a high manganese content to the pig iron collecting in the hearth previous to the first cast, and thus promote the separation of graphite, an action noted in furnaces producing ferromanganese. Inasmuch as, to be on the safe side, a furnace has to be blown in hot anyway, it is likely that neither an excessively limy burden nor one that will give high manganese iron is at all essential, and that graphite separation will take place in any case, whether helpful or not, at this stage of the run.

REGULARITY IN FLUSHING AND CASTING ESSENTIAL.

Regularity in the time of flushing and of casting is imperative. The higher the iron rises in the hearth, the greater the internal pressure and the danger of iron penetrating the walls and causing a breakout just before casting. While the interval between casts is governed somewhat by the rate of driving and the capacity of the hearth, there is nevertheless a considerable discrepancy between the tonnage per cast in furnaces of the same rated hearth capacity. This difference does not necessarily indicate that the furnace making the larger production is in a more hazardous condition, or that with the one making the smaller production the cast, in case of emergency, can be delayed until the hearth contains an amount of iron equivalent to the tonnage per cast of the faster driven furnace. This is because the tapping hole on the furnace making the larger tonnage is deeper and the hearth is deeper, so that the space between the bottom of the tapping hole and the center line of the cinder notch after cast—this distance representing the maximum capacity of the hearth for iron—is much greater than on a slower rate of driving. Figure 31 shows this condition. It is therefore dangerous to hold back the cast beyond the regular time, and there should be firm insistence that holding of casts be avoided.

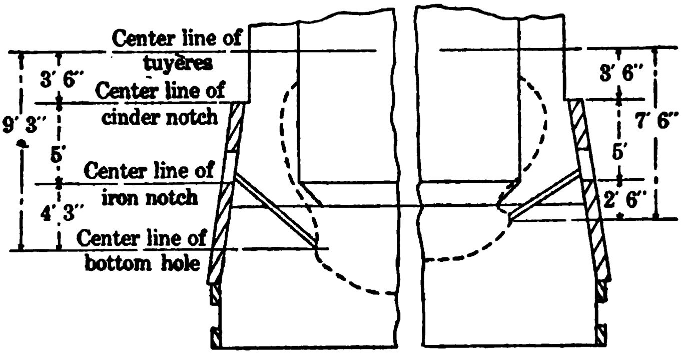
Center line of tuyères
3' 6"
Center line of cinder notch
9' 3"
5'
Center line of iron notch
4' 3"
Center line of bottom hole
3' 6"
7' 6"
5'
2' 6"

FIGURE 31.—Diagram showing hearth conditions of fast-driving furnace.

PRACTICE WHEN CASTING IS DELAYED.

METHODS OF OPENING HARD HOLES.

To avoid the probability of having to hold casts on account of hard or ironed-up holes, every plant should keep in stock at least a half dozen cylinders of compressed oxygen. By directing a jet of oxygen against the hot solid skull in the hole, or if the skull be cold against a handful of glowing kindling and aluminum powder placed against it, the hole can be opened very quickly.

Another method is to use an electric burner, consisting of a 3-inch carbon rod connected to the negative pole of a generator, the positive pole being applied to the hearth jacket. Such a burner takes care of mixtures of slag and iron that the oxygen jet may not eat through. The invention of these two appliances undoubtedly has prevented many breakouts, and they should be a part of the equipment at every furnace. The only objection ever urged against them is a singular one; it is stated that if permanent connections for an electric burner and a readily obtained resistance are provided, or if oxygen tanks are kept in readiness, the furnace crew becomes careless in the care of the tapping hole, and hard holes are more frequent, and that the men rapidly lose skill in sledging and become less resourceful. If these appliances are not used with some discretion by the foreman, such a state of affairs may exist. However, it has not been observed by the writer, and the objection is put forward only at plants where the management is inclined to be conservative.

Although a new tapping hole can usually be made, at the expense of some delay, by driving and drilling above the old tapping hole, or by driving a bar through the skull with sledges, rams, or dollies, it sometimes happens that the bar breaks off in the back of the hole when an attempt is made to withdraw it with the "Welshman," or freezes in the hole and can not be withdrawn, or the molten iron freezes in the hole, or the hole dribbles iron very slowly and the pricking rod can not be pushed in; then the fluid iron may cool, filling the tapping hole and trough with a solid mass of iron. Any of these mishaps leaves matters worse than before. Meanwhile the iron is rising higher and higher in the hearth, and if it rises above the line of safety it may run out, cut the "monkey" or cinder notch, and cause an explosion, or break through at some weak place in the hearth.

These misadventures and possible accidents can be avoided only by providing oxygen and electric burners. In the event of a "mucky" hole, one in which the iron is so pasty that it will not run freely but moves slowly up in the hole as it is burned out, neither type of burner is of much use; but this is about the only frequent serious condition that the burner can not overcome.

PRECAUTIONS TO PREVENT THE IRON RISING TOO HIGH.

In the event of an obstinate hole, or breakdown at the mixer or pig machine, delay in emptying the beds, or delay in spotting the ladles, care must be taken that the iron does not rise too high in the hearth. At some plants where the hearth walls are known to be weak, or the hearth bottom very low, the blast is taken off just as soon as any delay develops without promise of immediate correction. Other plants keep on full wind for not more than 40 minutes after the regular casting time. If the delay still persists the blast is reduced from a pressure of 15 to 9 pounds and thus to a pressure of 4 to 5 pounds. This low pressure is kept on not more than 20 to 30 minutes, as it may make the furnace stick; then the blast is taken off altogether and the furnace drafted back.

The furnace should be flushed more or less continually during such delays to keep the cinder away from the tuyères. Care should be taken not to let the iron come over the cinder notch, as it may cause an explosion, and especially not to keep the wind on until the iron is above the cinder notch and cinder at the tuyères, when the cinder can not be flushed without danger of cutting the "monkey" and cinder coolers nor the wind be taken off without the blowpipes filling.

ATTENTION TO SIGNS OF A BREAKOUT.

The cast-house crew should be instructed to pay particular attention to any indications or signs of breakouts. Although breakouts through the hearth jacket above the tapping-hole level are frequently sudden and without warning, nevertheless indications are not always lacking; the jacket may get unusually hot and steam heavily, or the water discharge from the cooling pipes set behind the jacket, or in plates or jacket segments, may start to steam and boil.

Impending breakouts beneath the tapping-hole level are indicated as follows: The pavement and brickwork about the jacket may become abnormally hot; the base itself, if not inclosed, may get unduly hot; the columns, if uncooled, may become too hot to touch, or if cooled, the discharge water becomes very steamy; steam or gas, the latter igniting and burning with a bluish flame, may issue from the cracks and joints in the pavement and in the brickwork about the columns and jacket.

PROCEDURE WHEN A FURNACE IS "LOSING" OR "HOLDING" ITS IRON.

"Losing the iron," and flames about the base, are unmistakable signs of lowering of the bottom, undue penetration of iron to moisture-impregnated brickwork, or a breakout. The significance of a furnace "losing the iron" has been described, but it should be mentioned that this condition is not to be confused with a furnace "holding the iron," the occasional failure of a furnace taking charges regularly to give anything like the normal amount of iron at cast. When a furnace loses its iron, there is usually a heavy run of cinder from the tap hole and a small cast or none, whereas if a furnace is holding up its iron, the tapping hole usually runs iron very slowly, with considerable blowing or blubbering, and the amount of cinder is somewhat scanty.

The best way of making sure that a furnace is holding iron and not losing it is that used at a large plant in the Pittsburgh district. In the event of an abnormally small cast, even with indications of holding up, the cast time is set forward 2 hours, and the next cast, instead of following 5½ hours later, as is the regular practice, is made 3 hours later. It is rarely found that the previous small cast indicates that the furnace is losing iron, the contrary is more frequently the case. Setting forward the casting time, besides developing the true state of affairs within the hearth, also obviates the possibility of an excessively high accumulation of iron being held in the hearth. The practice is one worth adopting at plants where it is not followed.

When a furnace is known to be losing its iron, or when gas generates and burns about the base, a breakout is indicated. There may simply be a lowering of the bottom, the iron may be seeping slowly into moist surroundings, or a very dangerous breakout may have started. Although the escaping iron may not cause damage or an explosion, the most careful plan is to flush the cinder, check the furnace, and cast immediately. The furnace should then be allowed to stand with the tuyères plugged for 24 or 48 hours, when more moderate driving and a more limy slag will enable the furnace to chill off in the bottom and effect its own repairs. It is also logical, if the brickwork about the base is inclosed with a filling of sand, brickbats, or other rubbish, to drill a number of holes down about the outside and insert long pipes attached to the water-supply mains to chill off the bottom. The water should be maintained under full head on or in the hearth jacket, subjacket, and cooling pipes. The plan more generally followed is simply to reduce the blast and make a limier slag. When this plan is followed, many advise against saturating the surroundings of the hearth brickwork with water, because of the danger of a destructive explosion if the iron eventually gets into the wet material. Whenever derangement of the hearth bottom is indicated and measures are being taken to overcome it, there is double need, until the indications mentioned have disappeared, of the blower and cast-house crew being on the alert for increasing emission of gas about the pavements or other signs of the breakout developing faster. Experience and knowledge of the construction and behavior of the particular furnace are the only safe guides when these conditions develop, but it may be said that it is better to err on the safe side when the signs become pronounced and to cast and take the blast off at once, or even, if the signs become especially alarming, to take the blast off without casting and simply flush the cinder. The crew should be dismissed from the cast house, all approaches to the cast house guarded, and the furnace left standing until either a breakout actually occurs or the signs quiet down. If in about four hours no breakout has occurred, the crew can with comparative safety return and take such measures to bank down or resume work as the superintendent decides.

PROCEDURE WHEN STEAM ISSUES FROM ABOUT THE FURNACE BASE.

Steam about the columns and other places in the floor adjacent to the hearth bottom may indicate, in case of wet filling about the hearth brickwork, that the base is becoming hotter, which is not without significance, or on the contrary may indicate only that water is penetrating into the fill, the hearth bottom surroundings being only normally hot. In either case, the appearance of steam indicates that a condition exists that may possibly cause an explosion should the hearth bottom be lowering, widening, or cracking, and the molten iron penetrate the wet material about the base. It should lead to prompt grouting of all ditches, both about the jacket and in the pavement about the cast-house wall, and such repairs to pavements as will eliminate pools of water and allow the water to run off quickly. Except at periods when, in the judgment of the superintendent, saturation of the already more or less wet fill about the hearth foundations with water is essential to prevent a breakout, these should be kept as nearly dry as possible. A breakout into dry sand, brickbats, or other material is harmless so far as possibilities of an explosion are concerned.

REPAIRING THE JACKET AND WALL AFTER A BREAKOUT.

In case a breakout above the tapping-hole level has cut the hearth jacket, the most effectual means of protecting the jacket and walls at this point against further failure is stated to be the chilling of the iron in the hole. If necessary the iron is cooled with an auxiliary spray during the remainder of the run, and its great relative heat conductivity makes it the most resistant part of the jacket. Chilling of the iron is frequently unfeasible. In this event, the hole is usually tightly rammed with clay and brickbats. Another method that has also been recommended is to drive an iron plug into the hole, if the latter is not too large, and wedge it in tightly. This serves in the same way as a chilled lump of iron left in from the breakout.

When the breakout is at the tapping-hole level or beneath it and comes into the ditch, the usual practice is to leave the solidified lump in, clean any cinder off the top of the lump, and raise the level of the water to cover the top. Such treatment of the hole cut is almost always efficient and prevents a repetition of the accident at that point.

When the ditch is provided with a lead plate, through which the iron melts and escapes before chilling against the tapping hole, or when the iron runs along the ditch beneath the water and escapes into the sewer, or when for any reason the iron does not chill in or against the hole, the latter is usually rammed full of crushed fire brick or clay, which is sometimes wet with tar. Large quantities are used and are packed by 6 to 12 men swinging a broad-faced "dollie" against the face as the material is thrown in. Sometimes hours are consumed at this task, but it is time well spent, as the emergency refractory wall must be tightly packed into the irregularities of the cut-out hole. The surface is, if necessary, faced with a thin brick wall and sprayed with water. Breakouts thus repaired, if not unusually and excessively large, prove as strong as the original wall. In one instance, where the hole was almost 2 feet in diameter and was cut near the cinder notch at the bottom of the ditch, a solid wall of concrete was built against the clay packing.

If cast-iron or steel jackets or vertical plates with cooling pipe cast in are used and there is no ditch, the furnace is usually allowed to make its own repairs, iron that has penetrated the jacket or between the segments being left. Any cooling pipe cut by the iron is disconnected, and the cooling-water feed pipes led across to the still available connections. A hole is sometimes drilled downward along the outside face of the jacket in the brickwork, and a half-inch pipe inserted for temporary water cooling.

When a destructive explosion within the cooling pipe blows out the jacket and surrounding brickwork, the hole made by the explosion is usually cleaned if possible, rammed full of refractory material, and the brickwork replaced about the jacket. No instance is known of an attempt to replace a jacket segment of a furnace in blast; in fact, this is unnecessary and practically impossible. Occasionally a space is cleared in front of the cut or blown-out section of the hearth jacket and concrete is poured in. When a breakout is in the depths of the furnace foundations, unless the base is uninclosed, and the hole is rammed with clay or other material, usually no effort is made to find the precise place and repair it, but after the breakout the furnace is allowed to stand with the wind off for 24 to 36 hours and its surroundings are thoroughly saturated with water to chill the runaway iron. A few plants have taken the opportunity, on relining, to dig out the sand or other filling surrounding the hearth foundations, and fill the entire space around and in front of the furnace with a monolithic mass of concrete. This, with elimination of the water-filled ditch about the hearth jacket, promises the greatest immunity from disastrous hearth breakouts.

TAPPING-HOLE BREAKOUTS.

FACTORS TENDING TO DESTROY THE IRON NOTCH.

With hearth jackets and hearth brickwork strongly constructed and copiously cooled, the weakest point is immediately over and at the tapping hole. The iron notch is expected to withstand the erosion of fluid iron forced out at high velocity, the slagging action of cinder, which begins to come out with the iron during the latter half of the cast, and the cutting of the gas or blast sweeping down from the tuyères when the iron sinks to a low level in the hearth near the end of the cast. To withstand this action there is provided simply a plug of fire clay, which fills the opening left in the hearth walls at lining, 6 inches by 24 inches by 36 inches in size.

Generally speaking, the limier and especially the richer in magnesia the slag is, the more severely is the clay in the tapping hole cut out and weakened. When the lime content becomes excessively high so that the slag is extremely mushy, the action is not pronounced, but the action of limy slags sufficiently fluid to work well is notably severe.

METHOD OF STOPPING THE HOLE AT BLOWING IN.

When the furnace is blown in the iron notch is rammed full of clay mixed with up to 33 per cent of coal or coke dust, the men working from the inside and outside of the furnace. The notch is sometimes left closed, but usually a 4-inch pipe is inserted in it when it is being clayed up. The pipe should project at least 6 inches inside of the brick lining, and clay be well packed about that part, so that the end will be well inside of any skull that chills on the walls. Holes are drilled in the outer end of the pipe, so that it may readily be withdrawn. Sometimes no pipe is put in, but the clay is packed about a tapered form 4 inches in diameter at one end, 6 inches at the other, and 6 feet long, which is withdrawn after the furnace is filled and ready to light. This is done to insure the hearth bottom being heated by the gas sweeping down through and out of the tapping hole, and also because on blowing in great difficulty is usually experienced in opening the tapping hole, if it is closed from the start, on account of the iron chilling against the cold walls, as frequently the first iron coming in contact with the hole seems to be cold and insufficiently carburized, so that it is very tough.

The bottom of the hole, which is put in horizontal or at a very slight angle, is usually at least 18 inches from the hearth bottom on the inside, so that the first slag collecting on top of the iron in the hearth either closes this opening in the iron notch or else, when it shows at the iron notch after 8 to 16 hours, the clay gun is swung around and one barrel of clay shot in. In either event, there should be only a skull of cinder to drill through when the first cast is made. An efficient method of keeping the hole shut when blowing in is by filling it, as shown in figure 32, page 81.

CARE OF TAPPING HOLE WHEN FURNACE IS IN BLAST.

The erosion of the tapping hole begins at the first cast. Usually the top and sides are worn away slightly by the iron, slag, and gas. When the hole is stopped with the clay gun at the end of the cast, more than enough clay to fill the hole is forced in, and the surplus tends to spread along the inside of the wall, on the top, sides, and bottom. The clay thus distributed, if it is of correct plasticity, tends to replace completely the burned-away wall.

With increasing production the bottom of the hole is made deeper and deeper as the hole is drilled, the angle of slope being increased until the desired capacity is reached. With higher blast and production the cutting effect is increased, until eventually as much as three handbarrow loads of clay are used to stop the hole. One barrowful, with the clay in the barrel of the gun, is more than enough to fill the hole, the two extra barrowfuls being used to insure that the walls and a substantial roof are built up. At this stage such excess clay, besides replacing the walls, tends to flatten out on the inside of the hearth wall and to thus form a key which locks the stopping clay in place within the hole and prevents it from being blown out by the internal pressure. The hole in a 36-inch hearth wall is frequently 5 feet long, but because of its slope does not project in as far as the length might indicate. It is desirable to keep the inner end of the hole away from the face of the hearth wall so as to protect the brickwork from the effects of the current of gas blowing down from the tuyères and out the hole. This may be overdone, however, and the hole have so long a roof built up that there will be danger of the roof breaking through while the men are at the drill.

USE OF COOLING PLATES OVER THE TAPPING HOLE.

To protect the walls over the tapping hole from being cut back to an unsafe thickness, a great many plants insert a bronze cooling plate in the brickwork about 2 to 3 feet above the hole. Another method, little used, is to place a cooling pipe in a pigeonhole in the hearth jacket over the tapping hole. To protect the sides and top of the tapping hole from being cut out, cooling boxes or plates have been placed in the brickwork about the hole. Such plates are very dangerous, for if the brickwork is cut back to the plate, as has happened repeatedly, the hot iron rushing past the plate, heats it until the plate wall melts. Then a terrific explosion that throws molten iron about the cast house is almost inevitable. These devices are used rarely and are condemned by 9 out of 10 furnace men.

TYPICAL TAPPING-HOLE BREAKOUTS.

EXAMPLE NO. 1.

After a cast the hole was stopped. The blast was turned on about 30 seconds after the gun was put in the hole and while the hole was still taking clay freely. The keeper removed the gun 20 minutes after stopping the hole. The helpers were working on the iron runners, "ridding up," and were in direct line with the hole. Three or four minutes after the gun had been swung away from the hole the clay plug blew out and a blast of hot coke, slag, and gas blew down the cast house, severely burning four of the helpers. It was stated that the furnace had been working with a short hole for several casts previously.

EXAMPLE NO. 2.

Six men were opening the hole with a churn drill. The hole was reported to be about 4 feet deep and rather flat. They were on the third spell and were probably close to the skull, when the tapping hole opened suddenly and blew a shower of iron and flame over the men, fatally burning the two nearest the hole. The accident is thought to have been due to molten iron breaking through the top of the hole, but as the keeper and helper were killed this fact could not be established.

EXAMPLE NO. 3.

The hole had been stopped with an unusual amount of clay, as it had been working short. About two hours after making the cast the clay blew out and the entire content of iron ran out into the ladles, which were standing where they had been spotted for the next cast. The hole was stopped again, but in less than an hour after the next cast the iron came through at the side of the tapping hole, cut the jacket to some extent, rapidly eroded the clay, and broke out again, although the blast was taken off immediately. No ladles had been spotted, and the mass of iron on the tracks had to be cleaned up.

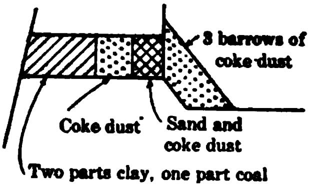

FIGURE 32.—Method of closing tapping hole when furnace is blown in.

Two hours later there was a similar breakout, the iron starting slowly but finally coming fast. This time the two tuyères over the tapping hole were plugged, and, as the hole was very wide, it was stopped by hand, two dollies being used to ram the clay in. Plugging the tuyères stopped the breakouts and enabled the wall adjacent and above the hole to build up. There was no cooling plate over the tapping hole. No one was injured.

EXAMPLE NO. 4.

A furnace was casting, and had run about half the cast. Pressure was still 16 pounds. The hole began to blow very hard, and almost immediately the entire stopping blew out, throwing a sheet of flame to the bottom of the cast house with a shower of coke, iron, and slag. At the same time a body or pool of iron, apparently held up in the furnace, broke out and overflowed the runners, sheeting the cast-house floor. Several men were severely burned.

EXAMPLE NO. 5.

The keeper and helper were engaged in ridding up the skimmer trough about an hour after a cast, when the clay blew out of the tapping hole. The clay gun had been swung back only a few minutes. Both of the men were killed.

EXAMPLE NO. 6.

At one plant the practice of allowing the furnace to blow at the tapping hole at the end of each cast was followed, the blast being kept at full pressure until the iron ran only in small dribbles; for about five minutes the blast would blow flame and coke the entire length of the cast house. This practice evidently had thinned the hearth walls and made the clay stopping virtually the sole means of holding in the iron over a wide and high area. A cooling plate over the tapping hole, 2 feet 10 inches in size, exploded and blew out just before casting time. It was thought that excessive blowing at the hole had thinned the walls until the cooling plate extended so far inside the hearth that it was unprotected by brickwork for a considerable depth, and, owing to stiff clay, melting of clay, or momentary severe scouring action, the protective covering over the plate failed and allowed the iron to cut through.

EXAMPLE NO. 7.

This furnace had two cast-iron cooling boxes, set 9 inches from the center line of the iron notch at each side. One of these exploded just as a brakeman entered the cast house. The concussion knocked him down the steps to the yard level, fracturing his back. Other men in the cast house were not injured.

EXAMPLE NO. 8.

A similar explosion occurred which threw molten iron and flame over two men and fatally burned them.

OTHER TAPPING-HOLE ACCIDENTS.

There have been in the past repeated instances of explosions at the tapping hole owing to cutting of the cooling blocks, but they are now a thing of the past.

A not infrequent type of accident, which is not regarded as a tapping-hole breakout, but is nevertheless essentially such, is when the men on the drill opening the hole are careless or uninformed as to the length of the hole, and break through the skull, or the skull opens suddenly when the tapping bar is driven through it. In any of these events there is a probability of the workmen being seriously burned.

PREVENTION OF TAPPING-HOLE BREAKOUTS.

PRECAUTIONS IN PREPARING AND LOADING THE CLAY.

Perhaps the chief difficulty in stopping the tapping hole is that the clay is likely, unless the gun is handled correctly and the clay mixed properly, to fill the hole straight ahead and not flatten out about the edges on the inside hearth walls or close up the uneven places and eroded channels in the hole itself, so that the stopping is neither safely keyed in nor firmly packed into the irregular contour of the tapping-hole surface. The clay is mixed to such plasticity that when squeezed in the hand it will flow between the fingers only slowly or with difficulty. As a rule a high grade of gray, weathered, flaky fire clay is used, because a red or yellow clay containing upward of 5 per cent of iron oxide may set too hard, although in certain districts yellow clay is used without difficulty.

When the hole does not take clay freely it is the custom to fill the gun barrel about half full of clay, then pour in half a pail of water, or loam and water, and force the clay and water into the hole. This thins the clay so that it fills the hole tightly. Many plants use water in the gun regularly. In this case the larger part of the first barrowful of clay, if two or more are used, is fed; then about half a pail of water is poured into the gun funnel after the barrel is well filled with clay, and the load shot into the hole. This procedure may be repeated a second and third time.

Another practice is to have a half-inch pipe with water connection directly over the funnel of the gun. After the first load of clay in the gun is pushed in and the gun is taking the clay, water is turned in continuously as the helper throws the clay into the funnel. This practice makes a very firm hole and one in which there is little probability of the iron working back and leaking through between the clay stopping and the walls.

There are two dangers in adding water to the gun charge. First, if water is used too quickly, before a large plug of relatively stiff clay has accumulated on the inside in front of the opening, the thin, watery clay may be shot into the hearth and, coming in contact with molten iron, cause an explosion. This has frequently happened with the result that a shower of slag and flame shot back through the hole, or has, in one instance, at least, fatally burned men engaged in changing tuyères. Second, the use of water may cause a "green" hole—one in which some part is not set thoroughly stiff, or even thoroughly dry, through which the drill may break and let the iron out before the men are expecting it and possibly cause a small explosion or shot inside the hole that will throw molten iron into the air.

OMISSION OF COOLING PLATES ABOUT THE NOTCH.

Water cooling about the iron notch is not necessary, and it should not be employed. It has rarely been found entirely satisfactory or safe in consequence of the inevitable, and occasionally severe, corrosion of the hole working back to the cooling plate. A disastrous explosion is invited by the employment of this device. It is entirely unnecessary, because careful stopping of the hole with the gun after every cast answers the purpose and is safe.

A GOOD PRACTICE IN STOPPING THE HOLE.

In stopping the hole, a good practice is to permit the nose of the gun to move back a couple of inches at a time as the hole becomes more nearly filled, in order to pack the clay firmly in that part of the hole nearest the jacket.

CLEANING THE HOLE.

At one plant the clay in the hole is dug out at least twice a week, or as much oftener as required, to a depth of 18 inches, and the sides and bottom are walled with 13½-inch by 4½-inch by 2½-inch bricks. The hole is then filled again with clay. This is to prevent cutting of the side walls where jamb pipes are inserted behind the jacket near the hole.

Another practice is to remove the clay to a depth of 18 inches and pack a stiff mixture of tar, clay, and crushed hearth-quality fire brick about a wooden form shaped like the nose of the gun. This method is employed where a cooling pipe, cast into the health jacket, bends about the iron notch at a radius of 12 to 18 inches from the center line of the notch.

USE OF A LONG PLUG OR STOPPING.

Most plants follow the practice of placing an unusually long stopping in the hole at the end of each shift. This is for three reasons: (1) So that there will be no possibility of danger to the oncoming crew from a short hole, which may break out unexpectedly; (2) to insure that the walls, which may be thinning unperceived, will build up, and (3) to assure a substantial roof or top thickness for the tapping hole. A commendable requirement followed at one plant is to have the keeper note after each cast the number of barrows of clay used in stopping, in order that the keeper on the following turn may know how the hole has been working and taking the clay, and thus proceed more safely in opening and stopping the hole.

BLOWING ON THE HOLE.

The practice of blowing on the hole for some minutes in finishing the cast, is becoming exceptional. It is admittedly poor practice, for it heats the furnace front, thins the walls, makes a safe stopping more difficult, and may lead to breakouts. It is, however, persisted in at a few plants. One reason is the rivalry between different shifts, each desiring to outdo the other in tonnage made, so the last particle of iron that can be obtained from the hearth is removed by prolonged blowing at the tapping hole. Recognition of the fact that the ultimate tonnage is governed both by conditions at the blowing room and stock house, and by watchful observance of the working conditions and adjustment of temperature, fluxing, tuyères, and other factors to meet the variations, and that an exceptionally large tonnage for a turn or a day means nothing, has led to sane treatment of the tapping hole at most furnaces, and probably will at all plants except those where other views prevail.

These views are as follows: Some furnace men still believe that the hearth should be blown absolutely dry, because any iron left in the hearth will begin to circulate, cut the walls, and cause a breakout. This condition, so they believe, is caused by currents of air from the blast finding their way down through the burden in some way, and starting a movement of the iron on the walls. As long as the iron can be prevented from moving, the walls will not be cut, and the best way to prevent this is to leave no iron in the crucible. Although this view may be held lightly by some, it is earnestly and seriously argued by older furnace men, which is the reason for recording it here. It is worthy of mention, however, that molten iron will necessarily accumulate in the hearth between casts, and that even after prolonged and heavy blowing a pool of liquid iron remains beneath the bottom of the tapping hole, which can not possibly be blown out. Nevertheless, their belief is strong enough to cause some furnace men to cling to their long-established practice. Another reason advanced is that the furnace is benefited by cleaning the hearth of "dead" iron and cinder. There are still other reasons which, for good and sufficient causes in the past, have established the practice. To dispute their validity is often hazardous, for, in spite of the advance in knowledge of conditions within the blast furnace, there are frequently actions within the furnace, or idiosyncrasies of individual furnaces, which are not comprised in the chemical-physical theories of their working. In regard to prolonged blowing at the tapping hole, the disadvantages and dangers arising from irregular working of the furnace and the tapping-hole difficulties have proved of greater weight than the benefits derived, and the practice is becoming obsolete except under abnormal working conditions.

ESPECIAL CARE NECESSARY WHEN CASTING IS DELAYED.

Especial care of the iron notch in stopping and precaution in working about the notch and in opening it are essential when the cast has had to be held for an hour or more. The load of iron in the furnace is higher and the interior pressure is more likely to blow the plug of clay out or to cause the iron to penetrate between the clay stopping and the walls or to break out suddenly when the hole is being drilled or the tapping bar is being driven in or removed. The high head of iron usually causes such rapid melting and erosion of the clay stopping that the hole quickly becomes double its ordinary or proper size and the carefully built roof of the tapping hole and interior wall of the hearth disappear. Similar results may be caused by high blast pressure, although in such a case checking the furnace at casting time has a more retarding effect.

Stopping the hole at such times is difficult. Cinder is nearly always lying in the hole and the nose of the gun does not fill the hole, and instead of the clay being forced into and beyond the interior face of the wall and packing against the face it sometimes escapes back about the nose, extends only partly into the hole, and fails to fill it completely; cinder or molten iron may escape and come back in the gun as the plunger is drawn back, necessitating a delay, the hole chills up, and no more clay can be forced in. These results may occur even with holes of normal size if the gun is not placed squarely in the hole, is not properly loaded, or if the operator is unskilled or unfamiliar with the method of operating the throttle lever. If a hole is short or hard, care must be used in drilling into it on the next cast to avoid a breakout if the drill goes through the skull unexpectedly.

STOPPING THE HOLE WITH THE BLAST ON.

To prevent the suction of the plunger and the interior pressure in the furnace displacing the clay in the hole and thus permitting cinder or iron to run into the barrel of the gun or burst out between the gun nose and the sides of the hole, one type of mud gun used at certain large furnaces has a single slide valve, worked by a hand lever, between the clay cylinder and the nozzle of the gun. The gun is placed in the tapping hole and the first load of clay driven into the hole. The slide valve is then jammed down and the plunger drawn back for recharging the clay barrel. The barrel being loaded, the valve is raised and a second load driven in, this process being repeated until one to three barrows of clay are placed in the hole.

With this type of gun the blast can be, and usually is, with a normal size of tapping hole, turned on immediately after the first load of clay is put in; but with the usual type of gun such action would occasionally be hazardous, for the reason mentioned in the preceding paragraph. It is therefore considered the safest practice not to turn the blast on full until the hole has ceased to take clay freely. However, it is desirable to have some pressure inside the furnace in order to flatten the clay on the inside of the walls within the hearth.

A typical practice, which has given minimum trouble with cinder or flame coming back and results in satisfactory keying of the stopping clay, is to drive in the first two barrels of clay with the blast off. The snort valve is then closed and the whistle signal for half pressure is blown. The pressure builds up slowly to 9 or 10 pounds, while the greater part of the second barrow of clay is forced in against this pressure, and the last two or three barrel loads of clay are pushed in with nearly full pressure on the furnace, the signal for full blast being given as the hole is almost stopped.

LEAVING THE GUN IN THE HOLE AFTER CASTING.

The gun should be left in the hole with the steam on for several minutes after the blast is fully on and the hole stopped, in order that the plunger shall hold the clay in the barrel firmly against the plug of clay in the hole and prevent it from blowing out. The gun should not be removed from the hole until the clay has set solid on the inside, which may require 10 or more minutes; 20 minutes allows a minimum factor of safety, and many plants require 30 to 40 minutes.

USE OF SPLASHER PLATE OR SHIELD.

A high, broad plate shield is sometimes provided to protect the men in case the stopping in the tapping hole breaks out violently. This is either placed over the skimmer, where it may be from 3 to 10 feet wide by 6 to 12 feet high, or is placed 10 to 15 feet farther down on the iron runners, on the skimmer side of the first junction. Many prefer the latter position, because the shield when placed over the skimmer makes difficult the use of pricking rods on the tapping hole. The skull was, of course, a necessity with the old sand skimmer, but is retained by several plants for its usefulness in protecting men from breakouts and also from coke blown down the cast house when the furnace is casting on a short hole.

MANIPULATION OF GUN.

The correct way of handling the gun, or, more specifically, the throttle lever, to meet any conditions at the tapping hole can not be given in detail. The experienced keeper or blower is guided by appearances about the nose of the gun, such as escape of gas or cinder, the "give" of the clay in the hole, the velocity or response of the plunger when the steam or air is thrown on, the sound of the steam exhaust, the way the iron has run, or the amount of blowing from the hole. By his knowledge of these and other obscure factors he is guided in the treatment of the hole at each cast, determining how much clay is required, and whether additions of water are necessary. An experienced keeper or an alert observant helper can and does handle the tapping hole so that accidents from any type of breakout are rare, whereas new crews or careless, unobservant, or unskilled keepers are in continual danger of poor holes.

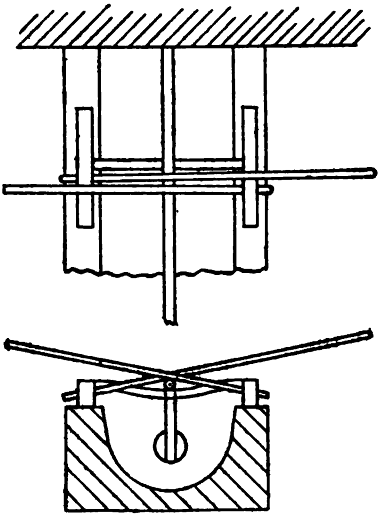

FIGURE 33.—Cross bars for holding the tapping bar.

PRECAUTIONS WITH SHORT HOLES.

A hole taking clay poorly, working short, or working high up indicates the necessity of a very deep stopping, extreme caution in drilling, and, if the trouble persists, inserting over the tapping hole a longer tuyère or one with a smaller nose. At times building up about the inside face is aided by temporarily plugging the tuyère for 24 hours. The opening of the tuyère will be easier if a bar hole is pierced through the clay to prevent it from ironing over. If the hole is very short or the wall over the hole is thin, a cold blast on the face of the clay helps to hold it between casts.

PRECAUTIONS IN OPENING THE HOLE.

In drilling up into the hole or driving the tapping bar through the skull, the splasher plate should always be placed to hold back any outburst of iron, slag, or flame and prevent it from flying over the men. On no account should any of the crew stand with their feet in the skimmer trough when holding the tapping bar. It can be held in line just as well by two pairs of tongs, one pair held by a man on each side, or more rigidly by a pair of cross bars. (See fig. 33.) There is no need or excuse for anyone getting into the trough near the hole in order to place the point of the tapping bar exactly in some particular spot unless the hole is hard, and there is no need of being there after the bar is placed. The blower should not permit such practices. The progress of the bar can be gaged by its drive when the crew are sledging it or by a chalk mark. Many plants have forbidden the practice of holding the bar by hand on account of the need of the men being near the tapping hole, where they may be caught if the iron breaks out.

Stopping is made much safer by dropping a shield in front of the splasher plate, as shown in figure 34. The use of this device is feasible at all plants. At a few of the plants where compressed air is used to blow the clay cuttings out of the hole, a plate is put over the skimmer trough and in front of the splasher, an angle iron being riveted on the front edge. This angle iron may be removable for convenience in using, if desired, a small stopping hook to clean the hole, or to observe its condition. (See fig. 35.)

This type of shield, though in use at comparatively few furnaces, has proved entirely satisfactory, whereas the solid plate has met with strenuous objection from many furnace crews who feel that they must be able to see the progress of the hole. At many plants one or two sheets of corrugated iron are laid over the trough with their front ends resting on the splasher, a hole being punched through them for the drill and tapping bar. This arrangement is satisfactory, although the sheets wear out so rapidly that the crews at times are inclined to neglect prompt replacing of a worn-out sheet and to open the hole without using any shield. Hence a shield of $\frac{3}{16}$-inch or $\frac{1}{4}$-inch steel plate is preferable.

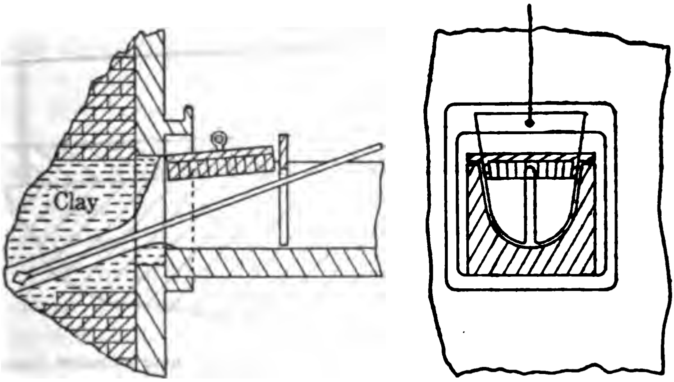

FIGURE 34.—Sections showing method of using shield.

Shields should always be used with air and electric drills. When a drill or bit strikes the iron skull at the bottom of the hole the motor usually plugs itself or works so jerkily that an experienced crew knows the skull is reached, stops drilling immediately, and uses a tapping bar to drive through the skull. Nevertheless, occasionally the drill may bite through the skull, bringing the iron out with a rush. The latest method of safely opening the tapping hole is with a steam or air percussion drill, and was developed by the Republic Iron &

Steel Co. in the spring of 1915. Mechanical drilling is preferable to hand drilling because a smaller hole may be drilled in the clay. This limits cutting of the hole by slag considerably. A 2½-inch hole is large enough for a mechanical drill.

DEPTH AND ANGLE OF TAPPING HOLE.

A final factor in the care of the tapping hole having a bearing upon hearth breakouts is the depth or angle of the hole. Infrequently the tapping hole has an angle of 45° after the rate of driving and capacity of the furnace have become normal. As has been mentioned, the depth of the hearth is variable, depending on the height of the bottom of the tapping hole, and it may be assumed that with the best constructed hearth brickwork the bottom of the hearth will be 3 to 6 feet below the bottom of the hole, the exact position of the hearth bottom being uncertain. On a 16-foot hearth, therefore, the tapping hole may with safety have a depth of, say, 2 feet 3 inches on a hearth jacket 4 feet 6 inches deep, but the same angle of tapping hole with a hearth jacket 3 feet deep, both jackets being water cooled to the bottom, would be hazardous. (See fig. 36.) The liability of a hearth breakout would certainly be greater in A than in B. Therefore the apparent advantages of a deep tapping hole in keeping the level of the iron in the hearth farther from the water-cooled cinder notch and hearth plates, giving cleaner casts and greater capacity, may conceivably give rise to an unperceived danger of breakouts. This danger should be kept in mind in endeavoring to get a production extremely large in proportion to the hearth area and depth and be given the same regard as the maintenance of the tapping hole at the proper angle with reference to the depth of the hearth jacket.

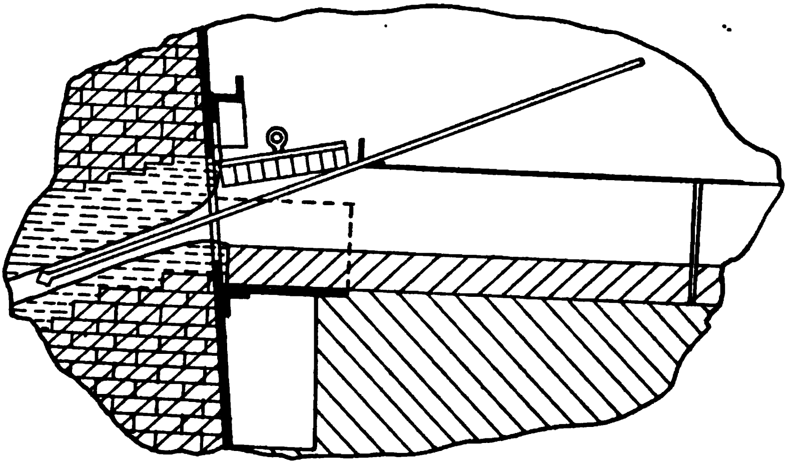

FIGURE 35.—Section of tapping hole, showing plate over skimmer trough.

CINDER-NOTCH BREAKOUTS.

The cinder notch, as it is used to "flush" or drain the slag from the hearth previous to casting, being subject to erosion and attack of slag during 12 to 18 "flushes" a day, would last but a short time if it were simply built up and stopped with clay without the blast being taken off. In fact, a clay notch could not be used unless stopped in exactly the same way as the iron notch, and this would mean turning off the blast a prohibitive number of times. Consequently, in the cinder-notch arch or opening is placed a bronze cooler, similar to a tuyère cooler; within this bronze cooler and extending inside 6 to 12 inches is an intermediate cooler, and inside the intermediate cooler is the "monkey," also water-cooled, which projects in about 3 inches. The monkey has an opening 2 to 2½ inches in diameter, and is stopped by an iron "bot" or plug which chills the cinder inside the face of the monkey. The only danger of breakouts about the cinder cooler is escape of cinder between the large cooler and the brick walls, a rush of cinder when the intermediate cooler is changed, outbursts of slag and flame when the monkey is changed, or an explosion of the intermediate and the large cooler. If iron comes over the monkey by accident, it may cut the bronze and come in contact with the water inside, when all three coolers may be blown out of the furnace, and an outrush of furnace contents follow.

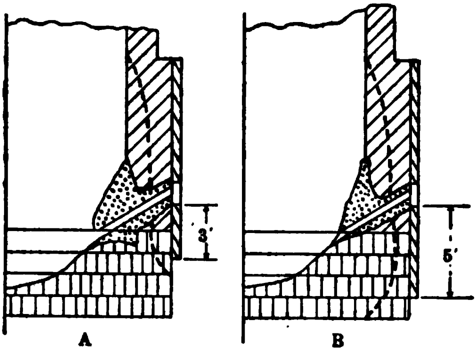

FIGURE 36.—Diagrams showing relation between depth of hearth jacket and slope of tapping hole as regards safety.

PACKING FOR CINDER-NOTCH COOLERS.

Breakouts about the cinder-notch cooler are minimized by painstaking care in setting and packing the cooler, which usually should be done after the kindling and scaffold have been placed in the hearth and bosh, and before the furnace is filled preparatory to blowing in.

The packing most generally used, and the firmest, is rope asbestos saturated with water, rammed in tightly all around from front and back with light blows of a sledge. Clay is usually plastered on the face of the brick before the cooler is inserted, and the asbestos rope is used to make a tight fit.

BREAKOUT FROM DEFECTIVE PACKING.

A serious breakout of this type happened at a furnace where the foreman packed the cooler in the brick arch with loam only. The cinder broke out beneath the cooler, cut a large hole in the hearth brickword, and ran over the cast-house floor and yard. No one was injured. The hole was filled with stiff clay and brickbats and the new cooler was packed with clay, tightly rammed in. No further trouble was encountered.

BREAKOUTS FROM CUTTING OF THE INTERMEDIATE COOLER.

Breakouts from cutting of the intermediate cooler are rare. The few known accidents of this type were caused by iron coming over on a flush or cutting the intermediate cooler from inside, which allowed the entire cooler to melt out and permitted the escape of cinder in uncontrollable quantity.

An instance of the intermediate cooler burning on the second flush through the monkey being cut illustrates a danger. Instead of waiting until casting time, 45 minutes later, or casting early, as could have been done if cinder showed at the tuyères, an attempt was made to change the intermediate bronze at once, the water being turned off and the cinder allowed to drain out. Owing to the small opening in the nose of the intermediate cooler the clay could not be placed effectively, and when a "puller" was set and the intermediate cooler jarred loose, slag burst out and severely burned the legs of a "cinder snapper." This happened in spite of the fact that the furnace had evidently been flushed dry, and shows how a furnace may hold its cinder.

Intermediate coolers should not be changed until after the furnace has been cast, unless they are cut on the first flush, and even then, depending on the hearth capacity, tonnage, slag volume, and working conditions, changing can be safely and profitably postponed to casting time.

BREAKOUTS FROM CHANGING THE MONKEY WITH THE BLAST ON.

Breakouts at the monkey are caused by trying to change a leaking monkey with the blast on. The change can frequently be made with impunity, but will almost inevitably lead to a serious accident sooner or later. Sometimes the skull of cinder which freezes on the face and sides of the monkey is stiff and thick enough to withstand the interior pressure, but it can not be relied on, however. In numerous instances where the monkey has been withdrawn with the blast on, the skull of cinder held for an instant, then bulged out, burst, and a stream of gas, slag, and coke spurted from the nose of the intermediate cooler over the men working in front. The monkey should not be changed unless the blast is completely off.

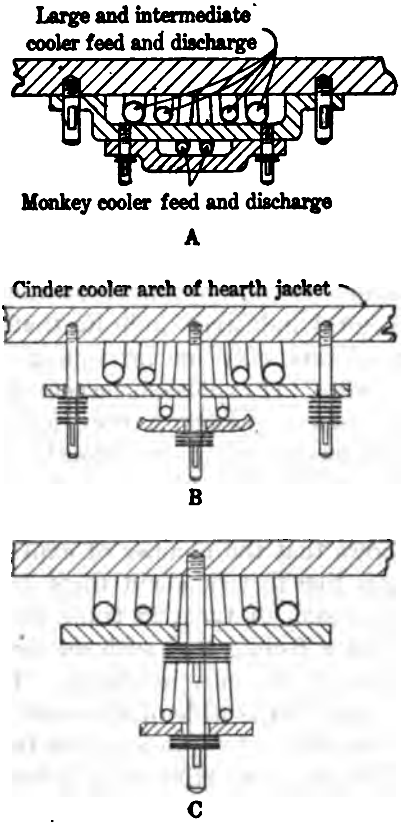
Large and intermediate cooler feed and discharge
Monkey cooler feed and discharge
A
Cinder cooler arch of hearth jacket
B
C

FIGURE 37.—Clamps for cinder cooler, intermediate, and monkey discharge pipes.

PACKING THE INTERMEDIATE AND LARGE COOLERS TO PREVENT CUTTING.

To prevent the intermediate and large coolers from being cut, clay is packed within the space inside them up to the monkey. The bot is withdrawn and the clay packed about a wooden form. This work may be performed without danger of a breakout, as the area of the skull face in front of the bot is only 3½ to 5 square inches, whereas when the monkey is withdrawn the unsupported area may be 40 or more square inches.

ARRANGEMENT OF COOLING SYSTEM.

The cooler system is held in the furnace by clamps, of which three types are shown in figure 37. Of these three, type C is least desirable. The feed and discharge pipes, in addition to having cocks near the cooler arch, should, rather than rising directly up to the supply pipe and trough, lead around the furnace at least one-third, or, better, one-half, of the circumference before being taken up, so that in event of cutting, burning, or a breakout at the cinder notch, the monkey boss can turn the water off, to avoid an explosion, without exposing himself to danger from escaping cinder, flame, or flying iron.

PART II.—GAS EXPLOSIONS AT BLAST FURNACES.

PURPOSE OF INVESTIGATION.

Explosions of gas at blast-furnace plants were in former years rather common and often disastrous to employees and to the equipment. Studies by the Bureau of Mines of the hazards in the metallurgical industries indicate that such explosions in themselves now constitute only an infrequent and inconsiderable hazard to employees. If explosions be considered with regard to damage to plant equipment, however, they assume decided significance. Gas explosions resulting disastrously are few, considering the number of furnaces in blast, but they do occur persistently.

A review of blast-furnace accidents during the past few years shows that the number of explosions is considerable. Few furnace men like to talk about them and there is virtually no mention or discussion of them in trade journals, possibly because there seems to be a feeling that such an accident reflects on the care and judgment of the men in charge. This view is warranted in some instances, but is wide of the mark in others. Attempts to contrast the proportion of explosions due to defective methods of handling gas with the proportion that is largely fortuitous are useless. It can be said that the number due to carelessness of the men in charge is extremely small, for probably nothing in the routine of furnace operation puts a man under a greater strain than his dread of a gas explosion when confronted with an unforeseen stop or the necessity of blowing in or blowing out. No matter how reliable the foreman or assistant may be, the superintendent is always present, if possible, when such jobs must be done, and the responsibility he must bear is always sufficient to insure his taking the greatest care. The uncertainty and unsatisfactoriness of attempts to state that a certain number of gas explosions are caused by unskillful handling of the gas lies in the fact that practically every furnace man has a particular way of handling the gas, each method differing radically or in some small detail from the others. These methods rest on personal experience and training and the exigencies of plant layout, or have been handed down from man to man; some of these methods provide sufficient safety in most cases, but have failed to provide for it when some abnormal factor was introduced. To illustrate:

Few furnace men will state that they can not handle furnace gas with practical immunity from explosions, yet few furnace men with an experience of 10 or more years have not had experience with a more or less serious explosion.

In undertaking to summarize and furnish useful information on the subject of explosions to those working about blast furnaces, it is believed that an investigation into the properties of blast-furnace gas, its ignition temperatures, the explosion limits of mixtures of gas and air, and the agencies accelerating and retarding explosions would be of less practical value than pointing out where the explosions occur, how they are caused and may be prevented, and giving a review of those methods of handling gas that promise greatest security. Although much work may be done on the explosive characteristics of blast-furnace gas, yet, even with the resulting information in hand, furnace men will be confronted from time to time with conditions necessitating possibly dangerous operations. At such emergencies a knowledge of the limits of explosibility would only serve to emphasize the danger, which is sufficiently well known, and would be of little practical use. On the other hand, the recognition and discussion of certain practices as being inherently hazardous not only separates them from careless or reckless methods or practices but places them definitely in the category of operations occasionally necessary but essentially dangerous. By recognizing them as such the incident difficulties can be met much more satisfactorily than by discussing certain conditions and dismissing them with the statement that under such circumstances gas may explode, and that therefore these circumstances are to be avoided. At times some circumstances can not be avoided, so the problem becomes one of so handling the gas that the least hazard is introduced.

For the above reasons it has seemed best to place emphasis on methods of practice, and that plan is followed in this report, with the exception of the following discussion of the theory of explosions.

THEORY OF GAS EXPLOSIONS.

Blast-furnace gas, which rarely contains more than 30 per cent of combustible gases, can not be classed as a dangerously explosive gas; and when the furnace is in normal operation the gas is easy to handle and the hazard from an explosion remote. In case the furnace has to be stopped for any reason, an explosion at some part of the equipment, either in taking the furnace off while the blast is off or in starting up again, is always possible if air gets into the furnace top or the gas mains. As a rule, such explosions, if they occur, are confined largely to small "top shots," "gas kicks," "puffs," or "cracks," and are usually harmless, although some are violent enough to lift gas-main explosion doors, knock burners off their stands, split dust catchers, and blow off the downcomers.

Blast-furnace gas varies in composition, but for the purpose in hand, may be assumed to contain 13 per cent carbon dioxide, 24 per cent carbon monoxide, 3.5 per cent hydrogen, and 59.5 per cent nitrogen. The reactions of the hydrogen and the carbon monoxide in an explosion are as follows:

$$2\,CO+O_2=2\,CO_2.$$
$$2\,H_2+O_2=2\,H_2O.$$

Dixon, an English chemist, stated in 1880 that a mixture of carbon monoxide and oxygen will not explode or even burn unless a trace of moisture or of some gas containing hydrogen is present, the accelerating effect from moisture being at a maximum when the gas has a temperature of 45° C. and is saturated with moisture. Materials such as heated fire-clay surfaces, nickel, copper, and oxides of these and other metals also accelerate combustion or explosion. Carbon monoxide in air ignites at 644° to 658° C., but in contact with a hot fire-clay surface it will ignite at 500° C., or with copper oxide, at 300° C. This lowering of the ignition temperature is caused by "catalysis" or contact action, which is the increased activity of the gases from being absorbed and concentrated on the clay or metal surface. The catalytic power of all these materials is highly stimulated by long previous exposure to the combustible gas, and this increase of power is usually very lasting. It is probable that in the handling of furnace gas the heated walls, partly reduced oxides, deposited carbon, and other materials possessing catalytic properties, are important factors in causing combustion at low temperatures and thus starting explosions.

The ignition of a mixture of furnace gas and air does not necessarily result in an explosion. The velocity of propagation of a carbon monoxide flame is relatively slow compared to that of hydrocarbon gases, and, as in gas mains or furnace tops, there is rarely a perfect mixture, such as is found in gas-engine cylinders, the maximum theoretical generation of explosive power is probably seldom reached.

The mechanism of an explosion must be followed to understand the results observed when a mixture of gas and air is ignited. An explosion of gas and air consists of an "initial combustion," a "detonation," and a "retonation." To have these take place the explosion must occur within a closed vessel or cylinder such as a gas main. When an explosive mixture of air and a combustible gas is ignited it burns outward from the point of ignition and a compression wave starts, which travels ahead of the flame and more rapidly, with the velocity of sound, through the unburned gas. This compression wave upon reaching a barrier is reflected back in the direction from which it came, and on meeting the advancing flame, retards the latter. The reflected compression wave, however, continues to pass through the hot and probably still burning gases and an instant later reaches the flame traveling in a direction opposite to that of the other flame, accelerates it, and quickens the combustion. This flame continues to move forward with rapidly accelerating velocity until detonation is set up. Detonation is abrupt, and the union of the oxygen of the air with the combustible gas proceeds in the form of an explosion wave that generates an enormous force as it advances with great rapidity through the gaseous mixture. When detonation is taking place, the explosion wave is transmitted through the explosive mixture by each successive layer of the mixture, being instantaneously heated to the ignition point by the compression resulting from the accelerated velocity and the augmented temperature of the burning gas, both being induced by the compression wave.

At the point where detonation begins a "retonation wave" starts back through the still burning gas. Combustion is much slower than detonation, so that at the moment detonation starts combustion is still taking place in layers of gas some distance behind the flame front. The "retonation wave," in passing back through these burning gases, quickens their combustion, and when it arrives at the other flame front causes detonation there, provided that ignition has taken place at such a distance from the barrier that the flame front reaches it just as the "retonation" wave arrives. This setting up of reflected waves through "retonation" explains much of the shattering effect sometimes observed after a severe explosion of blast-furnace gas.

This process of initial combustion, detonation, and retonation takes place in an increditably short space of time. As compared with ordinary combustion, its characteristics are extreme rapidity of chemical action, abrupt attainment of maximum temperature and pressure, and subsequent rapid, almost instantaneous, cooling of the gaseous products of explosion. The velocity of an explosion wave, hence the violence of a detonation, is independent of the shape of the inclosing walls, whether straight, circular, or zigzag, but is retarded by an inert gas. If hydrogen is added to an explosive mixture of carbon monoxide, its effect will be to increase the velocity of the explosion wave, as is illustrated by the propensity of furnace gas, rich in hydrogen, to "kick." Nitrogen, carbon dioxide, and oxygen retard the velocity. The velocity and violence of the explosion are greatest when the combustible gases, such as the carbon monoxide and hydrogen of blast-furnace gas, are present in slight excess of the amount theoretically required for complete combustion.

PRESSURE PRODUCED BY EXPLOSION.

The pressure produced by a gaseous explosion can be calculated. For instance, the calculated pressure of a blast-furnace gas analyzing 12 per cent CO_2, 26 per cent CO, 3 per cent H_2, and 59 per cent N_2, exploded in a closed vessel with the amount of air theoretically required for complete combustion, is 76 pounds gage. The pressure is generated entirely by the volume increase due to the high temperature of combustion, not by any increase in volume as the result of the reaction. Two volumes of carbon monoxide or hydrogen combined with one volume of oxygen give two volumes, not three or more. One cubic foot of the gas mentioned above burned with 0.7 cubic foot of air forms only 1.55 cubic feet of gaseous products, the explosion or combustion starting and ending at 60° F. and at atmospheric pressure. However, if the thermal capacity of the gas is divided by the specific heat of the gas the apparent theoretical temperature is 2972° F. At this temperature 1.55 cubic feet of products at constant volume would give a theoretical pressure of 76 pounds gage. Experimental explosions of a gas similar to the above have given gage pressures of 18 to 48 pounds; the larger the inclosing vessel the higher the observed pressure mounted. In a general way the maximum pressures observed in closed vessels are always less than the theoretical calculated values. This is because of the variation in specific heat with the temperature, not allowed for in the calculation, and because the calculation assumes that the whole heat of combustion is communicated without loss to the products, which is not true. It is safe to say that the maximum calculated pressures are rarely or never attained in explosions of blast-furnace gas in mains, just as in experimental observations, and chiefly for the same reasons:

EXPLOSIVE LIMITS OF BLAST-FURNACE GAS.

The results of an examination of the explosive limits of two samples of blast-furnace gas taken from the furnaces of the American Steel & Wire Co., at Pittsburgh, Pa., made by G. A. Burrell and A. W. Gauger, of the Bureau of Mines, are given below. These results do not cover a wide enough range of composition to be inclusive of all gases encountered with various practice, but are of interest in connection with the subject under discussion.

Composition of gas used in tests.

Constituent.		Sample 1.	Sample 2.	Average.
CO_2	per cent	9.97	10.01	9.99
O_2	do	.49	.49	.49
H_2	do	2.67	2.66	2.67
CO	do	27.57	27.42	27.50
CH_4	do	.39	.29	.34
N_2	do	-----	-----	59.01

The explosion apparatus consisted of glass vessels, approximately 8½ inches long and of 900 c. c. capacity. Ignition was made at the center with an arc formed by breaking the contact of the electrodes while current was passing through them. The results of explosion tests with various percentages of gas in the mixture were as follows:

Results of explosion tests.

Gas in mixture. per cent.	Notes by observer.
20.0	No ignition.
30.0	No ignition. Small yellow flame played iintermittently about arc.
39.2	No ignition. Large flame about arc, about ¾ inch high at first.
44.8	Slight puff heard when arc was first drawn. A ½-inch flame played about arc. No self-propagation through mixture.
44.9	A small tongue of flame started at the arc (which broke immediately) and rose upward through the mixture. Flame did not fill bottle and combustion was incomplete.
45.7	Ignited immediately; flame shot downward about 1 inch and then rolled upward in a ring. No other downward propagation noted.
46.5	Ignited immediately and blew waxed paper off top of vessel.
50.0	Immediate ignition. Flame traveled upwards, and more slowly downwards in rings.
56.2	Ignited immediately and blew paper off top of vessel.
57.0	Do.
61.0	Do.
62.0	Ignited immediately; flame filled bottle.
64.5	Did not ignite at first, but on drawing a wider arc the mixture ignited and blew off paper cover. Ignition was probably due to the mixture becoming diluted with air.
65.0	Slight inflammation about arc; snapping sound. No self-propagation.
66.0	No ignition.
67.9	Do.
72.0	Do.

The amount of air theoretically required for combustion in the first sample was 0.723 cubic foot per cubic foot of gas, and in the second sample it was 0.711 cubic foot per cubic foot of gas. These results being in close agreement, the average, 0.72 cubic foot of air per cubic foot of gas, may be taken as the theoretical amount of air required for complete combustion. This corresponds to a mixture of gas and air containing 58.1 per cent gas and 41.9 per cent air as being an ideal mixture for complete combustion.

These results show that the limits for complete inflammation of this gas lie between 46 and 62 per cent of gas or between 54 and 38 per cent of air in a gas-air mixture. The lower limit of explosibility is farther from the proportion theoretically required for complete combustion than is the high explosive limit, as shown below:

High limit (excess gas), 62 per cent gas, 38 per cent air.

Theoretical proportions for complete combustion, 58 per cent gas, 42 per cent air.

Low limit (excess air), 46 per cent gas, 54 per cent air.

This difference is of little significance as regards the possibilities of gas explosions about furnace plants, except that in the event of air entering a main and mixing with gas until the lower explosive limit, 38 per cent air, is reached, the possibility of explosion is not passed until a large excess of air has entered, the explosive range being 4 per cent less and 12 per cent more than the theoretical amount required for conditions most favorable for an explosion.

EXPLOSIONS AT HOT-BLAST STOVES.

That it is not always easy to burn blast-furnace gas instantly—that is, explosively—is demonstrated daily at blast-furnace stoves. The gas does not burn instantly but rather intermittently. As the air and gas mingle at the nose of the gas burner complete combustion does not take place immediately, even with approximately thorough mixing, as is shown by the length of the flame, the constant effort being to get a short flame. That it is possible to approach explosive conditions and set up "retonation waves" was demonstrated in 1913 at a plant in the Pittsburgh district, where a set of patented mixing burners were installed at a set of stoves using cleaned gas. From the minute the furnace was blown in and the gas turned into the stoves a deafening vibrating roar set up, and the burners had to be removed within 24 hours. A similar noise has been noted in carrying out explosion tests of blast-furnace gas in a 6-inch tube 21 feet long. When the gas inside the pipe was ignited the pipe vibrated and gave out a sound similar to that noted at the burners mentioned, though not as loud.

But as a rule, if the gas ignites promptly at the burner, there is no explosive tendency. The nearest approach to explosive combustion after the gas is lighted is seen when an excess of gas turned in the stove, and for some reason—for instance, to see how the gas is burning—the stove tender opens an air or cleaning door. Under these circumstances the burning gas sometimes puffs from the door of the well, a hazard of stove tending that at times results in painful if not serious burns..

A similar condition is encountered when the stove checkers are more or less plugged up, or the stack draft is insufficient, and at times when cold washed gas is burning in a cold well. In the latter case, the presence of excess air causes the flame to be extinguished at the base of the stove well, a mixture of air and gas fills the well, ignites at the checkers, and travels down in the well with almost explosive violence, and bursts out of the air-inlet doors. At such times it is not safe to go within 8 or 10 feet of the doors. The only practical precaution against these conditions is to so adjust the gas and air on turning the gas on that as long a flame as possible is produced, and then slightly to reduce the length of the flame by admitting a slight excess of air. This is, of course, the method of experienced stove tenders, and the close regulation of the air is often a source of surprise to those undertaking to increase the efficiency of stoves by analyses of the stack gases. Shortening the flame may also be produced by a deficiency of air, but this error will, of course, result in the liability of an outrush of flame from the well.

More nearly allied to true explosions is the hazard of permitting gas and air to pass into a stove, when it is put on gas, without its lighting immediately. When a stove well is hot, the gas should ignite almost instantly, and it usually does, with perhaps a slight puff. However, gas turned in a cold stove, or into a run-down stove, in which the well is cold and the checkers hot, does not ignite quickly. In the latter case the gas and air may fill the combustion chamber, ignite at the top, and then flash down into the well and "kick" out about the burner with explosive violence, throwing hot dust and flame in the face of the stove tender should he be standing about the burner watching for the gas to ignite or about to pull the gas burner back. In the case of a cold stove, the danger lies in a stove tender allowing the stove to fill with gas and air before lighting the gas at the burner. In such event the same "kick" of burning gas is to be expected. Such practice is especially dangerous when gas from a furnace that has been shut down from a leaking tuyère or other leaky bronze is turned into a stove, or when gas from a newly blown-in furnace is being turned into the stoves. Several years ago a stove was entirely destroyed by an explosion caused by lighting the gas after the stove was full, and the stove tender was killed. There have been several injuries from this cause, and the precaution of having a good fire in the well or at the burner door is especially called for in turning "wild" gas into stoves. In fact, the elementary and only practical precaution to obviate all such explosions lies in providing a fire of burning waste, wood, coke, or coal at the gas-door inlet of cold stoves before the gas is turned on.

In the event of gas being turned into a run-down stove and failing to ignite immediately the gas should be turned off instantly and a pilot fire provided, as noted above. In turning the gas on, the stove tender should step away quickly from the burner after racking it into the burner door frame. Many burners are so badly designed that the stove tender in racking the burner into the door is compelled to stand within 2 to 4 feet of the opening, and must spring away as he racks the burner in; also if the gas does not ignite promptly, he is in some danger should it ignite while he is racking the burner back. Burners should be so designed—as several types are—that the stove tender can stand 6 to 10 feet from the burner door when turning the gas on. Stove cleaners should be warned of the danger of flame flaring out and burning them if the stove door of a well is opened with the gas turned on full. The gas should be turned off in part before a well-cleaning door is opened.

EXPLOSIONS OF GAS AT BLAST-FURNACE BOILERS.

The causes of explosions at the boiler house are in general the same as at stoves. An explosion in the combustion chamber takes place, as a rule, when the flow of gas is interrupted and air works back in the gas main through the boiler burners; or when the gas is shut off, on account of the furnace being stopped, and, on starting up, the gas, mixed with air, completely fills the boiler setting before it ignites or is lighted; or when gas is turned into the boiler on blowing in or after cleaning the tubes and setting. Aside from explosions due to air in the mains, which should probably be discussed under "Explosions in gas mains," explosions at the boiler setting can be avoided only by never allowing gas to enter the boiler setting, beyond the combustion chamber, unburned.

Except when the furnace is shut down for changing tuyères, or some similar cause, in which case the gas is drafted back through a stove, checking the flow of gas at the boiler (or stoves) is not a matter of much hazard even at an isolated furnace, as there is always an appreciable amount of gas coming over from the furnace at a casting stop. Even if this amount of gas is not sufficient to keep up combustion in all boilers, combustion chambers, variously called "dog houses," "Dutch ovens," or "fore chambers," are usually provided which present a large surface of incandescent brickwork, which ignites the gas the instant it is reintroduced into the boiler upon the furnace being put on blast again.

In addition to this automatic safety provision, it is almost the universal custom to keep a small coal or coke-dust fire going on the grates of gas-fired boilers, and especially when the flow of gas has been reduced for any reason. Also where more than one furnace is connected to the boiler-house gas main there is always enough gas from the other furnace or furnaces in operation to keep up a steady flow to the boilers, whereas at an isolated furnace, and usually at a group of furnaces, it is usually necessary to fire coal during even a short furnace shutdown in order to generate steam for the pumps, compressors, and electrical equipment. For these reasons interruption of gas current at the boilers and loss of gas are only exceedingly remote sources of explosions.

In lighting cold boilers the gas occasionally explodes or "kicks," especially when the gas is "wild" after a shutdown or on blowing in. Although it is the rule to provide a good coal fire, there have been instances where the gas did not light until it had filled the boiler setting, and upon ignition puffed out and burned men standing near the boiler burners and doors. Sometimes the gas is cold, wet, or difficult to burn, and in these cases may fail to ignite promptly. When gas is turned into a cold boiler, burning oily waste should always be placed at the nose of the burner, in addition to a blazing coal fire on the grates. If the gas does not light promptly the gas burner should be shut off and time allowed for the setting to clear before another attempt is made to light the gas. Serious injuries have been caused by the nonobservance of these elementary precautions. The fact that these accidents can and do occur should lead to the use of every precaution which will either minimize their possibility or surely prevent them.

The handling of gas at stoves and boilers at shutdowns, blowing in, or blowing out is discussed under these respective headings.

EXPLOSIONS AT GAS MAINS AND IN FURNACE TOPS.

GAS-SEAL EXPLOSIONS.

Explosions of gas at the furnace top, known as "top shots," and in the dust catcher, downcomer, and gas mains for the most part take place on stopping the furnace, practically the only exception being at skip-filled furnaces, where the doors on the gas seals are sometimes opened while the furnace is in operation, in order to inspect the bells or to dislodge material from the little bell, lumps or scrap sometimes lodging between the bell and the extension or lips of the receiving hopper. Occasionally when the gas-seal door is opened to inspect the bell the gas which fills the space above the large bell, under the gas seal or hood, ignites spontaneously and flashes out of the door onto the men who have just opened it. At some plants it is the practice, in cleaning the top, to open the gas-seal doors and shovel into the opening over the large bell the débris of coke and ore spilled from the skip. Men have been injured in doing this, the gas puffing out of the door in a sheet of flame while they were standing near. The explosions are slight, as regards violence, and have not been known to result in damage to the furnace itself. They are noteworthy only because of the occasional serious burns inflicted; three such accidents resulting in disability for periods ranging up to 10 weeks are known.

In opening gas-seal doors for any purpose, the little bell should be kept closed until the door is opened to prevent any air drafting in at the instant the door is opened. As soon as the door is opened the little bell may be lowered, thus drafting the gas out. As a rule, it is necessary to open more than one door, either for illumination or for access to the bells, but before the second door is opened some burning waste should be thrown in on the large bell to light the gas. This gas will usually puff when ignited, so it is essential to keep away from the door until the gas has lighted. In carrying on the work the gas should be kept lighted by throwing in burning waste from time to time, and the little bell should be kept open. At one plant where injuries from this cause have received especial attention, the possibility of explosion has been effectually removed by leading a ½-inch steam line into the base of the gas seal, through which steam is blown, with the little bell open, for 10 minutes before the doors are opened. This displaces the gas entirely, and the only precaution necessary is to keep the gas lighted while the bell is being cleaned or other work is being done.

If the gas-seal doors are opened merely to examine the bell to see whether it is cleaning properly, or to inspect the distribution on the bell or the cleaning of the furnace top, the blast is not usually taken off the furnace, although sometimes it is checked. At a few plants, however, if it is necessary to open the gas seal, the water-seal valves at the dust catcher are closed in case the furnace is connected to another operating furnace through a common gas main, and steam is turned into the dust catcher, the bleeder is opened, the furnace shut down, and the gas drafted back through a stove. The doors on the gas seal are not opened until steam has appeared at the top of the bleeder pipe. When this is done there is little danger of a gas explosion in the gas seal on top while work is being carried on, provided care is taken to light the residual gas in the seal at the time the doors are first opened. These precautions are occasionally observed for work in which it is not even necessary to enter the gas seal. When it becomes necessary to enter the gas seal or receiving hopper, it is the almost universal practice to take the furnace off, and also to observe the precautions mentioned above. This rule is deviated from in some instances, as is known from accidents resulting thereby, but only infrequently with the open or tacit consent of the superintendent; more often a foreman or helper takes a chance of a slight "puff" or of being "gassed" in order to get the work done quickly. All work about gas seals should be done under the supervision and subject to the orders of the general foreman or boss blower, in order that the necessary precautions, the need of which may not be apparent to foremen and repair or labor gangs, may be taken.

TOP AND DOWNCOMER EXPLOSIONS.

EXPLOSIONS DURING EMERGENCY STOPS.

Probably the most dreaded emergency job about a blast furnace is that of repairing a broken water or steam main or encountering a similar misfortune that necessitates taking the furnace off instantly without any preparation or opportunity for consultation. At least one destructive explosion has occurred in such a stop, and there have been many explosions under similar circumstances which fortunately did not cause much or any damage. The trouble at such times is that the water is lost, the steam is lost, and the blast is either lost or has to be taken off as soon as it can be. In such event the assistance of water-seal valves in isolating an intricate gas-main system from the furnace and in isolating units of the gas main from each other is lacking, the use of steam in the gas mains is not possible, and the fact that the blast is off may make imperative such haste that the superintendent or foreman must delegate some detail of the work to some one else; confusion results, a blunder is made, or some hazardous expedient is resorted to, and the gas ignites and explodes. The usual result is only an extremely violent "crack" at the top of the furnace, but downcomers may be blown off and dust catchers wrecked, or the bell and hopper may be blown out of the furnace.

One practice in such emergencies is to shut all the gas burners at the stoves and boilers, shut down the fans or washers at the gas-cleaning plant, open the top bleeder, and let the furnace stand with two to four tuyères open. Every opening on the gas-main system, including burners, dust bells, explosion doors, and bleeders, is kept shut, and no attempt whatever is made to draft the gas out of the mains. The draft up through the furnace from the open tuyères generates sufficient gas to keep an appreciable stream passing out of the bleeder, and by partly closing the bleeder sufficient pressure can be maintained in the top of the furnace to prevent gas drafting up the downcomers and drawing air into the mains through leaky doors and burners. This method of meeting the emergency is costly because of the damage to uncooled plates, tuyères and tuyère coolers, the loss sometimes amounting to thousands of dollars, but it is sometimes preferred to the possibility of an even more expensive explosion.

Some plants prefer to draft back the gas through a stove. This procedure quickly stops any emission of gas on top, draws air in at the bleeder where it mixes with gas drifting from the gas mains up the downcomer, and may result in repeated sharp explosions in the top of the furnace. These "top shots" do little or no damage on top, and cease in the course of a half hour as the top of the furnace cools down, but are a source of alarm to the crew.

Another variation is to close all the burners along the gas main and to open a door at the end of the main. The top bleeder is then opened and the gas drafted out of the mains through the furnace top, the tuyères being plugged with clay. This procedure should displace all gas in the mains with air, the air as it is drafted in pushing the gas ahead of it. There is more or less mixing of air and gas, however, along the face and especially in the large space within whirlers and dust catchers. This mixture as it reaches the top of the furnace may cause a vigorous "top kick," which may sweep down the downcomer into the dust catcher and develop into a violent and costly explosion.

The most dangerous practice of all, and one probably seldom followed except by a blunder, is to have a burner or dust bell open near the furnace, in addition to having a burner or door open at the end of the gas line. Under these circumstances, with a current of gas moving toward the furnace top, air is drawn in with the gas, not behind it. This results in thorough mixing of gas and air, forming perhaps a very explosive mixture. These or similar methods or mistakes have caused very destructive explosions.

The method of action offering the least possibility of explosion at a shutdown, when the water, steam, and blast are lost simultaneously or in quick succession, is to close the cold-blast and hot-blast valves at the stoves and drop the blowpipes at once. The burners at the stoves and boilers should be shut at the same time. The tuyères can then be plugged with clay, the bleeder being opened sufficiently to afford partial relief to any tendency of gas blowing out at the tuyères, for, as the tuyères are usually burned at this time, the gas is apt to be "wild" and to blow out excessively. After the gas line has been inspected to make sure that all dust bells are tightly closed and the burners shut, the bells can be opened on top of the furnace. Although a few top shots may occur they are less likely to cause serious damage when a vent is provided for escape of pressure through the bells. The furnace can be allowed to stand in this condition indefinitely or after about four hours the farthermost burner, or door in the gas main, can be opened and the gas drafted out the furnace top through the bleeder.

The fact that explosions occur at the furnace top and in the downcomer at such times, when the temperature in the top immediately before stopping the furnace is only 350° to 550° C., may be explained by the assumption that the brickwork even at these temperatures may start combustion, and also from the fact that carbon, carried out of the stock by the gases, may have deposited in places on the walls. This carbon dust is highly pyrophoric, and in contact with air emits sparks or burns easily and may therefore ignite a mixture of air and gas at exceedingly low temperatures and thus initiate an explosion on top. The essential factor in preventing such top explosion obviously lies in not permitting any mixture of air and gas to draft up into the top of the furnace until sufficient time has been allowed for the top to cool. In view of the fact that there are innumerable small leaks along a gas main at burner doors, explosion doors, or at bleeders on dust catchers or ends of mains, and that gas in cooling from 450° to 200° F., contracts in volume approximately 30 per cent, and that with a bleeder or bells open on top the gas in the mains tends to drift up the furnace, there is every probability that, in case of shutdown, in the course of a few hours the mains will draw in a considerable amount of air and at a certain distance from the end be filled with air and gas in proportions closely approaching an explosive mixture. For this reason it is considered safest to draft out the mains rather than allow them to remain in such condition indefinitely, or even longer than is necessary for the top and downcomer lining to cool off.

On account of the uncertainty attending practice at these emergencies it is desirable to provide an easily closed valve, of the slide-gate or bell type, at each outlet from the dust catcher. These valves are most useful at routine shutdowns for repairs, or for temporarily isolating parts of the main, and should be supplemented by water-seal valves, as the mechanical type of valve is rarely gas tight, and in shutting down for repairs the prevention of gas leaks is essential. The use of the mechanical valve is largely to insure against high gas pressure blowing the water out of the water-seal valve, the latter taking care of any leakage of gas past the mechanical valve. This valve should therefore be inserted primarily with the idea of its supplementing the water-seal valve at ordinary shutdowns, and second, of being available in an emergency when the water-seal valve, steam pressure, and all routine auxiliaries are lacking. In order to fulfill these specifications the chief requisite of the valve is not necessarily tightness against gas leakage, but, rather, ability to be shut very quickly in order to isolate the furnace, downcomer, and dust catcher from the remaining cleaning and gas-main equipment. The ordinary goggle valve is an example of a type which does not conform to this requirement, as it is virtually useless in an emergency shutdown.

When such valves are provided, isolating the furnace from the gas mains is easy, the possibility of "top shots" or bad explosions becomes correspondingly remote, and the gas mains can easily be cleared by opening a burner at some point and drafting the gas out through a cold boiler or stove, a downleg at the other end of the main being opened to let air enter.

EXPLOSIONS AT SHUTDOWNS FOR REPAIRS.

Where the gas-main system is provided with water-seal valves or any type of mechanical valve it is easily isolated from the dust-catcher, and this is usually done when the furnace is shut down for repairs on top or at the mains. The possibility of an explosion is therefore practically eliminated, especially as steam may be used at such times as profusely as desired in drafting out the dust catcher and downcomer. Even at isolated furnaces where no valves are provided at the dust catcher and steam is not used, the chances of an explosion are remote. The work can be planned ahead, and performed deliberately, and aside from an occasional more or less vigorous "kick" on top, no trouble is encountered. As previously stated, three serious explosions within the past three years are known; in two of these instances the downcomers were split and in the third the stove burners were blown off the stands, no one being injured. In one of the cases where the downcomer was split, the explosion occurred several hours after the furnace had been taken off, and the furnace had been allowed to stand all night with the bleeder and bells shut, one gas burner open, and all dust bells shut. In the second case the explosion took place about half an hour after the furnace had been stopped, the bells had just been opened, and the gas was being drafted out through the top, but the gas on top had not been lighted.

PRACTICE DURING STOPS FOR REPAIRS TO TOP OR MAINS.

The following details of various practices at stops for repairs were furnished by the superintendents of a number of plants:

PROCEDURE AT PLANT 1.

In all shutdowns for repairs to the top or mains, the furnace is isolated from the other furnaces and the gas is lighted on top. The method of lighting the gas is as follows: The blast is thrown off the furnace, the tuyères are tightly plugged with clay, and the water valves are filled, sealing the gas off from other furnaces. The bleeder valve is opened and while the tuyères are being plugged the doors on the gas seal inclosing the hopper and the little bell are opened. After the foreman in charge has personally seen that this has been done, he opens the main bell. If after a certain time the gas has not lit, it is ignited by throwing a lighted piece of waste in, and a man is stationed on top whose sole duty is to see that the gas is kept lit. In order to have the top as open as possible, the explosion doors are also opened. In starting up after a stop, during which the gas was lit, the method of procedure is as follows: The bleeder valve and explosion doors on the top are closed. The explosion doors on the gas seal over the hopper are also closed, but the small bell is not closed until after the main bell is closed. After the foreman in charge finds everything safe, he closes the big bell and turns a low-pressure blast to the furnace. After the downcomer and dust catcher are filled with gas, as indicated by allowing gas to flow out of a dust pocket near the water-seal valve, the water valve is opened and gas allowed to flow over into the gas main.

PROCEDURE AT PLANT 2.

When possible, stops for repairs are always made directly after a cast. In taking the furnace off, when the first check is blown, the mixer valve is closed and water is turned into the water-seal valve with the idea of having it full at the end of the cast. As soon as the furnace is done casting and both water seals at the dust catcher are full both of the bells on the furnace and the dust-catcher bell are opened and gas allowed to draft up and out. The engines are kept turning over with the snort valve open, the mixer valve closed, the cold-blast valve closed, and the hot-blast valve and chimney valve open, thus drawing the gas away from the furnace. The blowpipes are then taken down, the tuyères plugged, and the engines stopped. In starting up, the blowing engine is started turning over with the snort and cold-blast valves open, the mixer valve shut, and the hot-blast valve closed. The blowpipes are put in place and the water-seal valves opened, permitting gas from other furnaces to flush the air out of the dust catchers and downcomer through the top of the furnace. When the dust catcher and furnace top are cleared of air the bells are closed, the starting signal blown in the engine room, and the hot-blast valve opened.

PROCEDURE AT PLANT 3.

In case men are required to go upon the large bell of the furnace, or into gas mains, or work about explosion doors or bleeders on top the procedure is as follows: The furnace is shut down and the bell valve at the dust catcher closed; the gas between the valve and the top of the furnace is expelled by steam. The openings of the dust catchers and downcomer between the valve and the top of the furnace are then opened, admitting air. While steam is being blown into the dust catcher the caps on the tuyère stocks are removed, the tuyères plugged with clay, and sufficient air blown through the hot-blast main to make a barely perceptible draft at the cap opening. The gas is then lighted under the bell and a small fire maintained by constantly feeding light wood down through the try-rod hole. The large bell is opened wide and the bleeder left open.

PROCEDURE AT PLANT 4.

In making short stops up to 24 hours in length the furnace is flushed and cast, the bleeder opened, and the mixer valve closed. The water-seal valves connecting the furnace dust catcher with the large general gas main and with the stoves are then closed, the large bell and small bell are opened, the blast-line snort valve opened, and the cold-blast valve in the stove closed. If, after a short interval, the gas in the furnace top does not ignite, it is lighted with flaming waste, after which several explosion doors are opened and the large and small bells blocked in their open position. The dust-catcher bell is locked closed to prevent anyone tampering with it, the blowpipes are taken down, and the blowing engines stopped. In starting up, the engines are started turning over against the open snort valve, the blowpipes are put up, the stoves opened, and a low-pressure blast turned into the furnace. As soon as gas appears on top, which is almost instantly, the large and then the small bell are closed, the bleeder being left open. The explosion doors are shut while the blowpipes are being put up. The dust-catcher bell is opened and when gas is escaping in quantity the water valves are opened, sending the gas over into the mains.

PROCEDURE AT PLANT 5.

When a stop of some length is made, precautions are first taken to see that the gas mains are sealed tightly at explosion doors and gas burners, in order that as little air as possible shall enter the mains with the gas, and to prevent accidental ignition of the gas. When the furnace is stopped the tuyères are plugged and the bleeder opened on top to draw off gas rising from the stock in the furnace. The gas mains are kept closed from the furnace top to the ends until the furnace is fairly cool. In case work is to be done on the gas mains, both bells are opened wide and the burners on the last boiler on the gas main are pushed in so that there is a suction through this boiler from the main. In starting up, the same boiler burners mentioned are opened so there will be a suction through to the end of the main.

SUMMARY OF METHODS AT THE FIVE PLANTS.

The chief points in the procedure at these plants may be summarized as follows:

1. Keep the engine turning over until the blowpipes are dropped.
2. Have the water seal or other valves closed at the end of the cast.
3. With gas valves shut, open the bleeder and draft gas back into the stoves.
4. Plug the tuyères and open the little bell and gas-seal doors.
5. Lower the large bell after the tuyères are plugged.
6. Light the gas on top if it does not ignite of itself, and keep it lighted.
7. Do not attempt to light the gas on top or open the large bell or draft out the downcomer while the tuyéres are being plugged, as a "top shot" may throw coke and flame over the men working about the tuyères.
8. In event of there being no seals at the dust catcher, be sure that the gas main is tightly sealed. Block or lock the dust-catcher valve shut, open both bells on top, and open the farthermost burner on the gas main.
9. In starting up, open the valve at the dust catcher and permit gas from another furnace to displace the air in the dust catcher, downcomer, and furnace top before closing the bells.
10. At an isolated furnace open the burner, or, still better, the bleeder, at the end of the gas main to displace the air in the furnace top, downcomer, dust catcher, and main.

IMPORTANCE OF DESIGN OF GAS OFFTAKE AND BLEEDER.

There should be mentioned the importance of the relation of design of gas offtake and bleeder to the probability of top explosions. If the gas offtake and bleeder pipes are arranged in the "castle type," then the gas from the furnace, when it is off, will rise up through the bleeder pipe and mix in the upper part of this pipe with any air coming up the downcomer. If the offtake has its center line at about the bottom of the big bell, with the bleeder pipe placed at the end, over the downcomer, then the air rising up in the downcomer has a certain opportunity to draft down the offtake and mix with the gas under the bell and hopper by diffusion.

EXPLOSIONS AT SHUTDOWNS FOR CHANGING TUYÈRES OR COOLING PLATES.

The practice in an ordinary shutdown for changing tuyères, or other similar or minor repair work, varies considerably, considering that no wide range of options is possible.

The practices that give the greatest probability of an explosion are those which allow the gas in the mains to mix with air during a brief shutdown of 10 minutes to one hour. As this gas is often high in hydrogen it may explode violently on ignition. The mixing of air and gas in the pipes may be caused in one or both of two ways: (1) Air is drawn into the main by diffusion into open burners or dust legs and through leaky burner seats by cooling or explosion doors when the gas pressure falls to less than atmospheric; and (2) by opening the top air is forced into the main by the atmospheric pressure at the openings below being greater than the pressure of the hot gas inside the main which is open at the top. When the furnace is started and this mixture is forced into a heated stove or boiler, an explosion results, usually only a slight one if the flow of gas from the burner is of sufficient velocity, but sometimes making a considerable noise or throwing the burner off the stand if the gas drifts from the burner slowly. Occasionally the explosion extends into the gas main and develops sufficient violence to split or burst the main. Virtually every blast-furnace plant provides against this contingency by installing so-called "explosion doors" on the gas main, which are supposed to prevent escape of gas at working pressure, but will be lifted by a gas pressure of one-half pound, or 13 inches of water or more. The use of such doors is of questionable benefit and is in disrepute with some furnace men because the explosion doors frequently permit a great deal of air to leak into the gas main during a shutdown, and, owing to their inertia, will not at times respond quickly enough to a gas explosion to save the apparatus. In one instance the explosion door and frame was blown off the main, evidently before the door had time to open.

It is easy to ascribe too much importance to explosions of gas in the mains or at the burners at tuyère stops (stops for changing tuyères or plates). Many plants are not troubled with them at all;

at others months may pass without a single "kick," and this period be followed by one in which nearly every stop is accompanied by one or more "cracks," or explosions, at the burner or in the main; in the latter case an explosion door may be forced open and a cloud of flue dust blown out.

DIFFERENT PRACTICES IN SHUTTING DOWN TO CHANGE BRONZE.

Although such explosions seldom occur, the fact that they do occur is the reason for explaining practices in detail, and a description of practices followed at various plants will illustrate the possible hazards.

FIRST METHOD.

Operators at furnaces having relief valves in the hot-blast main may, and occasionally do, change tuyères without drafting the gas back through a stove. In making such a stop all burners are closed upon opening the snort valve; the cold-blast valve and, preferably, the hot-blast valve of the stove on blast are also closed. The pressure being thus relieved on the hot-blast main, the gas-relief valve drops, affording a vent to the atmosphere through which any excessive pressure of gas in the furnace or gas mains is relieved. Owing to the generation of gas in the stock column there is always some gas under slight pressure left in the gas mains if the explosion doors and bleeders are tight, but at the same time the back pressure is usually sufficiently relieved at the tuyères to enable work to proceed. With such a method chance of an accident during a short stop is small. There is danger, however, in making an unexpectedly long stop, when the gas in the mains and dust catcher may cool sufficiently to permit air being drawn into the gas system through loosely fitting burners, dust bells, or explosion doors. One hundred cubic feet of gas at 500° F. and atmospheric pressure will contract to 60 cubic feet at 100° F., and if the contraction in volume is replaced with air the resulting mixture will be explosive and may be ignited by a spark or red-hot material of any description. A possible disadvantage is that the gas in the hot-blast main may be drawn back in some quantity into the stove. Frequently, in addition to the relief valve being opened, the gas is drafted back into a stove.

SECOND METHOD.

The furnace being on the last check, all burners are closed except the burner at the extreme end of the boiler-house gas main and stove gas main. The furnace is taken off, the stove burner shut, and the top bell on the furnace opened. Upon starting up the furnace, the bell or bells on the furnace top are left open until the gas has displaced the air from the furnace top and is issuing from the top in large quantities. Then the top bell is closed and gas forced through the mains to the open burners at the end of the boiler-house line. This method has nothing but custom to recommend it, as large quantities of air may be drawn into the gas mains and into the top of the furnace. Should the air displace the gas bodily, there would be no particular danger. However, such opportunity is afforded for mixing of air and gas, in drafting air through the mains when the furnace is stopped, and in displacing air from the mains with gas when the blast is turned on, that often sharp explosions, or "cracks" occur at the burners or furnace top when the air-gas mixture comes through; also red-hot material may ignite the mixture at some point inside the mains, dust catcher, or furnace top. There is seldom enough of this mixture to cause a violent explosion, but the frequent "cracks" at furnace top and at burners are sufficiently disquieting and demonstrate the possibilities of more serious explosions. There is at some plants a tendency to ignore apparently trivial dangers in handling gas and to regard minor "puffs" and "cracks" as incident to the method. Encouragement is afforded this attitude because certain practices which contain elements of danger may be followed with impunity for years, and some day the exact combination of gas mixture, confined space, and igniting medium required to cause an explosion is encountered, with the result that the plant is wrecked or damaged and men killed or injured.

THIRD METHOD.

A more common practice in changing tuyères is to close all the burners at the stoves and boilers, open the bleeder, and draft the gas back through a stove. Some furnace men keep the bleeder, bells, and burners shut, and draft back through the stove, using two stoves if the gas pressure is strong at the tuyères. Other furnace men open both of the furnace bells and the bleeder, and also the dust bell on the dust catcher, thus clearing the dust catcher and downcomers of gas, and may leave a stove or boiler burner open to draw the gas from the gas mains. Another practice is to simply open the bells to draw the gas away.

Opening the bells, or bells and bleeder, induces a draft of gas up the downcomer and air flows into the main at every crevice, with the result that after a prolonged stop the mains may have considerable air in them and the furnace men not know it, and upon starting up a sharp explosion at the burner may result. When relief from gas pressure is obtained by drafting back through a stove with the bells, bleeder, and burners closed, there may be a slight vacuum induced in the gas main sufficient to draw air in and mix it with the gas. In case a stop of unexpected duration is made and air is leaking in appreciable quantities into the gas main, the conditions in the gas mains and dust catcher, in drafting back through stoves with the burners closed, may be as dangerous as with them open.

In the latter case the sweep of air into the mains offers opportunity for mixing of air and gas only at the face of the gas and air volumes or in large spaces, like dust catchers, the explosive mixture is soon swept out, and only air follows. The mixture of gas and air may cause a "top shot" from time to time as it mixes with particularly hot gas coming from the stock, which usually is not of sufficient violence to do any damage. The danger lies in its being severe enough to blow gas out of the tuyère or other device about the bosh or jacket on which the men are working. With closed burners, the whole course of the mains and dust catcher may be filled with an explosive mixture as the result of air filtering in at several shortly separated points, and upon starting up the furnace and turning the gas into a stove the whole contents of the main may backfire. If these explosions are of sufficient violence to tear asunder riveted steel plate construction in mains, as they have been, it is useless to attempt to safeguard against them by providing explosion doors. Experience has proven that the doors do not always open in time to prevent the bursting of the apparatus.

USE OF STEAM IN GAS MAINS.

Inasmuch as possibilities of danger are presented with every variation and modification of method and use of apparatus in brief shutdowns, it is interesting to note one practice which is safe and in which few, if any, combinations of circumstances can introduce any possibility of danger. If steam is turned into the gas main at every shutdown of a longer duration than or differing in kind from a casting period, there can be no explosions of gas. The steam may be connected with the system at the dust catcher or at a point next to the water seal, goggle, or sand valve which separates the dry-cleaning apparatus from the rest of the gas-main system. The best way to make the connection is to have the steam inlet valve on the foundation level within reach or to have a hook hanging from the valve and within reach.

The use of steam prevents a vacuum in the gas system when drafting back through the stoves with the bells, bleeder, and burners closed, or when drafting through the top of the furnace with the burners open, because a blanket of steam is interposed between the gas and air; also any explosive mixture formed in drafting up the downcomer is saturated with steam and rendered inert. The use of steam may easily be made a regular part of the plant practice. The small cost and the brief time required for turning the steam in at shutdown are amply repaid by the assurance that there is no possibility of even small "cracks" at the burners or top, to say nothing of more serious explosions.

The use of steam in handling furnace gas was introduced and first given publicity in 1905 by J. W. Dougherty, who applied for and received a patent on its use. Some furnace men dispute his priority in this application of steam, but the writer has been unable to find any record of another claiming to have used it before this date. It is, however, claimed to have been used in gas flues of steel-works furnaces prior to 1905.

RELIEVING GAS PRESSURE IN THE MAINS.

In furnace plants consisting of two or more furnaces the gas pressure in the mains is always adequate, when only one furnace is off for short repairs, to prevent any air being drawn into the main. As a rule a number of boilers are taken off gas, and the stoves are taken off at the furnace on which the stop is necessary. The gas is then drafted back through a stove. Frequently on furnaces connected together there is a very high gas pressure at the tuyères, which is not relieved by drafting back through one, or even two, stoves. Where this pressure can not be relieved by the full use of available stove or boiler burners or bleeders on the other stacks, it is sometimes deemed necessary, in order to reduce the gas pressure so that the work can be done with the least danger to the furnace crew, to open the top by lowering the bells. If the gas pressure can not be sufficiently relieved by opening the bleeders at other furnaces so as to enable the bronze to be easily changed, the bells should be opened only in extreme need and closed at the first opportunity when the bronze is replaced. Opening the bells is in many cases a hazardous practice at best, and should not be resorted to when there are but two furnaces connected nor when, in the case of two or a larger number of furnaces, the furnace in question is located next to the main leading to the gas engine. In every case a man should be stationed to watch the gas on top, and in event of its becoming slack at the bells to order them closed.

The best method of all whenever a tuyère must be changed, whether at an isolated furnace or a group of furnaces, is to fill the water-seal valves at the dust catcher, flood the dust catcher with steam, draft back through a stove, and open the bleeder if gas is excessively rank at the tuyères. This procedure renders the possibility of even a small explosion remote or virtually impossible.

EXPLOSIONS DURING SHUTDOWNS AT CASTING TIME TO STOP TAPPING HOLE.

There is little or no excuse for explosions at casting time, when the blast is off, unless a gas-engine plant is pulling too hard on one or two furnaces.

At one isolated plant the following practice is followed, which may be taken as typical of safe practice: The furnace is provided with a gas pressure gage and also with a whistle for signaling the boiler house. As the furnace is checked, the stoves are taken off gas until there remains but one stove on the furnace at the time the last check is blown, all of the boilers being left on gas. Shortly before it is necessary, in the blower's judgment, to open the snort valve and take the furnace off, he signals the boiler house. The water tender or foreman thereupon shuts the gas off at all but two boilers at the end of the main farthest from the furnace. One of these is taken off in case the gas becomes weak, the necessity for this being judged by the appearance of the gas flame at the burners. There is always sufficient gas from the furnace to keep a good flame at one boiler. When the snort valve is opened, the stove tender takes the last stove off gas. When the blast is put back on the furnace he turns the gas into this stove as soon as the snorter is closed, and then turns on the remaining stoves in quick succession. In putting the boilers on gas, the blower signals the engine room for full blast, then signals the boiler-house crew to put the boilers back on gas, waits a half minute until the snort valve begins to roar, and then turns the blast into the furnace. The boilers are put on singly and in turn, not all at once, but without further instructions.

Where two furnaces comprise the plant, the stove burners are usually shut at the furnace being taken off and the boiler-house crew either permitted to use gas with no burners closed or signalled to slack off slightly. With three or more furnaces in blast on a common main, it is not usually necessary to shut off any of the boiler burners, unless the gas supply should be scant on account of a furnace sticking. Heavy demands on stoves or boilers result in one or the other being favored at such times, but this is aside from any consideration of possible "top shots" or "kicks." At shutdown for casting it is essential as a rule to close the bleeder on the furnace being taken off.

At the few plants where the engines are stopped, or where the butterfly valve is closed tightly in the cold-blast main, or where a relief valve is opened at casting, little or no air enters the furnace, and on that account very little gas is generated. In such a case it is necessary to choke back the gas burners to insure positive gas pressure inside the gas main, which is essential to safety.

ACCIDENTAL STOPPAGE OF GAS.

Where water seals are placed in the gas mains, care should be taken that they do not fill with water from a leaky water-inlet valve. Such a possibility is done away with by placing an open gate valve at a tee in the feed line next to the dust catcher. Occasionally, where there are water seals on the discharge end of scrubbers not cleaning the gas thoroughly or dry, the flow of gas may be stopped by the water seal filling with sludge or with moisture condensed from the gas. Such valves should be emptied or flushed out at the beginning of each shift.

CARE AND OPERATION OF ROTARY GAS WASHERS.

The addition of a gas-cleaning plant which includes some type of rotary washer introduces cause for special precautions. Such washers may create a vacuum on the suction side, and if, for some such cause as sticking and tightness of the furnace, or a furnace stop, the supply of gas is less than the washer's running capacity, air will be drawn into the supply main at any point where a perfect seal is not present, such as bells or explosion doors or open burners. If this air mixes with the gas, still being furnished in scanty amount, to make an explosive mixture, and this is sent over to the stoves or boilers where flame is present, the flame may flash back and explode the gaseous mixture in the mains. Most frequently, however, a rotary washer draws its gas from a main carrying partly cleaned gas (this main also supplying partly cleaned gas to the stoves or boilers, or both) and delivers the clean gas to gas engines. In this case, if the gas supply falls below the capacity of the washer, and the washer remains running, it will create a vacuum in the main which supplies gas to the washer and to the stoves or boilers. As this main is open to the air at its stove or boiler burner connections, air is drawn into the main and delivered through the gas-cleaning plant to the gas engines where, if back-firing occurs and the incoming charge is ignited, the gaseous air mixture filling the mains will be exploded.

TYPICAL ACCIDENTS.

An example of such an explosion is illustrated in the following description of an accident that occurred in 1909:

Two furnaces were in blast at the time of the accident. At 12.31 a. m. the blast was taken off the furnace nearest the Theisen scrubber, and as the other furnace did not furnish enough gas for the gas engine and boiler and stove burners, air was drawn into the gas-cleaning apparatus through some of the burners under the suctions of the Theisen apparatus. At 1 a. m. the ammeter at the gas engine began to fluctuate. The current gradually went down from 4,000 to 2,200 amperes at 1.30 a. m. It then went up again and reached nearly 5,000 amperes at 2.19 a. m., when an explosion took place in the gas main between engine and gas holder. The gas engine stopped. Three minutes later, at 2.22 a. m., a second explosion was heard which blew up the gas holder. Seven minutes later a third explosion occurred which wrecked the tower washers and dust catcher on the furnace side of the Theisen. The first two explosions were caused by flame blowing back through an inlet valve of the gas engine and igniting at the engine the mixture in the main leading from the engine to the gas holder. The ignition evidently traveled slowly in this main, as three minutes were required after the engine stopped at backfire until the gas holder blew up. Probably the ignition of gas in the main had attained a great velocity as it reached the gas holder and projected the flame with great speed into it axially, thus igniting a large quantity of gas-air mixture at once. The bell of the gas holder was blown up and landed on its side in the holder tank, and a part of the top of the bell was blown off and thrown several hundred feet. The third explosion was started at one of the burners of the stoves or boilers, the gaseous mixture formed at the nose of the burner by percolation of air into the stationary gas igniting and traveling back to the dust catcher near the tower washer. The roof of the dust catcher was blown off and the main from the dust catcher to the tower washer was lifted by the blown-off roof so that it was wrenched loose from the tower washer. The top of the tower washer was ripped open but not blown off in one case, and torn off completely in another case. The explosion extended to the inlet of the Theisen washer where it was evidently extinguished by the water.

In 1910 there occurred in Germany a somewhat similar explosion. A description by Dressler[a] is summarized below:

At the time of the explosion but one furnace was running. The gas from the furnace went through a dry-dust catcher and two tower washers, and the final cleaning took place either in a Theisen or in two fans. At the time of the explosion the two fans were running in series, the cleaned gas going in part to stoves and boilers and in part to gas blowers and a gas generator. There was no gas holder. The furnace was running regularly with the bell closed. This bell was a "single" bell with central tube gas offtake, the space between the bell and the tube being sealed with a water seal. The top fillers noticed that the water seal was suddenly emptied and immediately refilled it. This probably took at least a minute. Following this, the gas blowers stopped and the gas generator began to run more slowly. As soon as the gas blowers were seen to be stopping the steam blowers were set in motion, but immediately upon the stopping of the gas blowers a violent explosion took place in the clean-gas main between the engines and the gas-cleaning house, traveling as far back as the Theisen, which was not working but was connected with the clean-gas main. The explosion was so violent that the front of the Theisen was destroyed and hurled a distance of several feet. The explosion did not make itself felt in the gas main leading to the furnace, nor to the boilers and stoves, nor at the fans. Accordingly, it is evident that only a comparatively small amount of explosive gas was present in the clean-gas main and that the scrubbers as well as the uncleaned gas pipe was filled again with the normal furnace gas, the aspirated air and gas at the furnace top having pushed itself like a cushion in front of the following normal gas, so that the inflammable zone had passed the gas-cleaning station and filled the clean-gas main from the Theisen washer to the gas engine. That it did not pass into the main to the boilers and stoves can be explained only by assuming that there was a balancing of pressure between the chimney stacks and the aspirating of the gas engine.

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[a] Dressler, H., Eine unerklärte Horhofen Explosion: Stahl und Eisen, Jahrg. 31, 1911, p. 270.

The explosion in this case, as in the preceding example, was due to back-firing or premature ignition in the gas inlet of the gas-engine generator. Whenever the normal working of a four-cycle gas engine is disturbed, either by an excessively high hydrogen content of the gas or a sudden lowering of thermal value, the ignition recoils into the mixing chamber, as the inlet valve, and also the gas valve, is open during suction. When back-firing takes place under normal working conditions, the line being filled with gas only, and the flame shoots into the gas pipe, such premature ignition has no serious consequences. Should the gas piping be filled with a gas-air mixture, then the ignition obviously travels back into the gas main where it may develop into an explosion.

ESSENTIAL PRECAUTIONS IN HANDLING GAS.

These accidents show that one of the most important details in the supervision of gas cleaning is to prevent, by suitable regulation, the formation of a vacuum in the gas mains at any point between the furnace and the rotary scrubber, as the rotary scrubbers of the type used in this country can usually give a vacuum, on the suction side, of 3 or 4 inches of water, and a discharge pressure about 8 inches higher than the pressure on the suction side. This requirement is adequately met at all gas-engine and gas-cleaning plants by means varying from very elaborate to exceedingly simple systems. These consist of a series of gages indicating and recording the pressure in the gas mains, automatic alarm bells which ring when the gas pressure falls below a certain predetermined danger point, whistle signals, telephones, circuit breakers on the gas engines that break automatically or are tripped from the gas-cleaning house, and valves that close or open automatically in the gas mains with varying pressure, but chiefly dependence is placed in personal supervision by a responsible employee at the cleaning house.

The point of danger in the gas-main system is essentially the main supplying gas to the rotary washer. If the pressure in this main is never permitted to drop below a certain water pressure—one-half inch, for instance—it follows that gas will always be delivered to the engines in sufficient quantity and at a satisfactory pressure, and that as there is a positive pressure in the mains leading to and from the washer, air can not be drawn in. Although the supply main is the one watched with the greatest vigilance, gages are attached to other parts of the gas system. Examples of various systems follow. The placing of the gages is given in the order of their importance:

1. Gas (*a*) to boilers and primary tower washer; (*b*) from primary washer to stoves and secondary washer; (*c*) from secondary washer to gas engines. Gages placed at *b*, *a*, and *c*.

2. Gas (*a*) to primary tower washer and boiler; (*b*) from primary washer to fan; (*c*) from fan to stoves and Theisen washer; (*d*) from Theisen to gas engine. Gages at *c*, *a*, *d*, and *b*.

3. Gas (*a*) to primary tower washer; (*b*) from primary tower washer to stoves, boilers, and Theisen; (*c*) from Theisen to gas engine. Gages at *b*, *a*, and *c*.

4. Gas (*a*) to boilers, stoves, and tower washer; (*b*) from tower washer to rotary cleaner; (*c*) from rotary cleaner to engine. Gages at *a*, *c*, and *b*.

METHOD OF HANDLING GAS AT A LARGE PLANT.

A detailed report of the system of handling gas at a large plant is given here in abridged form, as it conveys an exceptionally good idea of the methods used and conditions encountered.

GAS CONSUMPTION AT FURNACE PLANT.

The blast-furnace plant consists of one 500-ton and three 450-ton furnaces, each having four hot-blast stoves. In addition to these stoves the furnaces supply gas to the following:

8 batteries of Stirling boilers	horsepower	4,800
4 batteries of Cahill boilers	do	4,000
4 gas electric engines	do	7,200
1 gas electric engine	do	4,000
2 ladle dry houses.		

DESCRIPTION OF GAS-CLEANING PLANT.

The gas-cleaning plant consists of two parts—the preliminary cleaners and the purifying plant.

The preliminary cleaners are situated at the furnace and clean all the gas. The cleaners at each of the four furnaces differ in many respects as to form and arrangement, but they are nearly alike in principle.

At furnace "A" the gas leaves through the downcomers and passes through a 30-foot common dry dust catcher, a 21-foot tangential dry dust catcher, and a 26-foot 3-inch Mullin or impinging washer. The gas leaves the Mullin and passes through a Crawford valve and then into the 6-foot common furnace main. Off the end of this main a 30-inch main leads to the ladle dry house. This common 6-foot main passes through the Stirling boiler house, and from this main a 6-foot main leads to the Cahill boilers.

At furnace "B" the arrangement is identical to that at furnace "A," except that there is one 5-foot main leaving the Mullin washer, which leads into the 5-foot main behind furnaces "B" and "C."

At furnace "C" the gas passes through one 30-foot, common dust catcher, one 26-foot tangential dust catcher, and one primary and one secondary "baffle washer." The gas leaving the washer passes into the upper 5-foot main behind furnaces "B" and "C." This main feeds gas to the boilers and to the primary 6-foot main connecting the gas mains from all the furnaces. The stoves of both furnaces "B" and "C" receive gas from this 6-foot main.

At furnace "D" the gas coming down from the top is conducted through four downcomers into two pipes leading to a common dry dust catcher 30 feet in diameter, thence through a series of three centrifugal dry dust catchers, and from there through a primary and a secondary washer identical to those at furnace "C." The gas leaving the washers is passed into the upper 5-foot main, which feeds the "D" furnace stoves and also connects with the common 6-foot main connecting all the furnaces.

Goggle and Crawford valves are so situated in the gas mains that any furnace, boiler house, or engine house can be shut down or disconnected without interfering with the operation of any of the others.

A 5-foot gas main connecting with the "D" furnace end of the common 6-foot main leads to the purifying plant. Between No. 8 and No. 9 stoves a 5-foot main connects with the common 5-foot main and leads along the trestle to the purifying plant, thus forming two gas supply mains to the purifying plant.

The purifying plant consists of three separate units in parallel. Two of these are identical, each having one baffle, one Zschockke and one Theisen washer in series. The third consists of only one Theisen washer.

From the Theisens the gas passes through a wafer separator into a common 5-foot main which leads to the gas holder. A regulating butterfly valve is placed in the main ahead of the gas holder. A 5-foot main leaves the gas holder and divides into two 5-foot mains, one leading to the gas electric engines, the other to the gas blowing engines.

The gas holder is 55 feet in diameter and has a lift of 14 feet, thus having a capacity of 33,000 cubic feet, and requires a pressure of 7¼ inches of water to raise the bell. A 5-foot by-pass has been connected to the mains entering and leaving the holder so that in case of emergency the holder can be cut out by turning three goggle valves.

A butterfly valve has been placed in each of the engine-house gas mains, which is regulated by a small gas meter 5 feet in diameter connected to the gas main by a 4-inch pipe. With this gas meter the pressure of gas can be regulated at the engines. At present this pressure is set at 3½ to 5 inches of water.

As the pressure in our gas mains at "B," "C," and "D" furnaces is much larger than at "A," owing to the smaller main, it is obvious that as we have all of our boilers between these conditions, and a ladle dry house on the opposite side of "A," gas very seldom reaches the gas engines from "A" furnace, the exception being in the case of a shutdown or furnace out of blast at "B," "C," and "D." The method of operation under these conditions is referred to later.

SAFETY OF THE PLANT.[a]

The only danger which exists for the safety of the installation has its source in the lack of sufficient gas supply.

The principal part of the gas-cleaning plant is the so-called purifying plant, the object of which is to refine the gas for use in the gas engines. As mentioned heretofore, this part of the plant contains the Theisen washers, and these are so constructed that there is a suction fan on the end of the gas supply and a discharge fan on the other end. Because of this construction, they are able to suck gas or air through the mains and discharge it under pressure. The tendency of the washers is to deliver gas to the gas holder at a pressure 8 to 12 inches higher than the pressure on the suction side, and the object of the gas holder is to act as a reservoir, and it maintains a constant pressure of about 7½ inches.

It is evident that if for any reason the gas supply to the holder should fail or continue to be pumped in after the holder has reached its uppermost position, there would be danger of a wreck; in the first case due to the suction from the engines on the empty holder, and in the second, to high internal pressure, as the butterfly valve is not gas-tight.

In mentioning the gas supply to the holder, we will have to consider air, because should the Theisen washer continue to operate when the gas had failed, air would be pumped into the main by way of the stove and boiler burners, bleeder and any similar openings in the supply main. If such were the case there would be no danger of a collapse, but an infinitely more serious one would be presented and in the nature of an explosion. The engines would receive a mixture of gas and air, and this change of mixture would cause the engines to "back-fire" into the main, and as the explosive mixture might extend all of the way back through the main, an explosion would be possible, wrecking the entire installation.

FURNACE OPERATION AS REGARDS GAS SUPPLY.

As the requirements of our gas-engine power house are practically the same regardless of the number of furnaces in blast, it is evident that there are times when it becomes very much of a problem as to how we can keep the engines running and take care of a brief "shutdown," owing to accidents at the furnaces, such as the burning of a blow pipe, or to make necessary changes. When the latter is the case we try to take the blast off at meal hours, when the load is the lightest and when we can shut down all but one gas engine and all of the engines at the steam power house. We then take care of the plant load by running the motor-generator set from the gas engine, and thereby economize on the gas consumption.

The worst condition which we can have and attempt to satisfy all of the requirements is that of only two furnaces in blast, and any rules of methods of operation are based on that condition. In case a shutdown is to occur, the general foreman or blower, if it be an accident, notifies by means of the telephone the office and Theisen washer operator. He then sends word to the boiler tender to make a fire, and to the stove tenders to be ready to operate the water-seal valves in the gas main to the furnace in question. If all furnaces are in blast, he also communicates with the men in charge so that everyone in control of the gas supply can assist in maintaining an excess.

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[a] All pressures mentioned are in inches of water.

If two furnaces only are in blast, he pulls back the stove burners on both furnaces before the blast is entirely off, and also opens the bleeder on top of the furnace so that the water-seal valve can be entirely filled while there is still a small supply of gas.

Having complied with these methods of operation, the furnace man has not eliminated the dangers, but he has done all in his power to put the supply on a safe basis, and, as is explained later, the responsibility has been passed on to others and he is free to hurry his various operations on the furnace. In the case of a prèmeditated shutdown, everything can be carried on deliberately, and possibly in the case of an accident, but if not, the dangers are taken care of partly by automatic means and partly by operators.

The furnace man has a pressure gage on the trough supplying the cooling water to the furnace and he maintains a pressure such that it will be lower than that of the blast, thereby reducing the possibility of hydrogen in the gas supply to the engines. He is also provided with a set of instructions and signals such as are applicable to the other parts of the system.

GENERAL SAFETY PRECAUTIONS.

The general features which have been installed for the purpose of avoiding accidents are as follows:

(A) A plant telephone system which connects the gas-electric engine house, blowing-engine houses, steam-power house, Theisen washers, furnaces, and offices, a telephone connecting the gas-engine power house and the steel works, also a telephone on the outside exchange. Where there is danger of operators not hearing the bell, a steam whistle is installed.

(B) Steam whistles electrically operated from the Theisen washers are located at each house and one between "C" and "D" furnaces. They are used in the case of "low gas" and can be heard at all of the furnaces and boilers, and, in fact, very distinctly from any part of the plant. They have a distinct tone of their own and can not be easily mistaken for others, so that they serve to keep posted all who are concerned with existing conditions.

In order to avoid the danger previously mentioned of pumping air or gas into the gas holder after it has reached its uppermost position, a butterfly valve is arranged in the main between the Theisen washers and the holder, and is operated by the holder bell in such a manner that when the bill is in its highest position or full, the valve shuts off the gas supply. The valve then is opened automatically as the bell descends and maintains a position which varies for each position of the bell.

In the case of an accident at this part of the plant, it is understood that a long blast on the "low-gas" whistle, located at the boiler houses and furnaces, will serve to notify the entire works.

Whistles are installed at the washers and at the gas-engine power house and serve as a means of communication in addition to the telephones. They are used in starting up and in stopping the engines and washers. The code of signals as posted and used is as follows:

SIGNALS AND GAGES FOR THEISEN WASHER.

To gas blowers from Theisen washer.—One blast, "Gas supply failing"; two blasts, "Gas supply O. K." (test at 6 a. m.).

To gas-electric engines from Theisen washer.—One blast, "Shut down engines"; two blasts, "Gas supply O. K."

To Theisen washer from gas-electric engines.—One blast, "Shut off gas supply"; two blasts, "Send over gas."

General signals from Theisen washer.—One blast (long or short), "Trouble, help wanted"; one long and two short blasts, "Pull stove burners back"; two blasts, "Pull boiler burners back"; three blasts, "Light boiler burners"; four blasts, "Gas supply O. K." (test 6 p. m.).

General signals from gas-blowing engines.—Any number over four blasts, "Gas engine failing"; four blasts, "Gas supply O. K." (test at 6 a. m.).

This code, in addition to being posted in a frame at the washer, is also posted at the gas-engine house, furnaces, boiler houses, and office.

The operators at the gas-blowing engine house upon receiving one blast from the Theisen will immediately put themselves in position to familiarize themselves with the gas situation, and, if they find that there is not enough supply to keep the engines running, they will advise the furnaces by blowing more than four blasts. When it is found that the engines must be shut down, the ignition will be cut off as quickly as possible.

At the furnace the stove tenders must be acquainted with the meaning of the whistles so that they will not wait for orders, but will pull their stove burners back immediately and make all preparation to take the furnace off as soon as possible if found necessary. The blowers upon hearing the whistle will get to their pressure gages as soon as possible and from them determine the course to be pursued. If the gage pressure does not indicate that the blast has failed entirely, it of course will mean that the condition has recovered to such an extent that the engines were not forced down, or that probably the blast is being supplied by speeding up the split steam engine that happens to be in operation.

At the steam-blowing engine house the operators will be prepared as soon as possible to put a steam engine into service on the furnace where the gas engine has failed. When they are ready the furnace can be advised by means of the telephone, in which case they will prepare to put the blast on again.

In the case of the stove burners, it is quite probable that if five or more whistles are blown "low-gas" signals will have been blown previously and that burners will already be back. The case may arise, however, where this may not be so, and it will be necessary that special precautions are taken in advising the men in charge of the stoves.

In case a furnace is shut down for any purpose, the furnace foreman will have an understanding with the stove tender that after the regular operations have been performed in taking the furnace off, he will also see that steam has been turned on, and not only turned on but **actually going into the dust catchers.** He will also take charge of the Crawford valve, shutting off the furnace in question, and see that it is operating in perfect condition. The stove tender is to do these things, and nothing else, while the furnace is off blast.

The following notices are placed in conspicuous places, the first being inside and the second on the outside, near the door of the clearing plant.

(1) "Smoking and the use of matches or open flames of any kind is strictly prohibited in this building."

(2) "Keep out except on business. Do not smoke or use red-hot metal or flames of any kind on or near these washers. Employees failing to comply with the above will subject themselves to discharge."

Similar signs are placed in and about all other parts of the gas-engine system and building.

It has been shown that the failure of gas supply is the most common danger which exists; therefore it is essential that some means be devised whereby this danger is guarded against. As has been shown, the furnace man does all in his power in the matter of warning, and the responsibility for the safety of the installation devolves upon the Theisen-washer operator, as he is in control of the train of mains and piping which conducts the gas to the engines. In order to assist him and to serve as a means of warning in case the gas gets "low" for any unforeseen reason or at casting time, a system of manometer tubes and recording pressure gages are installed.

The low-pressure signal consists of a manometer tube made of steel pipe and of sufficient length to hold the water for any pressure which may exist on the furnace side of the Theisen.

A copper float supporting a brass rod, on the end of which is a copper washer, is in the atmospheric end of the tube. The washer is set at such a point that when the pressure has been reduced to 2 inches, it will complete an electrical circuit and cause a whistle to be blown. After thus receiving his warning, the operator watches his glass tube and gage for critical conditions, as they are connected in the same line with the steel tube and on the supply side of the Theisen.

If the pressure falls below 2 inches, the whistle calling for the stove burners to be pulled back is blown, and the furnace man quickly responds. If this does not increase the supply to the engines, the signal for the boilers is blown. If the pressure does not resolve into a vacuum before the supply is increased, by pulling back the burners or the unusual conditions of supply readjusted, the operation is not changed. But should a vacuum occur, a gate valve which is placed between the holder and the Theisen washer is closed and the Theisen shut down in order to prevent air from being pumped into the mains and holders. The operator then watches his tank, also for a return of his supply pressure, and in case the tank gets beyond a marked position before the pressure returns, he signals the engine house to shut down, and his orders are imperative in so far as the engineers are concerned. In case it becomes necessary to stop the gas-power engines quickly, due to the sudden failure of gas, the Theisen-washer man has control of the ignition system on engines, which he immediately cuts out. The operator can of course vary his operation to suit conditions, such as allowing the tank to go lower when only one engine is pulling on the reserve gas supply.

One of the other glass tubes and the remaining pressure gage are connected to the main on the holder side of the butterfly valve, and indicate the pressure carried there. Under normal operation it is obvious that the pen of the gage should not vary from one position. If it does, it shows improper working of the butterfly valve or a shutdown at the engines and gas supply cut off. This recording gage has since been placed in the gas-blowing engine house.

The remaining glass tube indicates the pressure between the butterfly valve and Theisen washer. It has little or no effect on operating conditions, but shows the pressure which is built up by the Theisen.

The following list of duties is framed and hung up in the Theisen-washer building:

1. Change chart on 24-hours' pressure at 6 p. m.
2. Test automatic "low-gas" signal each day while 12-hour chart is off and when necessary to fill with water so that whistle blows at 2-inch gas pressure.
3. Test signal whistle at gas-engine house every 12 hours and boiler house at 6 p. m.
4. Drain water from gas main at tunnel and downlegs of by-pass valves each 12 hours.

5. Clean Crawford valve each 12 hours.

6. On closing down washers for any length of time, drain all water pipes, etc., to prevent freezing.

7. When starting washer after a long shutdown, let gas pass through system out of bleeder for 15 minutes before allowing engines to use it.

8. Do not give O. K. signal to start gas engines until engineer in power house has called up on telephone to say that bleeders are closed.

9. When we have "low gas" at gas washer and the electric engines are to be shut down, they are to be shut down by pulling out ignition switch.

Considering the system as a whole, changes will doubtless be made from time to time as their necessity becomes apparent, in order to decrease the possibilities of accidents, but as it now stands, those concerned in the control and maintenance of the gas supply appear to work harmoniously and to handle the "low-gas proposition" satisfactorily and intelligently.

METHOD USED AT ANOTHER PLANT.

Another method is described in brief by Freyn,[a] in the Transactions of the American Society of Mechanical Engineers, as follows:

The power house, gas-washing plant, and blast-furnace office were connected by two independent telephone lines, and recording instruments, in addition to ordinary U-tubes, were installed in the washer building and at the blast-furnace office, so that not only may the gas pressure be observed at any time but it is automatically recorded for each period of 24 hours. Moreover, an automatic alarm was installed at the blast-furnace office, which rings a bell as soon as the gas pressure in the raw gas descends below a certain danger point, and whistle signals operated by solenoids from the blast-furnace office were provided in the boiler house to inform the head fireman of the number of boilers to be "taken off" or put on gas. In addition an automatic bell was placed in this boiler house, calling the operators' attention to any drop below normal in the gas pressure.

Independently of the blast-furnace department, the gas-cleaning-plant operators were also carefully watching the gas pressure. The position of the gas-holder bell was made visible at any time by a system of incandescent lamps in the washer house, and strict orders regarding the use of the gas were issued by the blast-furnace superintendent, instructing the men to favor the gas engines under any circumstances, as it was fully recognized that, having taken care of the requirements of the hot-blast stoves, the remaining gas could not possibly be more efficiently utilized than in the gas engines. The practice was to shut off the gas immediately at a certain number of gas-fired boilers, as soon as the pressure in the overhead gas flue dropped below a predetermined point. Additional boilers were taken off if the gas pressure did not recover, so that sometimes as many as 24 boilers were being fired by coal exclusively. If this did not have the desired result, stoves were taken off for short periods to increase the gas pressure above the danger point. At last, if all the steps did not improve the situation, one or more gas engines were shut down. Fortunately in the majority of cases the blast-furnace operators know in advance if the gas supply is likely to fail, and communication could easily be established to warn the departments concerned of the impending gas shortage.

The system of close observation and of cooperation among the departments concerned worked to perfection, but nevertheless conditions existed at times which with all due optimism had to be called dangerous. It was frequently necessary to keep several gas engines running, with the gas pressure dropping below the danger point momentarily or even for periods of a few minutes. This was unavoidable if the operation of certain departments dependent upon a supply of electric power was to be maintained with any regularity. If the gas engines had been shut down every time a momentary drop in pressure occurred, it would often have meant an endless amount of shutting down and starting of engines, altogether too frequent for satisfactory operation of the mills and physically impossible for the gas-engine operators.

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[a] Freyn, H. J., Operating experiences with a blast-furnace power plant: Trans. Am Soc. Mech. Eng., vol. 32, 1910, pp. 368.

AUTOMATIC APPLIANCES TO REGULATE THE FLOW OF GAS.

The exclusive use of automatic appliances to regulate gas is unsatisfactory for several reasons. Such devices are never fool proof, and from various causes may work at unexpected times or may fail to work at a critical moment when dependence has been placed on them. Among the various types in use are automatic circuit breakers that stop the washer and at the same time break the ignition circuit on engines, butterfly valves on the mains that automatically open and close with varying gas pressure, and check valves to prevent back flow in downlegs to gas burners or in gas mains. The employment of a responsible man at each shift who through experience knows whether any pressure drop in the mains will be only incidental and brief, being caused, for instance, by casting, or from the characteristic fluctuations of the gas flow from the furnaces, is much safer and incomparably more satisfactory than depending entirely on automatic safety appliances. In case of occasional or somewhat frequent drops in the gas pressure the working of these safety appliances might shut down the engines so frequently that it would be physically impossible for the engine crew to keep them running; also, the automatic closing of butterfly valves might cause very serious accidents at the stoves or boilers when the valves again opened, if a rush of gas should arrive at the burners before sufficient time had elapsed for the men to assume control of putting the gas on again. Similarly, strict operation of the boilers, according to signals automatically operated by gas pressure, would be impossible under many conditions of furnace operation. Such signals are chiefly useful as warnings and indicating the need of watchfulness. Placing control of gas at the burners under a responsible person who can judge whether shutdowns are necessary is in the end most satisfactory, as stove tenders and the boiler-house crew will follow instructions and recognize the necessity of compliance with signals coming from a competent gage man, whereas repeated and annoying signals and interruptions from automatic valves, bells, or whistles will be frequently disregarded, only to result disastrously some day when compliance with the signal is essential to avoid accident.

Where the cleaning plant consists of tower or impinging washers alone, no rotary cleaner or fan being used, the putting on and taking off of boilers and stoves is usually controlled by the furnace blower, who can be provided with a pressure gage and signal whistle to the boiler house at a place convenient to the engine-room whistle switch and snort-valve lever or wheel. The control of gas with this type of washer is not greatly different from control of gas direct to stoves and boilers from the dust catcher.

BLOWING IN.

CAUSES OF EXPLOSIONS AT BLOWING IN.

There have been several explosions of gas at blast-furnace plants on blowing in. The critical point in this operation lies in the necessity of bringing the gases down from the furnace top to the gas burners. This operation is made comparatively simple at plants having one or more furnaces already in operation, as gas from the mains can be sent up through the dust catcher and downcomer to the furnace top, displacing the air, and when a good flow of gas is coming off at the top bleeders the bleeders may be closed, forcing the gas from the furnace just blown in down into the mains.

SPONTANEOUS EXPLOSION OF GAS MIXTURES.

At isolated furnaces, however, the problem is not as simple and the work of bringing the gas down causes much anxiety, owing to the dread of an explosion. Some furnace men believe that furnace gas and air will ignite or explode spontaneously at such times, that is, explode in the direct absence of fire or other igniting medium. Three explosions at blowing in occurring within recent years have already been mentioned, in which the circumstances and the testimony of witnesses pointed to spontaneous combustion as the cause. It is realized that persons at fault in these cases may have omitted to mention to their superintendent contributing causes, but the writer, at least, regardless of whether spontaneous or catalytic ignition of gas mixtures can or can not take place, is unable to offer any explanation other than spontaneous ignition.

In two of the accidents mentioned the gas upon being brought down into the dust catcher exploded within it, in one case breaking the dust-bell lever and in the other doing no damage. In the latter case it is stated a bleeder at the end of the gas main at the boiler was open, no burners were open, there was no fire near the mains, and gas had not appeared at the gas-line bleeder. The explosion occurred within a minute of closing the bleeders on top and opening the bleeder at the end of the boiler gas mains. The other case occurred in almost exactly the same manner, except that steam had been turned into the dust catcher at the instant of bringing the gas down.

The third explosion mentioned resulted in splitting the furnace jacket above the stock line. The gas had just been turned into the boilers and the men were about to turn it into a stove.

As a rule, the men who believe such explosions result spontaneously belong to the older school of furnace men, with a few converts to the theory among the younger group, conversion having been effected by experience or direct testimony. One furnace man who had an unexplainable gas explosion of this type years ago now follows the practice of blowing a considerable volume of cold air through the furnace after it is filled and before it is lighted in order to blow out any loose dust and fine ore which otherwise would presumably be blown out when the furnace is lighted and the blast put on. He attributes this type of explosion to the presence of these fine dusts. This is cited as an illustration of the very real belief in these explosions being caused by spontaneous combustion.

BRINGING GAS DOWN WITH BELLS OPEN.

A second cause of such explosions is found in taking the gas down to the burner and having the bells on top open, either inadvertently or by design. Two accidents have been noted which occurred in this manner, one resulting in blowing up the dust catcher and the other in bursting a gas main. In these cases the draft in the stove-chimney stack was sufficient to draw air in about the bell, where it mixed with gas, and the mixture, on arriving at the burners, back-fired and exploded.

BRINGING GAS DIRECT TO STOVES OR BOILERS CONTAINING A FIRE.

A third cause is in taking the gas down from the furnace top directly to boilers or stoves in which there is a fire burning. A somewhat recent explosion resulted in this manner. In this instance steam was turned into the dust catcher, the bells and bleeder on top shut simultaneously, a boiler burner opened over a lively coal fire on the grates, and the gas led in. The burner back-fired and the resulting explosion blew up a small auxiliary dust catcher about 100 feet from the burner. Repairs were effected, the blast was again put in the furnace, the same procedure followed, and the dust catcher was again blown up. The third time steam was led into the dust catcher several minutes before gas was brought down, and when steam appeared at the bleeder the gas was brought down to the burner without difficulty. It is to be emphasized that the use of steam in itself may not prevent accident, because, at 48° C., gas saturated with water vapor or steam (not entrained moisture) is at a maximum of explosibility. Steam should rather be used with the idea of interposing a blanket or cushion of steam between the air and the gas. An accident resulting from the same practice—that is, leading gas directly into a boiler having a fire in it—occurred several years ago, with disastrous consequences, gas mains and dust catchers being blown to pieces and a spectator killed.

This practice is followed quite widely, and rests on the assumption that the gas sweeps the air bodily out of the mains, and that there is no mingling of air and gas except at the face of the gas and air. It is probable that there is no sharp line of demarkation between the gas and air currents, and that at times the gas pushes ahead, owing to its higher temperature and tendency to travel along the top of the gas main, while the cold air, being of heavier density, tends to travel along the bottom, to say nothing of the opportunity afforded for mixing of air and gas in the dust catcher. Obviously, the greater the velocity of the gas current at this time the more completely is the air swept out. For this reason it is the practice at some plants to turn the blast at nearly full pressure into the furnace just before and while the gas is being brought down, in order to get a good sweep of gas. Such a method, however, does not offer the best possibility of immunity from explosion, and it is better to allow the gas to escape from the far end of the gas main through a bleeder, explosion door, or manhole door, or cold boiler, until gas is coming in good volume rather than immediately leading it over a fire.

BUILDING FIRES IN GAS MAINS.

A method of taking the gas down followed at a few plants where the gas is conveyed to boilers and stoves by an underground gas main is to build a large wood fire in this main, under the manhole nearest the gas downtake to the underground main. This fire is left burning strongly and the gas is brought down into the main, over the fire, and thence to the burners. At one plant where this practice is followed there is a belief that cold damp air in the main when mixed with gas will explode spontaneously. The essential idea of the fire is probably not to heat the main, but to push up a current of inert gases, CO_2 and N_2, into the mains from the underground main to the furnace top, so that inflammable furnace gas, when brought down, will arrive at the fire unmixed with air. It is observed that the gas extinguishes the fire, and a sheet of flame sweeps up the main ahead of the gas, burning out the air in the main, until it finally arrives at the burners. In one instance a violent explosion, probably caused from an inadequate fire or too much air being admitted to the fire and mixing with the gas, occurred and blew up the underground main along half its length. A variation of this method is to build up a large bed of burning coals, then close all manholes and wait about 30 minutes before bringing the gas down. Both of these methods are too hazardous to be undertaken by any except men having previous experience with the details of time intervals and practice.

SHUTTING OFF THE BLAST.

An infrequent source of explosions is when, shortly after the gas has been brought down and is at the burners, the blast has to be taken off in consequence of some mishap. If more than one furnace is on the gas line no danger is introduced, if care is taken to keep the gas pressure up, but if only one furnace is in operation an explosion may occur unless stringent precautions are taken to prevent air entering the main. In one instance where a furnace was taken off suddenly about two hours after the gas had been turned into the boilers and stoves and while the men were on the bustle pipe repairing the water supply, the dust catcher exploded, throwing them from the pipe to the cast-house floor. It is not known how the explosion was started as the gas temperatures were low, but as one burner was left open and the furnace bell was opened, the mixing of air and gas is apparent, the point being that air should not at such times be admitted to the mains or furnace top. The blast should be kept on at all cost and a positive gas pressure maintained.

METHODS OF BLOWING IN PRACTICED AT 18 PLANTS.

The point has been expressed by several furnace men that, in view of the variation in methods of blowing in furnaces, the Bureau of Mines could be of service by compiling and publishing a number of methods of blowing in. Accordingly there are tabulated herewith 18 methods of operation:

PLANT 1.

As soon as a furnace is filled and ready to be lit we light all the tuyères and leave the sight caps open and also the relief valve on the bustle pipe. One engine (84 by 60 inches) is put on at 14 revolutions and is kept on until all the wood is burned, which takes about two hours; the sight caps and relief valve are then closed. The bells are left open for about 10 hours and the bleeder for about 24 hours. The gas is turned into the flues after 24 hours.

PLANT 2.

If a single furnace and not connected to another furnace, have the top bells and bleeders open. Have all openings on the gas mains or flues closed except at the end of the main, and allow no gas to boilers or stoves until you are ready for it. Be sure that all valves are made tight by sand or some other sure way if you do not have water valves. When ready to light the furnace have the engine going, turn the blast on the furnace, and light by means of hot rods. After a good flow of gas is coming out of the top, close the bells; this will force gas out of the bleeders, throughout the mains, and out the end of mains to the air. Allow this to continue for sufficient time to clear all mains of air, then light the gas at a stove or boiler farthest away. Be sure to keep gas away from the fire until satisfied everything is O. K. If the furnace is connected to a group of furnaces, when ready to light the furnace open a gas-seal valve and allow gas from other furnaces to drive all air out of mains and out of top and through the stock in the furnace, the top being open. When satisfied the air is driven out, open the blast valve from the engine and ignite the furnace with hot rods through the peep holes. After the furnace is lighted for a short time and gas permitted to escape from the top into the air, close the top and go ahead, using gas at stoves and boilers.

PLANT 3.

The furnace is filled, according to standard methods, with coke to cinder-notch level, scaffold to top of coolers, kindling wood under scaffold, and about three tiers of cordwood on top of scaffold; then about 100 tons of coke is put in before any ore is used. The first ore charge consists of about 1 part ore to 2 parts coke; then we put in six charges at a ratio of about 1 to 1 and finish filling with a burden having a ratio of 1.25 parts ore to 1 part coke.

We usually blow in on 5-inch tuyères, of which half are open and the rest banked, and start up on about 15,000 cubic feet of air per minute, and this volume is blown for about 15 minutes, or long enough to get the furnace started nicely, when the volume is reduced to 10,000 cubic feet per minute, which is increased by 1,000 cubic feet every three hours until 18,000 cubic feet is reached.

The tapping hole is left open and blown through until coke appears, which is usually in about six hours. The bleeders and explosion doors on the offtake are left open, and the valves to the gas line are closed, and when we are ready to turn gas down to the stoves all valves but one are opened, all explosion doors except one are closed, all dust pocket bells are opened slightly to allow any air to escape, and a good wood fire is burning in the stoves.

After these preliminary preparations are arranged and the pressure of gas on top of the furnace is sufficient the last valve is slowly opened, driving all air ahead, and gas ignites at stove burner, when the remaining explosion door on top is closed, and the other stoves are put on gas as soon as the gas pressure will permit.

The gas is then turned down to boilers in the same manner, care being taken that the number of boilers is not too great for the quantity of gas produced.

PLANT 4.

When the furnace is dry and cool a wooden scaffold is built in the furnace, composed of upright posts made of 6 by 8 inch timbers, and a platform on top of posts is made of 3 by 12 inch plank spaced about 3 inches apart. The distance from the top of the scaffold to the center line of the tuyères is 15 inches.

A manhole 3 feet wide by 4 feet long is left in the center of scaffold to pass wood up through. On top of the scaffold, around each tuyère, is built a box 24 inches wide, 30 inches high, and 36 inches long, made of seven 2 by 2 inch strips. These boxes are filled with shavings. A wooden plug 5 feet long, 5 inches thick, and tapered down to 2½ inches is placed in the tapping hole and packed around with clay to form the tapping hole. Two feet of cinder is placed on bottom of the hearth and the hearth filled in with coke to the level of the cinder notch.

Fine wood is piled on the scaffold to about 6 inches above the top of the coolers; above this cordwood is piled three tiers high, standing the wood on end, and the manhole closed with heavy pieces of wood. The bottom underneath the scaffold is filled with fine wood. The cooler, small cooler, and "monkey" are placed in the cinder notch.

The furnace is filled with 30 skips of coke and no limestone, then 4 skips of coke and 2,500 pounds of limestone 10 times. Then about 20 per cent burden is used for about 10 charges, and the ore gradually increased about every 10 charges to full weight, although the full burden is never on the furnace before 12 to 14 days after lighting furnace.

When blowing in a very light wind is used until wood is burning good in the furnace. This is accomplished by running the engines very slow and with the snort valve wide open. In about three hours 13,000 cubic feet of air is on the furnace with the snort valve shut. The blast is increased about 700 cubic feet per hour to about 80 per cent of the full volume, depending on the size of the furnace. Our system is to light the furnaces about 4 to 5 p. m. and turn gas into main on the next morning.

PLANT 5.

After the furnace lining is dried out as well as possible a wood scaffold is built in the furnace about 2 feet 6 inches above the center line of the tuyère. This consists of a double thickness of 2 by 6 inch maple on four cross girders of 8 by 8 inch pine, resting on sixteen 8 by 8 inch pine posts 8 feet 3 inches long. In the center of the scaffold an opening about 2 feet square is left through which to pass wood. On top of the scaffold are piled 4-foot lengths of dry pine in three tiers, and on end, making 12 feet of wood above the scaffold. The hearth below the scaffold is then filled with dry kindling wood, with shavings around the tuyères.

The preliminary charging of the furnace is then done, as follows:

30,000 pounds of coke, 600 pounds of "spiegel" cupola slag.
15,000 pounds of coke, 600 pounds of stone.
800 pounds of "spiegel" cupola slag, 15,000 pounds of coke.
150 pounds of limestone, 500 pounds of "spiegel" cupola slag.
15,000 pounds of coke, 800 pounds of limestone.
700 pounds of "spiegel" cupola slag, 15,000 pounds of coke.
400 pounds of limestone, 400 pounds of "spiegel" cupola slag.
15,000 pounds of coke, 900 pounds of limestone.

The furnace now stands ready to light, with 4½ by 12 inch tuyères in place, blowpipes down, bell shut, bleeder valve open, and downcomer valve shut.

The shavings at the tuyères are now soaked with oil and the torch applied at all tuyères, care being taken to have the burning uniform all around. In two or three hours the scaffold inside will fall and bright coke will show at the tuyères. Soon after this the blowpipes are put up and the furnace fanned lightly by running one blowing engine slowly with the snort valve open, producing only a few ounces blast pressure on the furnace. After standing under this light draft for about 12 hours the snort valve is closed and the furnace takes all the blast from the engine, which at the start is about 3,300 cubic feet per minute, or 10 per cent of the full blast. With the other furnace in blast when blowing in, the initial blast temperature at the tuyères will be about 800° F.

When one furnace is already in blast all the gas from the furnace blowing in passes out of the bleeder until after the first cast, when the downcomer valve is opened, allowing the gas to enter the underground gas flue and mix with the gas from the other furnace. When one furnace remains out of blast the gas from the furnace blowing in is sent out of the bleeder until the wood gas is gone and rich coke gas appears at the bleeder. All gas outlets in the underground flue are closed except to the boiler at the extreme end of the flue. The downcomer valve is then opened and the gas is lighted under the farthest boiler. When this is burning safely gas is turned into the other boilers and then into the stoves.

The cold-air mixer valve is kept shut while blowing in. The bell is kept shut and no charging is done from the time the furnace is lighted to the time the snort valve is closed and the blast is put on the furnace.

Immediately after the blast is put on charging is resumed and the burden is gradually increased, full burden being reached about five days after the furnace is lighted. After the blast is put on the volume is increased about 600 cubic feet every two hours up to 9,200 cubic feet and then increased 300 cubic feet after each cast up to full blast.

It is aimed to bring the furnace in hot and limy. No. 1 furnace was blown in as per the above schedule and it was a successful "blow-in" in every way.

PLANT 6.

After cordwood is placed and coke and limestone are charged, the stock line is about 40 feet down. Both the large and the small bell are blocked open; the water-seal valves on each side of the dust catcher are closed and the dust-catcher bell valve left open; the blowing engine started turning over 14 revolutions per minute, bleeder and snort valve open, cold-blast, hot-blast, and mixer valves closed; cinder-notch cooler and intermediate cooler in place, with the monkey out; tuyères placed, blowpipes up, and caps off; the tapping hole is left open, and sheet iron covered with sand is placed over the iron runner from the tapping hole to the first spout. The furnace is lighted by inserting hot iron bars in all tuyères. From three to four hours after lighting, ore charging is started. and when the stock line is 20 feet down, the caps are placed on the penstock, the snort valve half closed, the mixer valve opened, and the dust-catcher bell valve is closed. As soon as the gas coming from the tapping hole will burn it is lighted and permitted to burn until the hole is closed. The bells are closed about 10 hours after lighting. When incandescent coke appears at all the tuyères the snort valve is closed, usually about 24 hours after lighting, and the speed of the engine is increased 1 revolution per hour until the engine is running at 21 revolutions per minute. One revolution per minute is equal to about 740 cubic feet of air.

From two to three hours after closing the snort valve the water-seal valves are opened and gas allowed to go to the stoves and boilers. The gas is first lighted at the stove and boiler farthest from the furnace. The bleeder is closed.

When cinder appears at the tapping hole the snort valve is opened and the hole is closed, usually about 40 hours after lighting. We try for cinder 50 to 52 hours after lighting and, depending upon the amount of cinder flushed, we time the first cast.

PLANT 7.

The scaffold is built in the furnace in the usual way, and four tiers of cordwood placed on it. The preliminary charges of coke, limestone, and slag are put in, and the furnace is lighted with the tuyères in place and the blowpipes down. The wood is permitted to burn until the scaffold falls, when the blowpipes are put in place and the blast put on the furnace with the snort valve partly closed. Twelve hours after the scaffold has fallen the snort valve is closed and the blast is increased at regular periods. Before the furnace is lighted all bleeders are closed, with the exception of those on top of the furnace, which are left open, together with both bells. Steam is turned into the large dust catcher and kept on until the gas is led into the common gas main, which occurs when the blast pressure on the furnace is between 3 and 4 pounds. When the gas can be turned into the main system, the bleeders are closed on top and the water valve opened, and after the gas has entered the main the steam on the dust catcher is turned off. No charges are put into the furnace while the blowpipes are down.

PLANT 8.

The scaffold is built 18 inches above the tuyères, a 6-inch space being left between each two planks. Three tiers of 4-foot cordwood, placed on end, are then built upon the scaffold. After the cordwood is all in the furnace, small pieces of wood and a generous quantity of kindling are placed underneath the scaffold and the iron notches. After the wood is all in the furnace, the following blanks of coke, limestone, and furnace slag are charged:

Initial furnace charge.

Blanks of coke.	Coke.	Limestone.	Slag.
	Pounds.	*Pounds.*	*Pounds.*
12	100,800	....	....
4	67,200	10,480	....
4	33,600	5,240	4,800
4	33,600	5,240	9,600
	235,200	20,960	14,400

Before the furnace is lighted the goosenecks are keyed up tight. The tuyères are put in place and securely braced by wedging 2 by 4 inch timbers between them and the goosenecks. This precaution is observed in order to minimize the danger of hot coke being ejected in event of a gas explosion. Kerosene is injected at each tuyère opening and the cinder and iron notches. The furnace is then lighted by means of flambeau torches through each of these openings. Both bells and all bleeders on top of the furnace are opened before lighting.

About an hour after the furnace is lighted and the scaffold begins to fall, the first regular round of coke, stone, and ore is charged. Each successive charge of coke, stone, and ore follows at intervals of one hour until the stock is up to 20 feet.

As soon as red-hot coke appears at the tuyères, the openings are plugged with clay and a 3-inch opening left, and the furnace permitted to burn under natural draft for about 36 hours. Clay at the tuyères is partly removed from time to time to ascertain if the furnace is burning properly. Both bells and all the bleeders are left open between the charges of coke, stone, and ore. Whistle signals are blown from the cast house before a charge is lowered into the furnace to warn all workmen to stand clear. At the end of 36 hours or thereabouts the blowpipes are put up and the blast, at 600° F., is put on at the rate of 10,980 cubic feet per minute and increased at the rate of one revolution per hour until 28 is reached, then one every 24 hours up to 50. This volume is maintained for about two months. Gas is brought down to the stoves and boilers two to three hours after the blast is put on. Before the water seal is emptied the bells are opened. The seal is then emptied and the large bell is closed very carefully. When a series of dry-dust catchers is used our practice has been to inject steam into these cleaners, a vent being provided at the top of each before the gas is brought down.

PLANT 9.

After the furnace has been filled preparatory to lighting, which is to be done so as to handle the gas on the day shift, the water-seal valve is filled and allowed to overflow, then the goggle valves are opened and left open. The steam is then turned on before the furnace is lighted at the tuyères by means of red-hot rods put through the tuyère stocks and blowpipes (which are in place) and into oiled waste and shavings.

If the lining has not been thoroughly dried, or if for any reason it is desired to heat the brickwork very slowly, the furnace is allowed to start up on natural draft, if possible, by leaving the eyesight plugs out or the caps open, with the bells and furnace bleeders still open.

All openings from the top of the furnace to the water-seal valve are kept closed. If the furnace does not light on natural draft, a very light blast is put on by closing the openings in the stocks and closing the snort valve. After the furnace is lighted and a good volume of gas coming off, the bells are closed and the gas permitted to escape out of the furnace bleeders. (This applies to one furnace only, as the others are short of bleeder capacity, and on these the bells are left open until the gas is "pulled down.")

When the furnace has been on long enough to form a good coke gas, the water-seal valve is opened and the gas in the common main beyond, the pressure in which has been built up by closing the necessary burners, is forced up through the dust catchers and downcomers, driving the steam ahead of it. When a good flow of gas out of the furnace top has been established, the bells are closed and the furnace then operated as usual.

In case the gas produced by the furnace may affect the gas engines, the operation of "pulling down" is delayed until better gas is formed. In any case one or two of the furnace bleeders are left open for some time, so as to permit part of the furnace gases to escape.

PLANT 10.

We build a heavy scaffold with 2-inch plank floor 18 inches above the tuyères, a 2-inch space being left between floor planks; fill bottom to cinder notch with cordwood placed on end; put two tiers of cordwood on end on scaffold, fill from cinder notch to scaffold with kindling, with space in front of each tuyère for shavings; block the tuyère coolers until ready to light the furnace; and then fill the furnace.

Our furnaces are entirely separate, and stoves on one furnace can not be heated with gas from another furnace.

The furnace is filled to within 20 feet of the top. The distribution inside the furnace is examined.

All gas burners at stoves and boilers are racked back, and the whole gas-main system is shut up tight from the top of furnace to the ends of the mains, We have no water seal or spectacle valves at either furnace. We are particularly careful to prevent gas coming in contact with fire until we are ready. On stoves we seal burner door from the time furnace is lighted until we are ready to turn gas into the stoves. At No. 2 boiler house, blanks are put in the main; at No. 1 boiler house the mushroom valves are closed. In addition the openings in boiler fronts are bricked up, and sheet iron placed over end of burners.

All bleeders, explosion doors on top, and both bells are opened. Steam through 2-inch line is turned into dust catcher and allowed to fill the mains, dust catcher, and downcomer.

Oily shavings are put inside at each tuyère, the blowpipes are put up, the iron hole left open, the cinder notch closed, a very light blast put on, and the furnace lighted at each tuyère. After one hour the snort valve is closed, the cold-air valve of stove on furnace is opened one-half, with one engine running at the minimum number of revolutions, or blowing 13,000 cubic feet per minute.

We blow in with 5-inch tuyères on each furnace, increasing these gradually to 6 inches on No. 1 furnace, and to 7 inches on No. 2 furnace, when the full blast is on. Seven hours after lighting, by which time the wood gas is practically gone, we close the large bell and enough other openings on top to give 3 inches of water pressure in mains. We then open the doors in the ends of the mains and the pockets at gas legs, let gas and steam escape freely, care being taken to insure there is no chance of ignition, and also drain all the pockets at the gas legs. We then turn off steam and repeat above with gas alone.

We turn gas into the stove nearest furnace, being sure of 3 inches of gas pressure at this time, and then into the other two stoves. After this, as fast as the gas supply increases, we give it to the boilers. The iron hole is stopped with the blast on about five hours after lighting. In case we do not feel it necessary to examine distribution, we leave blowpipes down and fill only to within 35 feet of top. After lighting at tuyères, we let furnace burn with natural draft for 18 hours, if we can keep all tuyères burning evenly. In this case we pile three tiers of cordwood on and above the scaffold. At the end of 18 hours we bring gas down to stoves and boilers as described above. The volume of blast, burden, and other furnace data are shown below:

Data on blowing in furnace.

Hour.	Volume of blast, cubic feet per minute.	Engine speed, revolutions per minute.	Blast pressure, pounds per square inch.	Hour.	Volume of blast, cubic feet per minute.	Engine speed, revolutions per minute.	Blast pressure, pounds per square inch.
1	(a)	(a)	0	12	....	20	1
2	(b)	(b)	0	13	....	20	1
3	13,000	17	¼	14	....	20	1
4	....	18	½	15	....	20	1
5	....	18	½	16	....	21	1¼
6	15,200	20	½	17	....	21	1¼
7	....	20	½	18	....	22	1½
8	....	20	½	19	....	22	2
9	....	20	½	20	....	23	2¼
10	....	20	½	21	....	23	2½
11	....	20	¾	22	....	24	3

[a] Snort valve open, cold-air valve slightly opened.
[b] Snort valve closed, cold-air valve half open.

On the last blow-in the first flush was obtained in 26½ hours, the second flush in 28 hours, and the first iron in 31 hours, 12½ tons of iron containing 3.68 per cent silicon and 0.021 per cent sulphur being obtained. Filling subsequent to that given in the table was as follows: Ten rounds (25 per cent Mesabi ore), with a burden ratio of ¾ to 1; seven rounds (50 per cent Mesabi ore) with a burden ratio of 1 to 1; seven rounds (50 per cent Mesabi ore) with a burden ratio of 1.25 to 1; seven rounds (50 per cent Mesabi ore) with a burden ratio of 1.30 to 1; the ratio was increased 5 per cent each seven rounds until a ratio of 1.75 to 1 was reached, which was 58 hours after lighting furnace with 25,000 cubic feet of air blowing and with 6-inch tuyères.

PLANT 11.

A wooden scaffold is built just above the tuyères and three tiers of wood placed on top of this scaffold, then the bottom of the furnace is filled with wood. We then start filling. When sufficient number of blanks of coke and first few charges of ore are in, we light the furnace (all blowpipes and tuyères being in place and tapping hole closed) by turning the blast on through a hot stove. Engine is kept turning over slowly, and the amount of blast regulated with the snort valve. During all this time the water seal is filled to keep the gases from the balance of the system from coming back and mixing with those of the furnace going on. The bleeders are kept open (both bells closed) until gas can be lighted on top of the bleeders (which requires 6 to 10 hours), then the gas from the furnace being blown in is turned into the system. During all this time a jet of steam is kept in the first dust catcher and gas main. Blowing in is done with tuyères and blowpipes in place, which eliminates any danger of men being burned when scaffold falls.

PLANT 12.

The furnace is filled as follows: First, fine kindling wood is laid on the bottom of the hearth to a depth of 3 or 4 feet. This wood is piled vertically, commencing against the walls at four equidistant points. When these piles are built out to within about 2 feet of each other the cross-shaped space between them is filled with kindling laid flat. One of the ends of this cross is directly in front of the tapping hole. On top of this kindling dry cordwood is piled on end to a height of 4 to 6 feet above the tuyères. No scaffold is built in the furnace. Coke and limestone are charged on top of the cordwood. The furnace is filled to the stock line. The last few charges contain a small and gradually increasing amount of ore, which in the last charge filled does not much exceed half the weight of coke. All dust-catcher and gas-washer bells and doors are closed, and the water seal connecting the furnace with the gas main is filled. No openings in the path of the gas from the top of the furnace are permitted. The bleeders or explosion doors on top of the furnace and both bells remain open, so as to give all the vent possible for gas at the top of the furnace after lighting.

The furnace is lighted at the tapping hole with a coal-oil torch made of waste wrapped around the end of a bar or by hot blast with no other ignition. Where hot blast is available its use for lighting is preferred. The air blown is delivered by one engine turning over as slowly as possible. The amount delivered to the furnace is at first a mere breath, the greater part going out the snort valve. The blast heat used at this time is not above 900° F. Coke will appear at the tuyères within two or three hours after lighting. The amount of blast is gradually increased by closing the snort valve, so that it is entirely closed in about 15 to 18 hours after the furnace has been lighted, the volume of air being then about 9,000 cubic feet per minute. This amount is not increased until after the bells have been closed, which is done 24 to 28 hours after the furnace has been lighted. The bells are always closed on the afternoon of the day after the one on which the furnace was lighted. The practice of lighting the gas on top of the furnace before closing the bells by lighted waste carried up by the skip car is followed. This is an old practice, intended to see whether the gas will ignite. We have never known it to fail to do so, even when the bells were closed within a few hours after lighting the furnace, and it is questionable whether there is any benefit derived from the practice. The method of handling the gas depends upon whether other furnaces at the plant are in blast and making gas. If so, the gas from the other furnaces is under pressure at the water seal of the furnace blowing in. A 3-inch steam line is connected into the furnace for several minutes to exhaust the air from the dust catchers and large gas containers between the water seal and the top of furnace. The gas on top having been lighted in the meantime, the water seal is broken and the gas from the other furnace goes to the top of the one blowing in. The steam is shut off as soon as the water seal is broken. As soon as it is certain that the gas from the main is coming out of the furnace top the main bell is closed and the gas from the new furnace turned into the main, after which the bleeders are closed and charging commenced. No charging is done from the time the furnace is lighted until the bells are closed, and the furnace will usually be about 16 feet below stock line at this time.

If there are no other furnaces in blast at the time of blowing a furnace in, it is necessary to close the bells and fill the gas system connected to the furnace with gas forced downward from the top. In this case all openings along the gas main are closed, excepting a stove at either end and an outlet at the water seal. We have never blown in a furnace under these conditions since the use of steam was commenced at this plant. In order to derive any benefit from its use in the dust catchers of the furnace blowing in it is necessary to close the bell and force the gas downward immediately after breaking the water seal. This is always done with or without steam.

Our practice, then, after lighting the gas on top is to fill the dust catchers with steam, break the water seals, and close the bells as nearly simultaneously as possible, force the gas to all ends of the main, providing a relief to prevent dead ends. As soon as this is done the gas is lighted and burned at convenient points along the main.

PLANT 13.

A layer of coke, extending from the hearth bottom up to the bottom of the iron notch, is put in by hand through the cooler. Cordwood is then put in on end, bringing the layer up to the bottom of the tuyère cooler. A layer of fine kindling is then put in, extending from the bottom of the nose of the cooler to the top, and at each tuyère is placed a bag of shavings with a ball of oily waste right at the nose of the tuyère. Cordwood is then placed in the iron notch on the bottom and the notch plugged with clay. The monkey is plugged with cinder "bot." From 150,000 pounds of coke for a 60-foot furnace to 300,000 pounds for a 100-foot furnace is then filled on top of the wood by dumping it in from the top. A burden of slag, coke, and ore is then filled in, starting with an 0.5 to 1 burden and aiming at a 1 to 1 burden when furnace is full. The amount of slag used on a 60-foot furnace is 88,000 pounds and on a 100-foot furnace 24,000 pounds.

When the furnace is full a careful inspection is made of all valves, cleaning doors, and dust legs to see that all are in proper condition for manipulation. A barrel of oil is distributed in the tuyères, the bleeder is opened up, the explosion doors on top, if any, are opened, all other openings are closed, and if the furnace is connected by gas mains with other furnaces the water or dry valves are closed, isolating the furnace to be blown in.

The blowing engine is now turned on, with the snort valve adjusted to give 5,000 cubic feet for a 60-foot furnace and 11,000 cubic feet for a 100-foot furnace. The stove next the furnace, which has previously been heated up with wood or gas from another furnace, is now opened up, the hot-blast valve being opened first, and then the cold-blast valve. When the cold-blast valve is opened hot rods are put in through the face plates and into oily waste at the nose of each tuyère. When the tuyères show light the rods are pulled out and the tuyère caps closed. The gas is lit at the iron notch and let burn until the hole plugs itself.

After the first movement of the furnace burden, which takes place generally in about two hours, the volume of air is increased at about 800 cubic feet per hour on a 100-foot furnace and 300 cubic feet on a 60-foot furnace.

In about 5 hours the bleeders and explosion doors on top are closed. In about 45 minutes the cleaning door at the base of the leg to the stove farthest from furnace is opened slightly and the gas permitted to escape. At this time the cleaning door or other opening on the extreme end of the main to the boiler house is also opened and gas allowed to escape until there is a certainty that all air has been expelled from the mains.

The blast is then increased about 2,400 cubic feet for a 100-foot furnace and about 300 cubic feet for a 60-foot furnace, and the gas is racked into the boiler and stoves on the end of the line. The volume of air is now increased 800 cubic feet on a 100-foot and 300 cubic feet on a 60-foot furnace per hour until the first flush of cinder, or about 18 hours after blowing in, when it is adjusted according to the amount of cinder. The furnace is allowed to move 6 or 8 feet before filling, and in the meantime the remaining stoves and boilers are lighted.

If the furnace is connected by a gas main to other furnaces, the water or dry valve is now opened and the furnace is put on the line. In drilling for the first cast the tapping hole is started just above the 2-inch pipe in the iron notch, which generally burns out on first cast, giving a chance for a good stop on the iron notch after the cast is made.

PLANT 14.

After the brickwork and top is complete a gas fire is allowed to burn in the hearth for a period of about two weeks, which insures thorough drying of the furnace, after which time all foreign material is taken from the hearth. If the hearth is built on an old salamander, then asbestos pads, placed on 10 brick piers 10 inches high, are distributed equally over the bottom. On top of these pads are placed posts 9 feet long by 8 inches square.

On top of these posts is built a strong but loosely laid platform of 2½-inch boarding. About 24,000 pounds of coke is then scattered on the hearth, bringing the filling up to the level of the cinder notch. A hole is then left in the scaffold floor and 4-foot lengths of coarse wood are laid loosely, so as to give plenty of air space. On top of the coarse wood on the scaffold a coke blank is filled in with the hoisting apparatus, as follows: Coke 120,000 pounds, limestone 1,800 pounds; coke, 72,000 pounds, limestone 1,800 pounds; coke 48,000 pounds, limestone 1,800 pounds; coke 48,000 pounds, limestone 1,800 pounds, making 288,000 pounds of coke and 7,200 pounds of limestone.

After all the coke blank is in, a short period is allowed to elapse in order to examine all the appliances.

In the meantime we have had an engine turning over slowly in the engine rooms. The blast is now up to the cold-blast valves on the stoves. If possible, gas has been turned into the stove mains on the previous day from another furnace, and after the gas is ignited by a wood fire in the wells of the stoves of the furnace to be blown in the heat on the stoves is raised as fast as consistent with good practice. The stove at the end of the gas line receives its gas first.

The furnace is now lighted through the tuyères, cinder notch, and tapping hole with the aid of long poking rods which have been dipped into a ladle of hot slag, care being taken to see that the furnace is lighted all the way round. Before lighting, the bells and bleeders are propped open. In about four or five minutes after lighting smoke will commence to appear at the furnace top; the fire in the furnace is allowed to burn with air it will draw in.

When we are certain that the furnace is burning nicely all around, the burden is started. Approximately 50 per cent Mesabi ore with a heavy ratio is used.

Burden used in filling a furnace to be blown in.

Number of rounds filled.	Ore,			Coke.	Stone.	Ratio of bases to acids.
	Mesabi.	Old range.	Siliceous.			
	Pounds.	*Pounds.*	*Pounds.*	*Pounds.*	*Pounds.*	
10	6,720	1,920	960	12,060	3,950	1.29:1.00
10	7,290	2,100	1,020	12,000	4,140	
10	7,890	2,280	1,020	12,000	4,200	
5	8,460	2,460	1,080	12,000	4,380	
5	9,030	2,640	1,110	12,000	4,620	
5	9,630	2,820	1,170	12,000	4,800	
5	10,200	3,000	1,200	12,000	4,980	1.33:1.00.
5	10,890	3,810	1,290	12,000	5,220	
5	11,610	3,990	1,410	12,000	5,520	
5	12,300	4,500	1,500	12,000	5,700	
5	12,900	5,010	1,590	12,000	6,000	
5	13,000	710	5,490	12,000	6,240	
(a)	14,400	6,000	1,800	12,000	6,480	1.32:1.00

[a] Full burden.

The coke blank and approximately 20 rounds on the schedule fills the furnace up. After the furnace is filled it is allowed to stand until the scaffold has fallen.

In the meantime the tapping hole has been closed, and after the scaffold has fallen caps are put on the tuyère stocks and the cinder notch is closed. The furnace is now closed up and the stove blast valve is opened. The snort valve on the pipe from the engine room is also closed. The engines are now running at approximately 12,000 cubic feet per minute. Then the blast is raised every two hours approximately 500 cubic feet per minute until 30,000 cubic feet per minute is reached. At this point the furnace is generally held for some time. The heat from the stove during this period will be approximately 800° or 900° F. The stoves are changed in rotation, one every hour.

As a rule, the first cinder flush will occur after 50 rounds of burden, or in about 36 hours after lighting. The first cast is usually made after about 54 rounds of filling, or about 39 or 40 hours' time after lighting.

In the meantime, as soon as the gas burns on the top of the furnace, all bells are closed and the gas is allowed to discharge through the bleeder stack. After the first cast is made the pressure on the gage will possibly be about 8 or 9 pounds. This will give a sufficient gas pressure on top to equalize that in the mains, at which time, after all air is expelled from dust catcher, mains, etc., the water valves on both the stove and the boiler-house lines are opened and the bleeder on top of the furnace is closed. After a certain period the burden on the furnace is changed to the regular allotment used at these works.

PLANT 15.

The furnace is always dried out with hot blast for by using this method the stoves, tuyère stocks, and everything pertaining to regular operation has been operated and the furnace man is left free to pay all of his attention to the furnace itself because he knows that everything else will work properly.

The hearth of the furnace is either filled with loose, small wood up to about a foot above the tuyères or a platform is built up to the same height with loose wood underneath. On top of this about four rows of cordwood are placed, which will reach nearly to the mantle. Then a large coke blank is placed on top of the wood and on top of this blank the burden is started with about one-half pound of ore per pound of coke, and the ore content increased so that when the furnace is full the ratio will be about 1 pound of ore per pound of coke. I always find it is a good practice not to fill a furnace as full as it is filled during the regular operation. In front of each tuyère light shavings or oily waste is placed and when the furnace is ready to be blown in this waste is saturated with kerosene. A hot stove is put in the furnace, with a light blast, and hot poking rods are inserted into the eyesight holes. During all this time the bells are necessarily lowered, as well as the bleeders being open, the gas cut-off valve being always closed. The tuyères are watched and a short time after good incandescent coke shows at the tuyères and the gas on the top of furnace looks right, the gas from the other furnaces, provided there is more than one furnace, is sent through the gas cut-off valve, which may be a Crawford seal or any other type, and allowed to fill the dust catchers and downcomers of the furnace being blown in, and as soon as sufficient pressure shows on top of the furnace through the open bells the large bell is closed and the furnace is in blast. The downcomers, dust catchers, and all gas piping up to the cut-off valve are kept filled with steam so as to exclude the air.

PLANT 16.

The furnace is dried out thoroughly, either by hot blast, or, if this is not available, by a wood and coke fire built in the crucible. In the latter case, combustion may be promoted by bulkheading all cooler, cinder, and iron-notch openings and inserting draft pipes leading to the bottom of the crucible, thus bringing an induced draft direct to the fire.

A scaffold is built in the furnace above the tuyère coolers, and the crucible is filled with coke to the bottom of the tuyère coolers. Cordwood is loosely stacked on end, usually three lengths or more. Filling is then begun and carried to such point as judgment and experience indicate there will be a structure which admits of good natural draft.

The furnace is lighted with inflammable material at all tuyères and iron and cinder-notch openings. Precaution is taken to see that the bell is fastened open, as are also the bleeder valves on the downcomers. The furnace is allowed to burn on natural draft until the scaffold has fallen or burned and until coke is burning thoroughly at all tuyères, then the blow pipes are put up and the blast applied.

All gas mains, both overhead and underground, are filled with steam before gas is turned in, one or more boiler or stove gas-burner valves are opened at the ends of the mains, and in the case of underground flues a manhole cover of large area is removed to facilitate a rapid passage of all gases through the mains.

In turning gas into the mains all gas is allowed to escape from the top of the furnace and bleeders until it lights and burns with a clear and steady flame on contact with air. Precaution is taken to see that live steam is still entering the mains. At a given signal the main sealing bell and all bleeder valves and other openings are quickly closed, and as soon as gas has lighted at the extreme end burner valves, the valves on the intermediate burners are opened. Lighted oily waste or a wood fire is always built at the burners on cold stoves or boilers so that gas is burned from the first, thus forestalling any possibility of an explosion in stoves or fire boxes.

Where water-seal valves are available, they are always kept closed so that no gas can get into dust catchers or mains not in use. It is considered essential that the blast pressure shall not be considerably slackened or taken off the furnace during the first few hours after turning gas in mains, as a rapid circulation well-nigh precludes any possibility of an explosion.

PLANT 17.

In preparing a furnace for blowing in, first a scaffold is constructed across the furnace at a point about 1 foot above the tuyères. Coke is thrown in at the tuyères, covering the furnace bottom to the top of the iron hole. About 8 cords of wood is placed on end on top of the scaffold and fine wood is filled in below the scaffold, with a pile of shavings in front of each tuyère.

The last blowing-in burden was as follows:

Number of rounds.	Material.	Weight, pounds.
5	Coke	5,700
5	....do.	5,700
	Blast-furnace slag	2,100
8	Coke	5,700
	Blast-furnace slag	1,400
	Limestone	800
6	Coke	5,700
	Blast-furnace slag	4,200
	Limestone	1,000
12	Mesabi A	2,775
	Mesabi B	925
	Old range	1,850
	Siliceous	300
	Total ore	5,850
	Limestone	1,900
	Blast-furnace slag	1,400
	Coke	5,700
4 [a]	Ore	[b] 6,300
	Limestone	2,000
	Coke	5,700

[a] Furnace full.

[b] Proportioned as above.

Both the explosion doors and the bleeder were fastened open and orders given to keep the charging bell closed until permission was given to start filling. A 3-inch opening was left to burn gas through iron hole. Alternate tuyères were plugged with clay, leaving five 6 by 12 inch tuyère openings, through each of which 6 gallons of torch oil was poured. The furnace was lighted with shavings partly burned in the hot-blast stove before blast was turned into it at 8.50 a. m. Data on the method of furnace operation follow:

Data on furnace operation at blowing in.

Time of day.	Time elapsed.	Volume of air per minute.	Blast temperature.	Number of charges.	General remarks.
a. m.	*Hours.*	*Cu. feet.*	*°F.*		
8.50	....	7,000	450	....	Lighted furnace.
9.20	½	....	....	....	Hot coke showing at tuyères.
p. m.					
12.50	4	....	....	....	Gas brought down.
2.10	5⅓	....	500	....	Plugged iron hole.
3.00	6⅙	8,400	....	....	Started to fill.
4.50	8	....	535	11	Stock line, 14 feet.
6.00	9⅙	9,800	550	12	Stove every hour.
6.50	10	10,500	585	14	
8.20	11½	11,900	625	16	
8.50	12	13,200	675	17	
9.30	12⅔	14,000	....	19	
10.00	13⅙	....	725	21	First cinder flush (A).
a. m.					
12.05	15¼	....	775	26	Second cinder flush (B).
2.00	17⅙	....	800	29	Third cinder flush (C).
4.00	19⅙	....	850	33	Fourth cinder flush (D).
5.30	20⅔	12,200	....	37	First cast (E), 6 tons of iron.
10.10	25⅓	....	950	40	Second cast (F), 11 tons of iron.
12.00	27⅙	14,000	960	44	
p. m.					
1.00	28⅙	14,700	990	46	
2.00	29⅙	15,400	1,020	49	
3.20	30½	....	1,040	53	Third cast (G), 20 tons.

Composition of iron and cinder.

Sample.	Silica.	Alumina.	Iron oxide.	Lime.	Magnesia.	Manganese.	Sulphur.	Silicon.
Cinder:								
A	34.40	15.33	0.51	44.50	3.40	.13	1.84	....
B	34.43	15.78	.51	44.00	3.40	.13	1.75	....
C	35.07	15.42	.38	44.20	3.04	.10	1.79	....
D	35.03	15.68	.32	44.30	2.80	.10	1.77	....
Iron:								
E	....	....	....	....	....	....	.105	3.95
F	....	....	....	....	....	....	.030	4.37
G	....	....	....	....	....	....	.037	3.46

General data on furnace operation during first week of run.

Day.	Volume of air per minute.	Blast temperature.	Blast pressure (per square inch.)	Tuyères.		Ratio of ore to coke (coke=1).	Iron produced.		
				Number.	Size.		Tons.	Silicon content.	Sulphur content.
	Cu. feet.	*°F.*	*Pounds.*		*Inches.*			*Per cent.*	*Per cent.*
First	10,080	670	3.1	5	6 by 12	1.28	6	3.95	0.105
Second	15,190	840	5.8	5	6 by 12	1.51	98	4.30	.023
Third	20,475	940	10.5	5	6 by 12	1.90	208	2.44	.015
Fourth	22,120	980	11.8	7	6 by 12	1.94	330	.97	.071
Fifth	27,090	1,000	13.4	8	6 by 12	1.97	323	1.28	.032
Sixth	28,630	1,010	13.2	4	6 by 12	2.60	406	1.26	.028
Seventh	29,960	1,000	13.5	5 5 5	6½ by 9 6 by 12 6½ by 9	2.03	419	1.20	.028

Gas was permitted to escape through the explosion doors and bleeder until after wood had been consumed and furnace had heated up sufficiently and ore descended far enough to produce more nearly normal gas. Up to this time all openings on gas flues and dust catchers had been kept closed. It should be noted that one furnace was in operation and that its gas was sealed off by the Crawford water-seal gas valve. The stove gas burners are supplied through bootlegs. At the base of these pipes, and also on the bootlegs supporting the boiler gas flue, cleaning doors are provided for the removal of flue dust. These doors are now fastened slightly open, leaving about the equivalent of a 1-inch hole, all other valves remaining closed. One of the explosion doors is then slowly closed with a set of blocks until the gas starts to force air through the openings just provided. It may be necessary to partly close the second explosion door before the air is forced out, but in any case the explosion doors are closed slowly and only far enough to cause sufficient pressure to start the flow of air, where they remain until the gas has forced out all the air. The explosion doors are closed far enough to give appreciable pressure to the gas when a wood fire is started in the stoves and the burners carefully opened. After the gas is lighted in the stoves the bootleg openings are closed and the water in the Crawford valve is released, uniting the gas from the two furnaces. Gas is permitted to blow through the iron hole until cinder starts out, when the hole is plugged.

If one furnace had not been in operation it would have been necessary, in bringing the gas down, to force the air out of the gas flues in the boiler houses. The procedure would then be as described, except that the Crawford valve would contain no water, and would have its water connections and bell closed. All boiler gas burners would be closed, except one at the end of the underground flue, which would be pulled slightly beyond the closed position, permitting the air to be forced out in front of the burner. The cleaning doors would be opened the same as at the stoves. The same system would be used for either furnace, the idea being to keep all flues tight except small openings on every dead end.

PLANT 18.

First we build in the hearth a scaffold of about 6 by 8 inch timber, the top of which is above the tuyères. Uprights are placed between every other tuyère opening, the top of the scaffold being boarded loosely. The hearth is then filled with clean coke to the level of the cinder notch. Cordwood, or any heavy wood, such as split ties, slabs, etc., is then taken into the cinder notch and piled on end on top of the scaffold. Two tiers of such material, or sufficient to reach about the fifth row of plates, are then placed in the furnace, the men working from the outside to the center. The center opening of the scaffold is then bridged across and wood placed in position by building down from the top of the wood to the original scaffold at the tuyère level. Split kindling wood is then filled into the opening formed between the coke in the hearth and the scaffold. Two bags of shavings are put in front of each tuyère. When the hearth is finally filled in in this manner the cinder-notch coolers and "monkey" are placed in position, and the furnace is filled as follows:

Burden used in blowing in furnace.

Material charged.	Pounds.
Ten rounds:	
4 buckets coke (3,800 pounds each)	13, 200
1 bucket limestone	2, 800
Eight rounds:	
4 buckets coke	13, 200
1 bucket limestone	3, 000
1 bucket blast-furnace slag	4, 000

Material charged.	Pounds.
Four rounds:	
4 buckets coke	13,200
1 bucket limestone	4,200
Old range	6,000
Mesabi	6,000
Blast-furnace slag	2,000
Until filled (24 rounds):	
4 buckets coke (3,800 pounds each)	13,200
1 bucket limestone	5,700
Old range	6,000
Mesabi A	10,000
Mesabi B	6,000
Blast-furnace slag	1,000

After the furnace is filled, care is taken that no fire or sparks can come in contact with the hearth. In case we are building on a hot salamander, we may put several feet of sand on the bottom and place the wooden scaffold uprights on brick piers to protect them from the heat. We generally aim to turn on the cooling water before the furnace is filled so that the men can inspect the plates and coolers and determine if any are leaking before the furnace is finally lighted. The mushroom valves on the dust catcher, as well as the water seals, are closed; all doors on gas mains are closed and sealed, the bleeder and two explosion doors opened. The furnace is now in position to be lit. The stoves are fired up three or four days beforehand by means of gas, coal, and wood. The stove nearest the furnace is filled with a wood fire and allowed to burn, so as to form a large body of embers. About 2 bucketfuls of carbon oil is injected in each tuyère, thus saturating the shavings, and the caps are then closed. The main bell and the small bell are also closed. While these preparations are being made, the engines have been started and are running at about 25 or 30 revolutions with the snort valve open. When the bells have been closed and everything around the bottom of the furnace made tight, the stove is then closed, the hot-blast valves and cold-blast valves are opened, and the blast turned into the furnace in the usual manner. Sparks are carried over from the stove and ignite the oil-saturated shavings, and in a few seconds gas is issuing from the bleeder and explosion doors. The tapping hole is left open and the gas issuing from it is ignited, as well as all gas which is leaking around the tuyères and plates. After the furnace is well ignited, the speed of the blowing engine is reduced to about 22 revolutions, and from this point is gradually increased as furnace conditions demand; in about 15 or 20 minutes coke appears at the tuyères and the furnace starts to take stock. If there is no reason for taking the gas off a furnace, the gas may be permitted to escape at the bleeder until the first flush of cinder is taken out, which is possibly 12 hours after lighting. Then the gas valves between the other furnaces and the one just blown in are opened, allowing the gas to mix. In case anything should happen to require a stop on the newly blown-in furnace, gas from the other furnaces is allowed to go over so as to maintain a constant gas pressure under the bell and in the mains. It is always our aim to keep a constant outward pressure at all points, the mains generally leaking slightly. After blowing air through the tapping hole for four or five hours, the gun is placed in position and the hole stopped up. After an hour or so from the time of blowing in, the explosion doors are closed and the bleeder is allowed to carry all the gas. This depends altogether, however, on the quantity of gas produced, care always being taken that we have sufficient pressure to keep the mains full. When the gas is taken to the boilers, it is drawn to the end of the line and into the farthest boiler, where it is allowed to ignite, and the boilers are put on in rotation from this end. This cleans out the main completely and does not allow any trap there.

METHOD IN USE AT ANOTHER PLANT.

One method not mentioned previously is as follows: When everything is ready for lighting the furnace, close the bells, bleeder, and mains, open a burner at the extreme end of the main into a boiler setting in which a hot fire is kept burning. The furnace is then lighted with the wind on and the top and mains full of air. No wood is used in the furnace, charcoal being placed at the tuyères. This method has been followed for several years by one operator, the gas lighting quickly at the burner within 10 minutes. But one explosion has resulted and this was due to the bell opening by accident as the gas was being taken down. This illustrates a possible hazard from this method; that is, some part of the machinery going wrong at the early stage of blowing.

PRECAUTIONS IN BLOWING IN.

The chief danger in blowing in furnaces is the danger of gas explosions. Other possible sources of danger and damage exist and are discussed in this report, but as in blowing out many details of practice must vary with circumstances and plant layout and only the general operations are noted here. The most important points to be watched in blowing in are as follows:

1. Drying the furnace is preferably done by hot blast when gas from another furnace is available. This enables the furnace man to assure himself that stoves, valves, stocks, and other equipment are in working condition and eliminates some chances for accident due to the unforeseen necessity of taking the blast off and allows him to devote his attention to the working of the furnace.

2. In case the hearth bottom, or the top of the salamander, is excessively hot it is advisable to place a 12-inch brick pier with asbestos padding on top beneath the scaffold uprights and to place about 20 inches of crushed slag on the hearth bottom before putting wood or coke in the hearth. It is also a good plan to turn the water into the tuyères coolers, etc., before the furnace is filled to see that all connections are water-tight and not stopped; the bronze should be inspected for leaks from the inside.

3. In placing wood or charcoal in the hearth and bosh, the use of electric lights should be compulsory, accident being invited by the use of candles, torches, or open lights, or by allowing men to smoke. All men not required for the work should be kept away, such as visitors and spectators.

4. Before lighting the furnace the foreman in charge should, by personal inspection, assure himself that all gas burners are sealed;

burner doors closed; openings on mains, dust catchers, and downcomers tightly closed; the bells, bleeders, and explosion doors on top fastened open; and the water seal or other valves in good condition, special attention being given to see that all mechanical valves are tight and the water connections open, and that the burners are tight; if necessary, some clay or dust should be put on the burner slides to act as a seal.

5. If the blast is put on immediately when the furnace is lit, the chances of men being burned by coke or flame shooting from the tuyères when the scaffold falls or from a "crack" of gas at the top are eliminated. If the furnace should be lit and allowed to burn on natural draft for a period of 6 to 36 hours, the following precautions are suggested:

(*a*) If the blowpipes are up and the caps off or the eyesights open, the foreman should be sure that the hot-blast valves are securely seated so that no draft will be induced into the blast main.

(*b*) If the blowpipes are not up, but the tuyères are placed, the tuyères should be blocked in place with a 2 by 4 inch timber against the stock, the latter being held by the bridle.

(*c*) The foreman should keep everyone away from the tuyères and stocks until he is satisfied that the scaffold has fallen, and should not start to put up blowpipes until this has happened. In drafting in, the scaffold will usually fall in three to five hours if care is taken that the furnace is nicely lit all the way around and it burns evenly at all the tuyères.

6. The method of handling gas depends upon whether other furnaces at the plant are in blast and are making gas, but if it be assumed that no other furnace is in blast the following notes are typical of safe practice:

(*a*) The gas pressure should be kept up at all cost after the gas has been brought down into the mains, 3 inches of water pressure being regarded as the minimum. Accidental lowering of the bell or bells so as to leave the top open after the gas is brought down should be guarded against.

(*b*) The foreman should be sure the gas is all right, clear of wood smoke and gas, before bringing it down. The first hour after lighting produces the most explosive gas, so time should be allowed for the ore to get into action before the gas is used. The time of taking the gas down varies much, ranging from immediately to 36 hours after the blast is on. A safe rule to follow is to wait eight hours after the scaffold has fallen, and then, after the furnace is filled and the coke is bright at the tuyères and the blast pressure amounts to 4 pounds, proceed to bring the gas down. This period of time will assure the smooth working of the bell and there will be less chance of some unforeseen difficulty preventing the bells from closing or opening. There is certain to be ample gas pressure in the mains, and the gas, though very explosive when mixed with air, is not markedly more so than later.

(*c*) The essential idea is to clear all mains, dust-catcher systems, etc., of air, being positive that none is trapped in any space or dead end before the gas is lighted, the aim being to constantly keep a positive outward pressure. To this end it is requisite that the blast shall not be taken off the furnace during the first few hours after turning the gas into the mains, or, if the blast is considerably slackened, that enough burners be closed to maintain 3 inches of water pressure in the gas mains.

(*d*) After personal inspection to see that all burners and burner doors are shut, the valve, if any, between the stoves and dust catchers is opened, and the bells closed if still open, at as nearly the same time as possible. The bell will usually have been shut previously, however. Before the gas is brought over into the main, the air should be expelled from the dust catcher through the dust bell and bleeder, if a bleeder is provided on the dust catcher. After making sure there is no fire near the gas mains and that the burner doors are shut, then the bleeder at the extreme end of the gas main, or the cleaning door at the base of the down leg to the stove or boiler farthest from the furnace should be opened. When sufficient time has elapsed to make sure that all the air has escaped from the mains the cleaning door may be shut, the burner door opened, and the gas lighted in the stove. Unless the stove is incandescent at the bottom, a large bunch of oily waste or a wood fire at the gas inlet should be used to insure instantaneous lighting. It is dangerous to permit gas of this nature to pass into a stove without lighting it immediately, as, if the well fills with an explosive mixture of gas and air, an explosion or a "crack" of sufficient violence to throw many types of burners off their seats may result. Such damage is especially serious and fraught with danger at this stage of blowing in, because a shut-down may be necessitated, and more serious explosions are possible.

(*e*) Similar methods are followed at the boilers, the exception being that instead of allowing the gas to escape into the boiler house when no bleeder is provided at the end of the line, it is preferable to have the boiler at the end of the line cold, and draft the boiler-house main through this boiler until the gas is coming strong, when it may be lighted at an adjacent boiler.

(*f*) In lighting stoves and boilers, the danger, after successful lighting of the first one, lies in putting too many on. It is better to have too few lighted than too many; the gas pressure should be maintained at at least 3 inches. In lighting gas from a furnace newly blown in, additional supervision, both at boilers and stoves, is advisable, and this step should never be undertaken without the supervision of the furnace foreman or blower.

(*g*) In case a shutdown becomes necessary on an isolated furnace, it is better not to draft back. Shut all the burners and keep the bells and bleeders closed before taking the blast off. This insures a positive pressure under the bell and avoids an updraft through the bleeder, and if the pressure is too strong at the tuyères a few caps can be knocked off to afford relief. When a furnace is started up after a stop of this nature it is best, in lighting the gas, to observe all precautions taken in first bringing the gas down.

(*h*) If a steam line is connected to the dust-catcher system, the system should be filled with steam prior to shutting down, and the steam kept on until the blast is put on again. Steam is a valuable auxiliary in blowing in, and if a connection is provided steam should be turned into the dust catchers shortly before the furnace is lit, and kept on until the gas has been brought down and is passing out of the mains through the bleeder.

7. If one or more furnaces are in blast, connecting them with the gas mains of the furnace being blown in simplifies taking the gas down, as one is always assured of ample gas pressure in the mains.

(*a*) If gas from another furnace is available, the gas is usually being burned in the stoves of the new furnace at the time it is being blown in. If not it is a simple matter to bring the gas up to the valve separating the dust catcher from the main of the new furnace, clearing the downlegs of the stove burners of this furnace. At the determined time, which may be earlier than with an isolated furnace but not before the scaffold has fallen, with bright coke showing at the tuyères and the snort valves shut, the stoves along the line should be closed temporarily to be sure that the gas pressure in the mains is greater than that at the top of the furnace being blown in, the dust-catcher valve opened, and, the furnace bleeder and bells being open, the gas from the other furnace is permitted to pass through the dust catcher and up the downtake, driving the air out through the furnace top.

(*b*) When the air is displaced, the bells and bleeder may be shut and the stove and boilers put on, care being taken to clear all dead ends in the gas mains, and observe the usual precautions with regard to lighting gas.

(*c*) If there is a steam connection to the dust-catcher system, it may be turned on prior to admitting gas from the other furnace; indeed, it is the safest practice to keep steam in the dust catcher from the time the furnace is lit until gas has been brought over from the adjoining furnace.

(*d*) The old practice of keeping a coke fire burning on top has much to recommend it and may be adapted to modern closed-top furnaces by putting a large bunch of burning waste on the skip, hoisting and dumping it into the hopper, the bells being open. Upon the gas lighting, the bells may be closed and gas led off from the bleeder. This takes more time and trouble than many consider necessary, but it is a practice which is followed by several men and warmly believed in.

(*e*) In case of an accidental shutdown, gas may be brought over from the other operation to the furnace being blown in, and the air having been displaced, the bells and bleeders can be closed prior to shutting down or taking the blast off. This insures a positive pressure at the furnace top under the bell and in the gas-main system.

BLOWING OUT.

CAUSES OF EXPLOSIONS IN BLOWING OUT.

One hazard in blowing out lies in letting the top of the stock column get too low with too high a blast pressure, so that air containing oxygen channels through the coke and forms an explosive mixture with the carbon monoxide and hydrogen of the gas in the furnace stack. The writer personally does not know of any explosion that has been caused in this way, although the possibility of such an explosion is frequently spoken of by furnace men. The enforced use of water, sprayed in at the top of the furnace, keeps the temperature down as the stock descends, and generates a large volume of steam which should tend to prevent such explosions. This dampening effect is, however, counteracted by the decomposition of water by the incandescent carbon in the charge, forming a large amount of hydrogen, which, with the excessive amount of carbon monoxide, makes a very explosive gas.

The composition of a typical "blowing-out" gas, as analyzed by the plant chemist, is as follows:

Results of sampling blowing-out gas.

Hour.	Temperature at downcomer.	Proportion in gas.					
		CO_2.	O_2.	H_2.	CO.	CH_4.	N_2.
	°*F.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
1st	320	12.30	....	2.60	26.73	0.17	58.20
2d	365	9.40	....	2.58	30.58	.12	57.32
3d	585	7.80	....	4.60	31.20	.40	56.00
4th	560	6.90	....	2.46	31.20	.20	58.84
5th	550	4.30	0.10	2.68	30.48	.72	61.72
6th	535	3.30	....	3.62	32.82	.18	60.08
7th	600	3.50	....	5.28	31.82	.18	59.22
8th	615	2.80	....	4.98	32.58	.22	59.42
9th	575	2.70	....	5.60	32.26	.14	59.30
10th	660	4.10	0.10	6.22	29.96	.04	59.50
11th	670	4.90	....	7.94	28.74	.06	59.36
12th	720	3.20	....	7.96	31.60	.00	57.24
13th	550	4.30	....	6.80	29.34	.06	59.50

The temperatures of blast-furnace gases, from observations by the writer, are unusually low, and it is to be mentioned that temperatures taken at the furnace top were from 100° to 415° F. higher than those observed at the downcomer.

The composition of another "blowing-out" gas, as determined by the plant chemist, was as follows:

Composition of gas from furnace being blown out.

[Furnace blow-out started at 10 a. m.; analysis started at 3 p. m. and continued hourly.]

CO_2.	H_2.	CO.	N_2.
Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
5.1	5.1	28.8	61.0
5.5	5.5	34.5	57.5
3.4	3.4	33.6	59.07
3.2	3.2	33.8	59.4
2.6	2.6	33.3	59.4
3.1	3.1	32.5	59.7
5.9	5.9	29.7	59.0

DANGER FROM ACCUMULATIONS OF WATER.

In one instance a gas sample taken by accident just as a furnace made a slip in blowing out contained 20 per cent hydrogen. This points out a second danger in blowing out, that if the stock wedges and hangs, and water is continuously sprayed in, the hanging part may become excessively saturated with water, possibly a pool collecting on top of the stock. When the slip occurs, this water is precipitated into the incandescent coke and walls beneath, and the generation of steam and water gas may result in an explosion. In one case, where two rings of the furnace shell were split in a "blowing-out" explosion, the evidence indicates that this had occurred.

In addition to this danger is the one encountered when the large bell hopper on top fills with water and the bell is dropped or opened at the end of blowing the furnace out. Two instances of this kind are known to have occurred within the past few years. In one of these the bell and hopper were sealed with flue dust and the bell to keep it cool was being sprayed. It was supposed that the seal was insufficient to retain water on the bell, but the unexpected happened. The hopper became full of water, and when the bell was lowered at the end of the last cast, the water dropped into the furnace, in which the red-hot stock was at a little below the mantle and the walls presumably were red-hot. The resulting explosion blew the bell and hopper out of the top and wrecked the top rigging. Several explosion doors on the gas offtakes did not, apparently, relieve the pressure. In the other case the hopper was filled with water by design, in order to keep the bell and hopper from warping, and it was opened by mistake with a less serious explosion.

ACCIDENT FROM DRAFTING GAS FROM DOWNCOMER.

The final danger lies at the end of blowing out, when an attempt is sometimes made to draft gas out of the downcomer and dust catchers. The following description is typical of the manner in which such accidents occur:

After the cast was made the furnace was drafted back until all blowpipes were dropped, when the engines were stopped. Eight tuyères had been pulled and men were working on the ninth when a violent explosion of gas took place in the dust catcher and downcomer, which forced a sheet of flame and red-hot coke out of each tuyère opening. The condition of the gas-main system was: Bleeder on top of the furnace open, little bell open, large bell closed, bleeder on the dust catcher open, and bleeder at the end of the gas line open. The explosion took place between 15 and 20 minutes after the blast was taken off the furnace. Inasmuch as the downcomer, dust catcher, and mains were full of gas, the draft induced by the open bleeder on top caused air to be drawn in at the dust-catcher and gas-line bleeders. Whatever air entered through the dust-catcher bleeder was intimately mixed with the current of gas moving into the dust catcher by virtue of the open bleeder at the end of the gas main. From the high top temperature obtained in blowing out, the brickwork in the top of the furnace was at better than a bright-red temperature, and thus ignited and exploded the intimate mixture of gas and air. From the point of view of plant damage, the most destructive effect of the explosion was in the bleeder, downcomer, and dust catcher. The ejection of hot coke and flame at the tuyère caused the death or severe injury of several men.

This explosion was directly caused by opening the dust-catcher bleeder. Had only the bleeder at the end of the main been opened, the effect of the draft would have been to exhaust all the combustible gas from the system, as the gas, being followed by air from the end bleeder, would not have had as great a chance for mixing with the air. It would probably have been still better not to draft out the mains until the furnace had partly cooled at the top.

METHODS OF BLOWING OUT PRACTICED AT THE 18 PLANTS.

The methods of blowing out followed at the 18 plants for which the proceduce in blowing in has been described (see pp. 130 to 146) are summarized in the following:

PLANT 1.

We use water through the test holes, watching the temperature of the gases, and as the temperature rises we increase the water, and after about six hours' time we drop from one to two revolutions per hour, according to our gas temperature. It takes about 20 to 24 hours to get all the material out above the tuyères. As soon as we find that all the material has been consumed, before taking off the blast we close the main gas valve and open the bleeder. The bells are allowed to remain closed, and also the relief valve on the bustle pipe and the caps on the tuyères. The furnace is allowed to stand for several hours with water on the test holes, but when water is being put on the large bell to keep it cool care is taken that it does not accumulate, and it should be examined before opening the large bell, as a large amount of water being thrown into the furnace is liable to cause an explosion. Care should also be taken that the large bell is kept open before any work, such as taking down the tuyères or blow pipes, is attempted.

PLANT 2.

We see that we have a sure supply of water for blowing down and good pumps for same. The further preparation is to place ¾-inch water pipe in each try hole at the top and a 1-inch pipe in the receiving hopper. The bells are kept so as to not close tight, but to allow the water put on the bells to pass through. No stock is kept on the bells while blowing down.

While blowing out the furnace is checked every hour and the blast is reduced as the stock goes down. About 15 hours is needed to complete the work. The water is started to pump into the top about 1 to 1½ hours after we stop charging, and this is generally at a time when we can finish the "blowout" in a day's time. In order to keep the top gas from getting too hot we put a regular charge of stock into the furnace every 2 to 2½ hours until about 4 hours of the furnace being out. We keep watch of the top heat, and if the temperature goes up to 700° to 800° F. we add water by the use of the skip, half a skip at a time. The pumps are so arranged that at casting time we can stop and open the valve, thus permitting all water to run out of the pipes three to five minutes before we take the blast off to close the iron hole. The gas while blowing out is used as much possible and the balance goes to waste. When the furnace is down close to the tuyères, which can be told by its appearance, the valve which cuts off the gas from the other furnace is closed. The water pump into the furnace is taken off and the bells, bleeders, and explosion doors opened before the blast is taken off. Dust catchers and gas mains are kept closed.

As soon as the blast is off we take down a couple of blowpipes, pull the tuyères, and pull coke to get a couple of good holes through the stock to burn all gas produced in the stock in the bottom. As soon as this is done, we take down all blowpipes, warn all men away from the blast furnace, and start the pump to drown out the furnace, all the openings in the top of the furnace being kept open. After 12 or 15 hours of pumping the furnace is in condition to dismantle.

PLANT 3.

A 2-inch water line is run to the top of the furnace with four connections to regulating valves on same and all controlled by a master valve at the bottom of the furnace, and three spray pipes are put into the gaging holes, one in each hole, with the holes in the pipes turned toward the center of the furnace.

In the big bell we place about 10,000 pounds of good coarse ore around the apex. The bell is clamped closed and all valves or dust catchers left in running position.

After the top is put in condition, as stated above, the blast is turned on and a small amount of water is sent through all four connections. As blowing out proceeds, the volume of air is reduced as conditions warrant and the quantity of water increased, so that the gas temperature approximates 900° C. or less. When the stock gages about the top of the bosh, the bleeders and explosion doors are opened, and shortly after that the valve at the dust catcher is closed, and all gas led out of the top of the furnace.

The furnace is now cast and blown as dry as possible, leaving about 8 feet of stock about the tuyères, the blast taken off, and the gas drafted back through the stoves, all explosion doors and bleeders on top of the furnace opened, and a full head of water turned in on top; and the furnace is let stand for 12 hours. Then the blowpipes are removed and the tuyères pulled, what little stock is left in the furnace is raked out, the furnace now being cold, and the big bell is lowered.

As a general proposition our big bells leak enough to permit all the water from the circle pipe at the apex of the bell to run into the furnace. If such is not the case, the bell is opened and closed against two pieces of ½-inch round or square steel supported from the gas-sealing hood doors, so as to permit the water to run into the furnace, but this is a rare condition.

When the blast is taken off the furnace for casting, the water is shut off at the top until the blast is again put on.

PLANT 4.

Our procedure is as follows: Start about 10 a. m. Take 12 per cent of the weight off the furnace; take one-half the lime content off the burden every second charge for eight times. At 2 p. m. let the furnace work down slowly to about 32 feet; put in one charge about every half hour to keep the top cool. At 3 p. m. reduce the blast about 14 to 17 per cent. When furnace is down to about 32 feet, put in 20 skips of coke, then put 2,000 pounds Mesabi ore on the big bell. At the next cast hold the blast off the furnace to get spray on the small bell. The spray on the small bell is 1-inch pipe bent in a circle about 2 feet in diameter; the holes in the pipe are three-eighths inch in diameter and are 3 inches apart. Also put in two spray pipes, one on each side of furnace, in test holes; these sprays are made of 1-inch pipe, 11 feet long, one end of the pipe being plugged. The holes in these sprays are one-eighth inch in diameter and about 1 inch apart, and there are three rows of holes extending up 5 feet from the bottom end of the pipe. These sprays are connected by a 1-inch hose to a manifold placed on top between No. 1 and No. 2 stoves. There is a 2-inch pipe leading from a 150-pound pressure line to the manifold. This manifold has connection for spray on bleeder stack or for other sprays, if needed.

As soon as water is turned on, the bleeder is opened wide, and the 1½-inch valve on the connecting sprays is opened two full turns of the wheels after the blast is turned on the furnace. The water is turned off in the sprays about three minutes before the blast is taken off at cast time; the furnace is then gaged with a long wire with a weight on to see how far down the stock is in the furnace.

After the blast is on the furnace the water is turned on in the sprays. When the stock is down about 2 feet below the mantle the last cast is taken from the furnace, the blowpipes taken down, and the tuyères plugged with clay; then the gas valves are closed, the large bell opened wide, and the bleeder is left open. The sprays are then taken off and a full head of water is turned on the furnace. The tuyères and coolers are left in the furnace until the gas has died out and water is running out of the tuyères and coolers. The tuyères and coolers are then removed.

PLANT 5.

After the last charge the stock is covered with about 10 tons of coke dust, and 1½-inch water pipes are placed in the four try holes in the top of the furnace to keep the top heat down. This water runs steadily up to 24 hours after the furnace is out. The bell is kept shut and is covered with wet ore to help keep it cool. The bleeder is kept shut until the top temperature gets high, when the bleeder is easily opened.

After the stock is covered with coke dust the wind is gradually cut by slacking the engines one revolution, or about 300 cubic feet every 20 minutes, down to half blast, which is reached in five or six hours. One engine alone then maintains this half blast until the furnace is out. It would be safer to keep two engines on, but they could not run slowly enough to deliver the small amount of air necessary.

Just before the last cast, when the stock level is at about the level of the tuyères, the iron runner is removed. The last cast is then taken out through a hole in the hearth jacket, 18 inches below the iron notch, for the purpose of reducing the size of the remaining salamander. After the last cast, and while the blast is still on the furnace, the bell is opened and is left open, and the downcomer valve is shut. The wind is now taken off the furnace, the blowpipes taken down, and the remaining blowing engine stopped. The door in the downcomer, just over the downcomer valve, and the valve in the bottom of the dust catcher are now opened, and any remaining gas passes up and out of the top of the furnace.

After blowing out, the furnace and stoves are roped off for at least 24 hours, to keep the men out of danger of possible explosion.

A furnace was blown out according to the above schedule. The gas was quiet; there were no puffs nor any explosions whatever. The stock level was blown to below the tuyères and a salamander about 8 inches thick was all that remained.

PLANT 6.

Perforated water pipes, 1-inch, placed in two of the test holes on opposite sides of the furnace, with valves controlling the flow at the bottom of furnace. The furnace is tested with a rod at intervals to make sure that the stock is descending regularly. When the top temperature reaches 900° to 1,000° F. water is turned on and allowed to spray into the top.

The air blown per minute is reduced as the stock descending and the condition of gas on top necessitates. When the stock level is in the neighborhood of the mantle, the water-seal valves on each side of the main dust catcher are closed and the bleeder opened. When the stock is level with the tuyères the last cast is made, and when the snort valve is opened, both bells are opened and the water shut off.

One man is sent around the furnace to loosen the caps on the penstocks and no one else goes near the furnace. After everyone is away from the furnace a large volume of water is turned in and the furnace is allowed to stand under these conditions for at least 18 hours.

PLANT 7.

The stock level is lowered and about a carload of boiler-house ashes put in. Chains or cables are hung from the lugs of the large bell hopper to keep the bell open, making it impossible for a large amount of water to collect on the bell. Perforated pipes are run down the try holes to wet the stock and keep the top cool. A pressure gage is put in the main water-feed line to insure against the stoppage of a sufficient flow of water. The blast is reduced as the top temperature rises, and it is the aim to keep this temperature below 800° F. When the gas gets in such a condition that it does not burn freely in the stoves, or trouble is experienced in the gas-engine room, the water seal between the furnace and the overhead main is filled and immediately steam is put into the dust catcher between the furnace and the water seal. All bleeders are closed with the exception of those on top of the furnace, and the gas is thereby driven up through the top of the furnace. After the furnace is cast, and before the blast is taken off, both bells are lowered. The blast is then taken off and the blowpipes are taken down one at a time by one gang of men, in order to minimize the danger. The furnace is then roped off for a period of 48 hours.

PLANT 8.

Water is introduced by means of spray pipes through the try holes as soon as possible after charging has stopped. The sprays are made from ¾-inch pipe with staggered holes punched over an area of about 10 feet of its length with the ends of the pipes closed. The sprays are connected with a high-pressure system, thus insuring adequate water at all times.

After the stock has settled to about 20 feet the blast is taken out and about 30 tons of coke dust thoroughly saturated with water is charged into the furnace with enough limestone to flux the ash. During the time the coke dust is being charged, no water is allowed to enter the top of the furnaces until the blast is again turned on. The furnace is gaged every hour or oftener, although if the contrary is found to be the case the blast is reduced and the furnace is slipped in the usual manner. The blast pressure is gradually lowered to meet existing conditions. Practically full blast is maintained during the "blow-out." As soon as the furnace shows signs of getting wild, the volume of blast is reduced.

The amount of water is increased to keep the top temperature as low as possible, and at no time is it allowed to get above 1,000° F. The gas is regulated at the stoves to meet conditions. The water seal is filled immediately after the final cast is finished and the tapping hole is left open. Before the blast is taken off, both bells and bleeders are opened. The blowpipes are taken down and the tuyères withdrawn right after the blast is taken off. The water is turned off at the top while this is being done. As soon as the tuyères are all removed, water is put at the tuyère arches and again turned on top of the furnace through try holes. The water is left on until the stock is thoroughly cooled.

As soon as the blowpipes are disconnected from the bustle pipe, engines are shut down.

PLANT 9.

When the blowing out of a furnace is not extemporaneous, it is planned to start operations at such a time as to allow their completion on an afternoon of a day turn. Plenty of time is taken to make each operation completely and carefully. The ore burden is decreased for 12 hours previous, depending upon the heat margin on which the furnace has been working.

The limestone burden is decreased 12 to 24 hours previous to putting in the last charge, 300 to 500 pounds, depending upon the size of the charge and the general working conditions of the furnace. In the case of a thin-lined furnace, the temperature in the water buckets would be allowed to increase to near the boiling point, but not to it. All steam lines to the dust catchers, etc., are thoroughly tried out previous to starting, to make sure that there will be a maximum flow of steam provided when required.

The last two or three charges are made up of ore alone, and for several previous to this about one-half of the limestone is taken off, or the stone is left off entirely for two or three charges. Shortly after the last filling has been put in, four 1¼-inch pipes about 10 feet long are inserted into the top of the furnace through the try-rod holes, the blast being taken off meanwhile. These pipes are drilled with ⅛-inch holes, and the holes pointed toward the center of the furnace. A gage is put in the water line at the bottom of the furnace so that an idea may be obtained from the pressures as to the relative amount of water being used. While these pipes are being put in the receiving hopper, all of the filling apparatus is examined to make sure that there is not an undue amount of material accumulated which may wash down later onto the large bell and seal the small opening between the bell and hopper, which will be made at this time by inserting ½-inch or ¾-inch flat bars between them and closing the bell.

Blowing down is then continued and the top heats kept under 800° F. by means of water through the spray pipes, and small amounts of water at short intervals, if necessary, through the opening between the bell and hopper, the water being conveyed to this point by means of the skip cars. The volume of blast is reduced meanwhile as the pressure falls off and is lowered if necessary to assist in reducing the top temperature. Only enough heat is supplied to the blast to keep the furnace from working cold. Two or three hours before blowing down is completed it is made sure that the dust catchers are all empty.

Steam is turned on for three or four hours before the blast is taken off, and when the quality and quantity of the gas is such as to affect any of the operating points materially the water-seal valves are filled and the mixture of gas and steam blown out through the furnace bleeders. After these valves have been filled, everything between them and the top of the furnace is kept closed.

When it is apparent that blowing down is practically complete (as shown by coke just above the tuyères) the blast is reduced slowly, and finally some of the air is permitted to escape through the snort valve. Then the bells are closed, after which the blast is taken off and the hot and cold blast valves closed.

One man is then assigned to open the caps on the tuyère stocks, after which the engines are shut down. All passageways to the cast house are then closed and kept closed until the morning of the second day following (about 36 hours), when the blowpipes are dropped and a few men put to raking out the coke, no other laborers being permitted around the furnace.

A noticeable quantity of steam will be kept coming out of the furnace top until enough coke has been raked out at the bottom to allow free burning of any gases formed. After this has been assured other labor will be permitted around the furnace.

PLANT 10.

When preparing to blow out, we lighten burden to a 1 to 1 ratio, and take off 25 per cent of the limestone during the last 10 rounds. Before the last four rounds are filled, we charge two rounds of coke only, and end by charging in the same way. We then shut down and put 1-inch spray pipes in the test-rod holes and around big bell. The spray pipes in test-rod holes are drilled with two rows of holes to direct the water toward the center of the furnace and keep it off the walls.

The blast is continued at the normal number of revolutions until top heat becomes 800° F., and then blow enough less air to keep the top heat at not over 800° F., with water going through spray pipes at a pressure of 75 to 100 pounds. Enough gas is wasted at bleeders to keep the gas pressure less than 8 inches of water. All air is cut off around the gas burner on the stove and less air than is needed to burn the gas is admitted through the blow-off doors, to keep the stoves from melting in the bottom.

We blow down to a point about halfway between the tuyères and mantle. After making the last cast, we drill in as low as we can in the iron hole to drain as well as possible.

As soon as the last cast is made, the blast is taken off the furnace, the blowpipes taken down, bleeders left open, all doors on gas main kept closed, and burners at stoves and boilers racked back tight. The water spraying is continued on top until it begins to show up on tuyères. Then the tuyères and cinder notch are pulled and the furnace raked out through the tuyères and cinder notch. The gas mains from the furnace top to the ends are kept closed until all material is out below the tuyère level and the furnace is fairly cool.

PLANT 11.

When ready to blow down, we have perforated water pipes in try holes. We aim to clean out coke pockets, and use the small coke with converter cinder to blow down on, or in other words this is the last material charged. All flue dust is drawn out of the dust catcher and all valves closed as tightly as possible. As the stock in the furnace keeps going down, we turn water (through the perforated pipes) into the top of the furnace and keep increasing the volume as the top temperature rises. When furnace gets below the reach of the test rod, a cable with a weight is used to find out how far down the stock is. This is done every hour. We analyze the gas when blowing a furnace down and when it is found that the gas is not usable, the furnace is cut off from the other gas systems by the water seal and its gas blows out of the bleeders. Steam is turned into the dust catcher and gas main at this time. When the stock is down nearly to the tuyères, the tapping hole is opened, the iron taken out, and the hole left open so all cinder possible may drain out. The blast is then taken off the furnace and both bells opened. After the blast has been off for one hour, the caps of three or four blowpipes are pulled back and water hose put into these blowpipes in order to drown the furnace. After three or four hours, the blowpipes and tuyères are removed, one at a time.

PLANT 12.

Before blowing out is commenced the furnace is shut down and sealed off, while the main bell is opened and four ¼-inch bent rods are introduced through doors on the seal and placed so as to hold the bell open ¾ inch when it is closed again. The hoppers above the bell are thoroughly cleaned out and two sprays made of 1-inch pipe bent nearly to a circle, and perforated so as to spray inward and downward, are placed so that one sprays upon the small bell and one upon the main bell. The last charge before shutting down generally carries some extra coke, more or less depending upon the condition of the furnace. Nothing is charged after the sprays are placed and no material is carried on either bell during the blowing out. The blast is slackened shortly after the charging is discontinued. The rate of blowing out is shown below:

Time after charging stopped, hours.	Volume of air, cubic feet per minute.
None (blast at usual rate)	40,500
1	30,000
3	26,250
6	18,750
7	16,875
8	16,500
14½	Blast off.

Enough water is used to keep the top of the furnace below 800° F., and the blast is reduced at any time when necessary to do this. The furnace is checked once an hour to make sure that it is moving. The gas from the furnace becomes wet; it goes into the gas main, however, until the blowout is completed. The bleeders on top of the furnace are often opened at the latter end of a blowout in order to reduce the amount of wet gas going into the main. The furnace is gaged by hand by means of a weight on a long wire clothesline. It has been our practice to blow partly down the bosh but not to attempt to blow completely down to the tuyères, partly because of the possibility of blowing uncombined air into the gas main and partly because a little pressure in the furnace enables some iron to be blown out on the last cast and the level of the salamander lowered.

After the last cast is completed, the engine is slackened to the minimum rate and allowed to turn over against the snort valve; the water seal is closed.

PLANT 13.

The furnace is stopped and spray pipes put on all pipe work on top, where the brick is gone. Pipes are run down try-rod holes, two if possible, one being left open for gaging the furnace. A circular spray is put around the apex of the bell with the bell shut. Blast is put on the furnace and the water regulated according to supply needed to keep the exposed plate work wet and have bell spray going strong. One-inch pipe is used in the try hole, or 2-inch if possible. The engine speed is regulated to keep the top heat below 1,000° F. and the furnace is gaged every 30 to 45 minutes until about 70 feet is reached. Casting and flushing are continued, with an effort to have the furnace about empty when through with the last cast.

If the furnace does not move regularly, it is checked, and if serious hanging takes place, say for four hours, the blast is checked, but as a rule any "scabs" come off as the furnace empties. The last cast is made when the furnace is judged to be about empty, and the iron notch is kept open until the stock gets down to the level of the tuyères. The blast is taken off and stoves and boiler-house burners closed. The bleeders are left open and the bell is left closed for at least six hours.

The gas is drawn back through the stove, the blowpipes are taken down, four coolers taken out, and the stock raked out. Water is left running into the try-rod holes and onto the bell for about six hours, then shut off. The furnace is then considered blown out.

PLANT 14.

Water connections are made on top to give about six ½-inch lines running about three-fourths full. We have never estimated the amount of water exactly, but use sufficient to keep the gas and top cooled down. If the furnace has been working irregularly and given signs of a scaffold, the coke blank on the last period of filling may be 300,000 pounds, but if we are not uneasy about the furnace, this may be reduced to 200,000 pounds. All water valves, etc., on the lines are examined to see that they are in good condition.

After the coke blank is put in, the large bell is propped open about three-fourths of an inch. The small bell is kept closed and the bleeder stack is opened. The six water sprayers are now passed into the gas seal and rodding hole. The blast, in the meantime, has been taken off the furnace.

The blast is then put back on the furnace and water permitted to run through the six ½-inch pipes into the furnace. Men are on the stoves during this time watching the top closely. The volume of air is now reduced to approximately 60 per cent of the original amount and is gradually reduced every two hours.

After approximately 20 hours' time the blast will be down to about 10,000 cubic feet per minute, and we can as a rule see across from tuyère to tuyère. In the meantime, when the furnace is beginning to get this low, the tapping hole is opened and the last cinder and iron allowed to run out. The water on the top is then shut off. After as much of the cinder and iron is removed as possible the snort valve on the blast line is opened and the stove valve closed.

The engine is allowed to turn over slowly for some time after this before the tuyères are taken down. After 30 or 40 minutes three of the tuyères at a time are opened at different points around the furnace and plugged, water pipes are placed in on the coke and water is allowed to flow through them. This has the effect of forcing a little more cinder and iron out through the tapping hole.

This may be kept up for two or three hours until no more material can be drained from the hearth, after which time all blowpipes are taken down and the tuyères pulled and the furnace allowed to stand until thoroughly cool inside.

PLANT 15.

Our practice in blowing a furnace out is to place ¾-inch pipe in at least two of the try-rod holes. The bottoms of these pipes are perforated so as to throw the water in the form of a spray underneath the bell, and this has a tendency to keep the top of the furnace cooler than any of the other methods we have tried. There is also a small spray pipe on the top of the bell. A little coarse ore is spread around the seat of the bell and hopper. During the last hour or two that the furnace is being blown out the water seal is closed and the blowing out finished through the bleeders. At least one-half hour before taking the blast off and during the time that the furnace is putting out her last cast we always close the valve on the water line and break the union so as to disconnect the pipe entirely from the furnace; this to make sure that no water is going into the furnace, which would be liable to generate hydrogen gas. After the blast is taken off, one blowpipe is dropped at a time and each tuyère plugged immediately with clay, so that the furnace can not draft any air through the bottom and continue making gas. As soon as all the blowpipes are down and the tuyères plugged the large bell is lowered, and the furnace is then drowned with water. It has always been our practice before taking the blast off to fill the dust catcher and gas main from the water seal to the furnace with steam and we have not had any trouble as regards having any "kicks" from gas explosions.

We always start to blow a furnace out about 9 p. m., so that the night shift can get along without any supervision, and the furnace can be finished in the daytime under the direct supervision of the superintendent. This, we think, is a better plan, as it is a good deal easier to do the work in the daytime than at night.

PLANT 16.

When possible, filling is always stopped at such times as will permit the furnace being blown out during daylight. No change in burden or addition of coke is made in the last filling. All gas is directed into mains in the usual manner until the gas becomes wild, owing to the addition of water for cooling the top. Excess gas is taken care of through bleeder valves. The blast is reduced as the stock line falls. When low pressure, coke, cinder, time elapsed, etc., indicate that little stock remains in the furnace, the gas valves leading from the dust catchers into the gas mains are closed and the gas from the furnace being blown out is allowed to escape from the bleeders until the process is completed.

The furnace top is kept cool by water sprays, and the main sealing bell is kept closed or only slightly open in case it is considered essential for saving the edge of the bell projecting below the sealing point on the hopper. A hole is usually drilled in the apex of the bell and a pressure spray pipe inserted for additional cooling of the bell from inside. No water other than from this pipe, or from the external cooling of the bell and top, is permitted to go into the furnace. Special precautions should be taken to see that there is no accumulation of water on the furnace top, as the precipitation of any considerable quantity into the incandescent crucible or bosh will cause an explosion which may be disastrous.

The iron notch is usually left open after the last cast so that the remaining cinder may run out as it is melted down during the last stages of blowing out. Our usual practice is to blow the furnace down to within a close distance to the tuyères.

PLANT 17.

As soon as the regular filling is discontinued, we charge into the furnace about 15,000 pounds of wet flue dust and dump onto the charging bell about 2,000 pounds of sand or other semifine material that will not bake but will partly close any large openings between the bell and the hopper. Then we take the blast off the furnace long enough to block the bell shut, put a spray pipe around the apex of the bell, and spray pipes about 16 feet long down each of the four gage holes. The amount of water sprayed on the bell is adjusted to keep the bell nearly covered with water. The quantity of water on the gage-hole spray pipes is regulated to keep the gas temperature less than 1,000° F. The amount of air blown is usually reduced about 15 per cent as the furnace is blown down. The furnace is cast every three hours and checked whenever the appearance of gas and other indications show that the top of the stock is blown down to the neighborhood of the tuyères, and the furnace is cast as dry as possible. During this cast the water level in the dust catchers on the furnace being blown out is raised from the regular operating level, which is carried 30 inches below the downtake, to 72 inches above the downtake, which seals off the gas and confines it to the furnace and downcomers with no opportunity for the ingress of air. All peep sights around the furnace are wired shut. After the gas has been cut off in the above manner, the snort valve is fastened open, the cold-blast and hot-blast valves closed, and a blowing engine kept turning over for 12 hours, when the blowpipes are taken down and the dismantling begun. During this 12-hour period the bleeder and explosion doors remain open, no opportunity is given for the admission of air around the lower part of the furnace, and water is constantly sprayed through the gage holes.

Our experience and the attitude observed in others, leads us to believe that many of the obscure precautions in handling blast-furnace gas are apt to be neglected as not essential. It strikes us as pertinent to mention an incident which occurred during our last blow-out. About three hours after casting, after blowing out, a mammoth scab dropped a distance of 30 feet, dislodged several water blocks, and punched a hole in the bosh.

PLANT 18.

The water seal valve separating the furnace from the general gas main or from the other furnaces is closed. Four spray pipes are put down through the rodding holes on the top of the furnace. A rodding device also is reserved in one of the holes in order to determine the movement of the furnace. The little bell is propped open and a spray pipe placed across the top of the gas seal. Coarse ore, gravel, or limestone is placed on the bell. After this preparation the bleeder is opened, the blast put on the furnace, and a little water allowed to flow through the pipes. After the stock goes down the gas will become pretty wild, but this is overcome by allowing more water to enter and by reducing the volume of the blast. The furnace is rodded and checked every half hour. Care is exercised at all times to keep a strong outward pressure on the gas. Three or four hours after the water is allowed to enter the top the stoves are taken off entirely and the gas is taken to the boilers, one boiler after another being taken off until only the last battery remains at the end of the blowing-out period. The furnace is generally blown down until such time as pressure can not be maintained on the furnace, and also, very often, water is noticed dripping down in front of the tuyères. The tapping hole is then opened and the furnace drained, after which the blast is taken off, the main bell lowered, explosion doors opened, blowpipes dropped, the water turned fully on at the top; also water is introduced through four or five tuyères by means of hose and $\frac{3}{4}$-inch pipe, which is pushed into the hearth of the furnace. Water is then allowed to flow until the furnace is cooled down sufficiently to scrape the material out.

OTHER METHODS OF BLOWING OUT.

Two methods of blowing out not brought out in the foregoing are:

1. The furnace is charged with 20,000 to 30,000 pounds of limestone before blowing down is started, or limestone is charged in 300 to 5,000 pound lots at intervals during the blowing down, as the top temperature becomes high. Not only does this keep the temperature at the top considerably lower, but the calcining of the limestone furnishes a certain amount of carbon dioxide, thus diluting the gases and making them less explosive. This is an old method infrequently used at present.

2. The furnace is not blown down at all, but when the furnace is to be blown out the furnace is cast as usual, the blowpipes dropped, and the gas mains drafted out. Then by means of a number of spray pipes or hose water is introduced at the top and the furnace is drowned. The contents of the furnace are then raked out through the tuyère arches, thrown into cars, and dumped in the stockyard, to be recharged in small amounts when the furnace is again in blast.

The method is of course safe, and is claimed to be not unduly expensive. It is laborious, however, and finds a negligible following at present, only two establishments following this method as far as is known.

PRECAUTIONS IN BLOWING OUT.

The essential features of the foregoing descriptions, showing the steps taken and the reasons therefor, are summarized below. No attempt is made to specify procedure or rules, as innumerable points must of necessity be taken care of as they arise and according to the plant layout and furnace design.

1. Before starting to blow down put a water spray on the large bell and three spray pipes through the try holes. The holes in these pipes should be below the lip of the large bell, and be so drilled that the water is sprayed toward the center of the furnace. The spray cools the gas and prevents it from heating the top excessively, warping the bells and hopper, or damaging the offtakes and downcomers; the steam generated also tends to prevent gas explosions.

2. Examine the receiving hopper and the large bell hopper to make sure that not enough material has collected to wash down and seal the opening between the large bell and its hopper. It is inadvisable to seal the bell and hopper with flue dust, ore, or sand when water is used on the bell, as the water may accumulate in quantity on the bell and be dropped into the furnace. It is better to provide an opening between the bell and hopper by hanging rods, cables, or chains from the gas-seal doors between the large bell and hopper extension, and then to close the bell against them. This prevents the accumulation of water, provided the bells and hoppers are cleaned before starting to blow down.

3. The bell should be locked, blocked, or clamped in position after these distance pieces are placed, in order to prevent its being opened by mistake. Also block or fasten shut the dust-catcher bells, bleeders, etc., to prevent any possibility of gas drafting up through the downcomer in case the blast is taken off unexpectedly.

4. Provide a water-pressure gage on the supply line to the top sprays, at the cast-house floor level, so that an idea may be had of the adequacy of the pressure and supply of water. It is also a good idea to provide a water-escape valve on a tee at the same place, so that the line may be drained at casting, or when it is essential to stop the water going into the furnace. Such a valve removes any uncertainty as to the positive cut-off of the valve controlling the water supply. A water meter is used on the spray-supply line at a few plants, and is an excellent idea.

5. Keep the top temperature at about 800° F. This temperature has been shown to require approximately the correct amount of

water to prevent the gas from becoming wild, provided the blast is reduced rationally, and at the same time prevents water accumulating on top of the stock where it may cause an explosion in case of a heavy slip or fall.

6. When the blast is taken off for any purpose, or when it is necessary to check the furnace, the water should be shut off 5 to 15 minutes before this is done, and the relief valve at the bottom of the water-supply line opened to insure that no water is going in.

7. It is safest to keep both engines on the furnace in reducing the blast as long as they can be kept on without danger of stalling. When it becomes impossible to run two engines slowly enough and one engine alone can be kept on the furnace, it is advisable to keep the idle engine in readiness in case of any accident to the one in operation.

8. When the gas becomes too poor or too wet to burn, or the gas engines start to backfire, close the valves between the dust-catcher system and the gas mains. If sand or mechanical valves are used, it may be advisable to take the blast off at the next to the last cast and close them at this time rather than wait until the last moment, inasmuch as many types of these valves require considerable work, which must be done in proximity to gas, to get them closed. If no valves are provided, close the stove and boiler burners, unless the bleeder capacity is inadequate. Many furnace men make a practice of using the gas through the stoves until the furnace is down, but there seems to be no especial advantage in this. There is little heat value in the gas and the time of closing the valves is postponed until the last moment, when as a rule there are many other important details to be attended to.

9. See that all water connections at sprays, valves, and other equipment are in working condition, not plugged with dust, and that all dust bells, bleeders, and doors are secured in position before starting to blow down. About half an hour before the gas is shut off, the dust catcher should be emptied, as red-hot material may have blown or been thrown over into it from the furnace slipping, and if permitted to remain there may cause an explosion.

10. The furnace should be rodded every half hour by a trustworthy and experienced man and should not be allowed to hang longer than half an hour, and an excessive amount of water should not be put in the top while the furnace is hanging. If the furnace has shown evidences of being scaffolded, it is a good plan to insert shorter tuyères a few hours before blowing down, or to lighten the burden, reduce the proportion of limestone, or charge a coke blank. Little is gained by rapid blowing out. The stock descends more regularly and less difficulty is encountered with wild and excessively

hot gas if the engine speed is slackened 20 per cent at the start and the number of revolutions lowered progressively while blowing down.

11. Charging a carload of coarse coke dust on top of the burden aids in keeping the temperature of the gas down, as this interposes a blanket of permeable material which becomes thoroughly saturated with water from the top sprays.

12. Blowing out should be started so as to bring the last cast in on the morning of the day following.

13. When the stock line is at the top of the mantle or about 2 feet below it the furnace should be cast. Blowing down to the tuyères is hazardous, because unburned air may be blown into the space above the stock and form an explosive mixture, and is unwarranted because the hearth will not, as a rule, drain well, and the top of the salamander will be higher when there is little pressure in the furnace on account of there being no stock above the tuyères.

14. Be sure that no water is going into the top sprays at the last cast; that no bells, bleeders, burners, explosion doors, or manhole doors are open on the gas-main system below the furnace top when the last cast is made. Examine the bell to insure that there is no water on the big bell to be thrown into the furnace when the bell is lowered.

15. After the last cast is made open the snort valve, close the cold-blast and hot-blast valves and open the chimney valve. The mixer valve should have been shut at the beginning of the cast. Fasten the mixer valve shut and the snort valve open. One man, under the direction of the foreman, should then knock the caps off the tuyère stocks. When this is done and the men are out of the cast house turn the sprays on full to drown the furnace, and open the large and small bells.

16. If the furnace is blown out for reasons other than relining, and preservation of the lining is essential, it is sometimes advisable to keep the caps on, thus eliminating any chance of a draft up through the furnace, causing a hot gas flame and heating of the lining.

17. The blowing engines should be kept turning over until the blowpipes are dropped. These should not be dropped until 18 to 24 hours after the furnace has been taken off, and every one should be kept out of the cast house during this period, as a scab may fall and break the bosh or throw hot material or flame through the tuyères. The gas should be drafted out of the dust catcher before work is undertaken about the bosh openings or on top.

18. Do not attempt to draft out the dust catcher and downcomer until the furnace top is cooled off. This may be four to six hours after casting, provided that the sprays have been turned on at the top in the meantime to drown the burning coke and cool the furnace walls. Sometimes in blowing out the pyrometer may indicate a

temperature of 600° or 700° F. in the downcomer or dust catcher; when the offtakes and interior of the furnace may be red hot and time must be allowed for the furnace to cool before the gas is drafted up through.

19. In drafting out, open only the door or bell farthest from the furnace top in the usual line of travel of the gas. Sprays should be kept on full at the top while this is being done.

20. If the gas-main system is equipped with a steam connection, steam should be turned into the system from the moment one stops using the gas until the gas-main system is full of air after the blowing out is complete.

EXPLOSIONS IN COLD-BLAST MAINS.

There have been few blast-main explosions during the past few years. Such explosions are very rare, but, as the possibility of their happening is always present, mention is given them. When these explosions do occur they are of a most violent character, usually extending back into the manifolds and blowing tubs at the engines. As descriptive of what may occur the following incidents may be taken as typical:

TYPICAL EXAMPLES.

At one plant the practice was followed of taking the blast off the furnace entirely at casting time, when the tapping hole was closed, the hot-blast valve also being closed to prevent gas from working back into the stove from the furnace. The furnace cast at the regular time, and as the tapping hole was closed, the blower signaled the engine room to start, and the stove tender opened the hot-blast valve. No air was turned into the furnace, however, and the blower ran to the blowing room, which was close to the furnace, to see what was the trouble. Before arriving he found that the work of adjusting a part on the blowing tub was unexpectedly prolonged, and he at once ordered the hot-blast valve shut. Upon entering the blowing room he smelled gas and as soon as possible gave orders to the stove tender to open the chimney valve on the stove to draw the gas undoubtedly present in the cold-blast main out into the stove chimney. He then returned to the blowing room, but had hardly entered it when an explosion occurred. The windows were shattered and doors blown off their hinges. The three-eighths inch steel plate of the cold-blast main was split and the air valves and top of the blowing engine blown off. Three men were killed. This explosion was caused by opening the hot-blast valves before the engine started, thus making it possible for gas to draft back from the tuyère through the stove and into the cold-blast main to the air tub. The gaseous mixture probably ignited in the hot stove and burned back through the main to the engine room where it exploded.

A similar explosion occurred at a plant still using pipe stoves. At a furnace stop to change tuyères the usual practice was to stop the engine and open a relief valve on the hot-blast main. For some cause the relief valve did not open, nor was the hot-blast valve closed at the stove. Gas worked back into the cold-blast main, and after the tuyère was replaced and at the instant the engine was started there was an explosion in the cold-blast main, which split or bulged it at several places and blew the manifold off the air tub.

Two other explosions have been noted at plants having modern equipment. In each case there was lack of definite knowledge as to the circumstances surrounding the explosion. In one of these the men in the blowing room were killed. It is thought that a heavy slip may have stalled the blowing engines through gas surging back in the hot-blast main and stoves or may have even started them turning backward, as the pressure chart showed that the pressure suddenly increased and then fell rapidly.

In the second case it is stated that the blowing engines were running slowly—at about 15 revolutions per minute—when the explosion occurred. On account of the apparent impossibility of gas working back in the main with a somewhat vigorous volume of air going countercurrent to it, this statement of the condition of affairs has often been viewed with incredulity. It is useless to speculate about it, but an incident which occurred at another plant may throw some light on it.

The blast was on full and the furnace had been running regularly for two hours after casting. The stoves were of small capacity, were being forced very hard, and the temperatures at the chimney stack were excessively high. The cold blast entered the stove at the chimney stack. Soon after a fresh stove had been taken off gas and put on blast the cold-blast main was discovered to be red-hot. The night foreman shut the blast off, slowed the engines down till they were only turning over with the snort valve open, and ran to the office and called up the superintendent, who ordered him to put the engines up to full speed immediately. By the time this was done the cold-blast main was at a bright red heat nearly to the engine room. By running the engines at full speed the danger of an explosion was averted. There was no oil drain on the cold-blast line, and it was found that oil had been blown over into the base of the stoves, where the mixture evidently had ignited and the flame had traveled back in the cold-blast main. With a current of air, after the blast was taken off, the oil lying in the main naturally burned vigorously, and it was possible that an explosion in the oil separator near the engines might have resulted. The high velocity of the air current, when the engines were run at full speed, blew the fire out.

Fires in the cold-blast main caused by the furnace slipping and gas surging back through the mixing valve are not uncommon, but the writer does not know of any having resulted in an explosion.

Another explosion occurred at a plant where the by-pass led from the end of the cold-blast main at the back of the stoves to the hot-blast main between the furnace and adjacent stove. The stove tender forgot to close the by-pass mixer valve at casting time, and as the butterfly valve in the cold-blast main closed tightly gas from the furnace evidently drifted back into the cold-blast main, where it exploded with a sharp detonation, not doing much damage, however.

PREVENTIVE MEASURES.

To prevent explosions or fires in the cold-blast main there are three elementary precautions:

(1) Always close the mixer valve at the first check on the furnace; (2) provide oil drains at the blowing rooms, and in close proximity to the stove, on the cold-blast line, and then (3) at a stop of any description always keep the engine turning over with the snort valve open until the blowpipes are dropped.

Other precautions that may be taken, in addition to those given above, are as follows:

Provide a steam connection with the cold-blast line near the engine manifold, which may be turned on from the floor in case a fire develops in the cold-blast main, and if the engines are accidentally suddenly stopped close the air valves at the engine at once.

In case of accidentally "loosing the blast" on the furnace close the hot-blast valve and immediately open the chimney valve of the stove on the furnace. Drop the blowpipes, drafting back through another stove.

Install an automatic check valve in the mixer or by-pass line between the hot and cold blast mains.

Provide a check valve at the junction of the cold-blast main with the air-tub manifold, or, if the blowing engine is of the type which will turn over backwards from excess blast pressure, install an automatic stop or check device to prevent this.

CHECK VALVES.

Check valves should be of the type that gives external indications as to whether it is working properly or sticking. All check valves, with two exceptions, in which the valve is operated by an actuating piston and cylinder, are capable of being freely opened by the pressure of the entering blast, so that when the cold blast is by-passed to the hot-blast main by opening the mixer value the check valve remains open. When the blast is cut off from the by-pass line the valve automatically closes of its own weight. In addition, if air and gas surge back in the by-pass or mixer pipe, the valve is thrown against its seat.

Check valves are of two general types, a completely inclosed swivel clack valve with a horizontal turning joint and a valve which with the blast on, closes a vent to the atmosphere, but with the blast off or at any predetermined pressure, such as 4 pounds, closes against back travel of gas and opens the by-pass main to the atmosphere. The difficulty with some valves of this latter type is that they may not close automatically against a back surge of gas when a heavy slip occurs at 10 to 12 pounds blast pressure; that is, when the furnace is being checked.

OTHER AUTOMATIC VALVES.

There are a few additional appliances in use for preventing a back surge of gas reaching the cold-blast main; chief among them are the Schutte-McHugh valve for the hot-blast main and the Dyblie-Larimer appliance for hot-blast valves.

The Schutte-McHugh valve is used to prevent back-flow of gases from the furnace to the hot-blast stoves and cold-blast mains. It consists of a valve casing, located in the hot-blast main, with a water-cooled valve seat placed at an oblique angle from the vertical, in the direction from which the blast travels. The valve is also water cooled, the pipe extending out through the valve stem. One end of the stem is provided with a counterweight lever; on the other end is a lever connected to a piston, the piston-cylinder being fastened to the valve casing. This cylinder is connected by a pipe to the cold-blast main on the engine side of the snort valve. With the blast on the furnace, the pressure transmitted to the cylinder keeps the valve from being actuated by fluctuations in the pressure of the blast. When for any reason the blast is reduced as at checking or casting, the valve will gradually approach the seat, and when the blast is entirely off the furnace, the valve will rest on the seat; when it is in this position a back flow of gas from the furnace through the hot-blast main can not pass it and enter the stoves and cold-blast main.

When the gas must be drawn back into the stoves for repairs to the furnace, the valve may be held open by a cable attached to the counterweight arm. If when the furnace is being checked it is desired to have the valve free to act independent of the actuating cylinder and close instantly, a two-way valve may be used which cuts off the cylinder from the cold-blast connection and opens it to the air.

This valve has received some adverse criticism by furnace men, but it is working satisfactorily at some furnaces.

The Dyblie-Larimer valve automatically opens and closes the hot-blast valve as the blast is shut off or turned on, thus preventing gas from passing from the furnace through the stove to the cold-blast pipe at casting time or checking. This valve consists of a cylinder and piston, with a cable from the piston rod to the hot-blast valve stem, and a special piston valve connecting the cold-blast pipe on the stove side of the cold-blast valve to the top of this cylinder. When air is admitted to the stove it raises the valve; then the air pressure on top of the piston lifts the hot-blast valve. The hot-blast valve is heavier than its counterweight, so that when the blast is taken off it closes, the pressure in the actuating cylinder being automatically released by the special valve. When the stove is taken off the furnace a pull on the release trigger of the special valve allows the air in the cylinder to escape, causing the hot-blast valve to descend to closed position, where it can not be raised until the blast is on again. One merit of this safety device is that it should keep in working order because of its being used on routine operations at least twice every four hours.

VALVES IN TUYÈRE STOCKS.

Another provision, sometimes found at old plants, is the placing of valves in each tuyère stock or gooseneck.

As mixer valves are sometimes neglected and not closed, and because a shutdown may be a matter of seconds, a check valve in the mixer pipe is necessary. The utilization of other automatic valves must be at the discretion of the furnace superintendent. When any automatic or semiautomatic valves are used, it is necessary to see that they are always in working condition, and also that workmen do not fasten such valves open or make them inoperative. Unless this is done confidence in them may prove misplaced.

EXPLOSIONS OF AIR COMPRESSORS AT BLAST-FURNACE PLANTS.

Air compressor or receiver explosions occur occasionally at blast-furnace plants and sometimes cause considerable damage. As they are somewhat analogous to blast-main explosions they are briefly discussed here, although such accidents are not, strictly speaking, metallurgical hazards.

Fires in the receiver tanks of air compressors are not uncommon, and point to the cause of the explosions, as the fire is started by the temperature of the compressed air becoming higher than the flash point of the oil used to lubricate the air cylinder. Although the formula for the generation of heat by adiabatic compression of dry air to a pressure of 100 pounds does not account for sufficient increase in temperature to flash ordinary lubricating oil, it may be that oil oxidizes more readily under pressure than under ordinary atmospheric conditions. This may account for a lower flash point and more vigorous combustion than is anticipated from the grade of oil used. Other factors introducing the danger of explosion are defective discharge valves or piston packing rings. A leaky discharge valve or badly fitting piston rings may let the heated air in the compression end of the cylinder leak past the piston to the suction or admission side of the cylinder to such an extent that the initial temperature of the air to be compressed is so high that the temperature on compression reaches the flash point of the oil.

The precautions are proper upkeep of the valves and rings and the use of a noncoking oil of high flash point. Too much oil should not be used, and an oil drain should be led from the air receiver to remove deposits or accumulation of surplus oil carried over. In some cases a steam line is led into the receiver, this being available for extinguishing a fire, should one start, before it develops into an explosion. On no account should kerosene be thrown into the air inlets to clean the walls of the compressor cylinder. The intake should always be from outside the engine room, as the lower the initial temperature of the air the lower is the temperature reached during compression. The most recently installed air compressors are, as a rule, of the two-stage type, and this is in itself a safety provision, as compression in two stages results in a lower temperature than one-stage compression for a given pressure.

EXPLOSIONS OF MOLTEN IRON.

Explosions of molten iron are due to the sudden and confined generation of steam when the molten iron comes in contact with moisture. A secondary cause may be the formation of hydrogen by the action of incandescent iron upon water, the hydrogen in turn exploding in contact with air. There are some peculiar features about the action of water on molten metal. For instance, molten iron may be run into water and be granulated or made into shot without any explosion, provided the water is deep and in large quantity. However, a blue hydrogen flame always appears upon the surface of the water. If iron in great excess enters water, however, a violent explosion takes place. This sometimes occurs in dumping ladle cleanings into quenching pits or in dumping cinder ladles into water at the bank of a river, if there is a pocket of molten iron in the bottom of the ladle. Other illustrations are the overturning of a ladle of iron into a pig-machine cooling trough, ladles filled with iron breaking away from an engine and overturning at the dead end of a hot-metal spur track into an accumulation of water, ladles being derailed into a river, or the collapse of a bridge and plunging of ladles filled with hot metal into a stream. The resulting explosions are terrific and destructive.

If molten iron runs over a damp spot in an iron runner, frequently great quantities of hot metal are blown out, the amount being entirely disproportional to the amount of water present. On the other hand, provided sand runners or beds are evenly tamped, hot metal may be run in them with impunity, even though they are much damper than can be permitted in a clayed, loamed, or sanded iron runner. A wet or even a cold bar touched to the surface or plunged beneath hot metal causes a snappy explosion; often dangerously large amounts of metal are thrown out. If a freshly loamed hand ladle is plunged into or filled with hot metal, the entire contents may be explosively ejected. Hot metal spilled on a damp or wet floor, whether of iron, brick, or packed soil, will explode very violently. The turning of water into the skimmer trough onto the skull of iron and pockets of molten iron left after casting in the bottom depressions rarely causes an explosion. Occasionally, however, if a fragment of solid cinder is lying on top of a small pool of hot metal, the force of the stream of water may carry the wet piece of cinder down into the molten iron. The inevitable result is a violent detonation, and serious injury may follow. In one instance a man lost both eyes from flying iron from such an explosion. Usually water can be sprayed on the surface of hot metal without an explosion, although such procedure is, of course, hazardous, as is illustrated by accidents resulting from attempting to chill iron leaking from ladle cars, the water penetrating beneath the surface and causing an explosion. Other causes of explosions are when hot metal is poured into a ladle with a wet bottom, or into one containing large lumps of cold wet scrap, or when cold or wet heavy scrap is placed in the runners to be melted at cast, or when at a hearth breakout hot metal comes out in contact with water, or when at the end of the cast a large amount of iron escapes from the skimmer and plunges into the cinder granulating pit, or when water-cooled plates at the tapping hole are cut by iron at casting time.

The cause of these explosions is the sudden generation of steam under confinement. When the steam is free to escape, as downward in sand beds or upward from the surface of the hot metal, there is no explosion. If the steam is generated within the hot metal, then it breaks through the overlying iron, as it does through any insufficiently strong barrier, and throws the hot metal with great force and with a loud detonation. A similar action results when a quantity of molten iron is plunged into water, the water preventing the escape of steam. Analogous to hot-metal explosions are explosions of the fluid contents of cinder-ladle cars when the cinder is run in on top of wet rubbish, particularly if iron is carried over with the cinder and has accumulated on the bottom. Infrequently, as at a breakout, hydrogen gas is generated and bursts into flames, adding to the danger and destructiveness of the explosion.

The precautions against such explosions of metal and cinder are well known and as a rule are carefully followed. The furnace crew occasionally trangresses the rules and some one uses a cold or wet bar or hand ladle, puts scrap in the runners or iron ladles, throws rubbish into cinder ladles, stands too close to the skimmer trough in turning the hose on at ridup, allows too many ladle cleanings to rush into the quenching pit, or is careless in skimming. Where plant or equipment design can provide against hot-metal explosions, they are adequately guarded against as a rule by design of ladle quenching pits, skimmers, catch basins, iron and cylinder ladle cars, hearth construction, etc. These safeguards have been fully described under "Blast-furnace breakouts" on pages 41–78.

TUYÈRE EXPLOSIONS.

Explosions at the tuyères are among the most unpreventable and unforeseen accidents encountered at blast furnaces. The cause is obscure, but lies essentially in the sudden generation of steam from the failure of a tuyère in consequence of faulty water cooling. The forming of a crack or hole in a tuyère does not necessarily result in an explosion, as leaking tuyères are common. Even with nearly all the tuyères leaking there is usually no tuyère explosion, the water leaking into the furnace being entirely harmless as far as explosion of steam is concerned, except under peculiar circumstances. Usually the only result is to "iron up" the tuyères with low-carbon iron and slag, as the formation of steam and hydrogen merely abstracts heat and chills the furnace, the gases escaping up the stack. Evidently steam must form suddenly and not escape immediately in order to cause a tuyère explosion. This may occur in two ways: First, if the tuyère fails and a large amount of water impinges upon incandescent material inside the furnace; or second, if molten iron penetrates suddenly into the tuyère. The first may take place if the tuyère is suddenly broken open, but even then, with water suddenly discharged into the furnace, explosions do not always take place. The second cause is lack of water, permitting bubbles of steam to form in the tuyère nose, allowing the walls to melt. Other factors are uneven expansion, sudden bosh slips, deposits of mud or lime, and slagging of the wall about the tuyère.

For an explosion to take place it seems to be requisite that molten metal meet the water when the nose of the tuyère collapses, or rush back into the tuyère, as tuyère explosions usually develop shortly before casting time, or when iron is lying in front of or over the tuyère.

The violence of some of these explosions is almost incredible. Instances have been related of a tuyère being blown out and breaking a cast-iron water main, and of a cinder cooler being blown a distance of 300 yards. Several men have been killed from tuyère explosions, one plant having two such accidents within five weeks of each other, and innumerable injuries, such as burns, ignited clothing, and bruises, have been caused.

The best and virtually the only safeguards are careful supervision of the water supply, strict watch for leaks, and, especially, regular furnace practice, as tuyère explosions are especially characteristic of rough and irregular furnace work. The essential precaution lies in not standing about any more than is necessary in line with a tuyère which is "sloppy" or leaking, especially at casting time.

BLASTING OUT HEARTH BOTTOMS AT RELINING.

HAZARDS FROM THE USE OF EXPLOSIVES.

The only noteworthy use of dynamite about blast furnaces is in blasting out "salamanders" in the furnace hearth bottom at relining. In former years the use of explosives was sometimes resorted to in dislodging scaffolds in blast furnaces. This practice is discussed in detail in Part III of this report (see pp. 249, 250). Several accidents have occurred in blasting salamanders in the past 15 years, but in any one year the accidents from this cause are almost negligible, particularly in recent years. This is because, at a blast furnace, the opportunity or need of having a salamander dug out occurs only once in every three or four years at the most, and it may be dug out only at alternate relinings, or not at all. Another reason for the infrequency of such explosions lies in the fact that expert contractors are increasingly employed to remove the salamander.

The salamander, as previously stated, consists of a solid lump of pig iron, usually from 16 to 18 feet in diameter and from 3 to 30 feet deep, embedded in the foundation, the original fire brick being gradually replaced with iron during the years the furnace has been in blast. Most furnace men think it advisable on blowing out the furnace to remove the salamander, as a large salamander in the hearth foundation is believed to give rise to breakouts, split hearth jackets, displaced columns, cracked base plates, and off-grade iron at blowing in and especially to low bottoms in operating.

The salamander can not be removed until the brick lining is completely torn out of the stack and bosh, and the relining can not be started until the salamander has been removed, as every day the furnace is not in blast means a loss in production of about 500 tons of iron. The work of blasting the salamander is likely to be hurried.

This adds to the hazard in two ways, (1) the salamander was originally at red or white heat in the interior and cools off slowly, and consequently the temperature of the blast holes may be as much as 500° F. when ready to charge, and (2) under the drive and rush of relining work, men will not always take time to cool an originally hot hole, to cool a hole heated in "springing," or wait long enough or take the proper precautions in event of a misfire.

The work of removing a salamander is usually tedious and slow. Not only is the fire brick surrounding it excessively tough and hard in consequence of being exposed to high temperatures and pressure, being much harder than when laid, but the salamander itself may contain segregations of low-carbon iron which the drills can hardly penetrate or may require the starting of a new hole, so that by the time the holes are ready to charge, the men are impatient to finish the task. Hasty charging and blasting is not only dangerous but is foolish, because in removing a salamander drilling the hole is the time-consuming and expensive factor. Too much hurry in shooting may destroy the hole, which is relatively of much greater importance than a possible 24 hours saved in shooting and breaking up the salamander. The hole can easily be destroyed by using too heavy a charge of dynamite, which fills the hole too full, or by insufficient tamping on top of the charge. Such a heavy charge may widen the hole from top to bottom and destroy its effectiveness by blasting out a cone-shaped opening in the upper part of the hole. Therefore the importance of saving the holes, as well as safety considerations, should dictate care in placing and firing the charges.

CUSTOMARY PRACTICE IN BLASTING SALAMANDERS.

PREPARATION OF HOLES.

The holes are drilled either by air or steam percussion drills or by electric twist drills. Electric drills are faster, make a truer hole from top to bottom, and the drill bit freezes and sticks less than a percussion drill. Burning holes with an oxygen flame has been tried, but with little success. The size of the drill holes ranges from 1 to 2 inches in diameter, the usual size with twist drills being 1¼ inches. The depth of the hole varies. For a salamander of moderate size, resulting from one or two blasts with the furnace, the holes are 18 inches to 6 feet deep, the aim being to go only slightly over half the depth so as to avoid blowing the bottom of the hole out when the charges are fired. The arrangement of holes varies. Sometimes when only the edge of the salamander is to be split off, a concentric circle of holes is drilled about the periphery, 15 to 24 inches from the edge. If only the top is to be split off, horizontal holes are sometimes drilled into the face. The most common method is to drill a row straight across the top, dividing the salamander in halves. Occasionally sufficient rows are put in to quarter the mass. These holes are drilled 15 to 36 inches apart, but are most effective at close intervals.

CHARGING THE HOLES.

The explosives used vary from ordinary "40 per cent" to "gelatin" dynamite; a few plants use potassium-chlorate explosives. The amount used varies from 1 to 10 or more sticks (as much as can be put in) per hole. Exceedingly heavy charges are put in at some plants, sometimes sufficient to shatter windows in adjoining stores and residences. The explosive is charged both in stick form or loose, the sticks sometimes being wrapped with asbestos paper. The hole is tamped with sand or clay, a couple of plants using plaster of Paris.

FIRING THE CHARGES.

The charges are connected in series and fired with a blasting machine at practically all plants, although a few use the electric-light circuit. It is important in order to get the maximum breaking power of the explosive charged that all holes be fired simultaneously, and on that account fuses are not used.

PRECAUTIONS TO BE TAKEN IN BLASTING SALAMANDERS.

PREPARING THE HOLES.

One method, which is typical, consists in "chambering" or "springing" the hole and is believed to be the most efficient method of shooting blast-furnace salamanders available for furnace men, who encounter the necessity only infrequently. Blasting contractors, of course, have their individual methods which they adapt to the needs of each particular task.

Before the holes are drilled the salamander should be undermined or bared along the face as much as possible, or at least to the bottom of the hearth jacket, in order to determine the depth of drill hole necessary. In case only the sides of the salamander are to be removed the hearth jacket, if left in, should be protected by heavy beams, and enough of the sides of the salamander should be removed to permit building a brick wall as thick as the original wall. At every opportunity while the furnace is being torn out, the top of the salamander should be flooded with water, and also where no drilling is being done, so that the salamander will be cooled, and there will be less danger of premature firing of shots.

QUARTER DRILLING.

Small holes, not over 1¼ inches in diameter, should be drilled with electric or air drills. These are preferable to the larger holes made with rock drills, because there is less flying of pieces when the shot is fired. The holes should be drilled at intervals of 12 to 15 inches, quartering the top face to depths not more than three-fifths of the depth of salamander. Six-foot holes should be sufficient for any salamander formed from not more than two blasts with the furnace.

When the holes are drilled and ready for charging, the temperature of each one should be tested with a thermometer. For safety, the holes should not be more than 100° F. at the hottest part; if any hotter, they should be thoroughly cooled with water before shooting is begun. The holes should be loaded by a special foreman with two special helpers. All other men should keep away and only enough dynamite should be carried to the furnace to charge the holes.

The holes along one axis only should be fired at one time, and should be fired by electric detonators connected to a blasting machine, or to an electric-light circuit. Only a small charge should be employed at the start, one stick of dynamite in each hole. With this small charge and good tamping, the bottom of each hole is expanded. Each succeeding charge may be increased as the hole becomes larger, and after six to eight charges have been fired, cracks or fissures will start to form at the holes, in the direction of the line of holes. By this time as much as three or four sticks may be charged per hole, and the fissures rapidly develop into a clean split of the salamander with little flying of material. The two halves can then be similarly split.

DISTRIBUTING HOLES OVER THE FACE.

Another method of drilling is to put down a large number of holes over the surface, 15 to 24 inches apart. Light shots can then be placed in three or four holes at one time, the holes "sprung," reloaded, and shot, and so on, until the piece is cracked off. This breaks the salamander up smaller but requires more drilling, which can as well be done on larger pieces after they are removed from the furnace.

HOLES SHOULD BE THOROUGHLY COOLED.

In reloading holes when springing and chambering them, plenty of water should be used and enough time permitted for them to cool. Swabbing the hole with a wet rag is not sufficient, as the swab only extinguishes burning sparks and does not cool the hole. The water should be blown out of the hole with compressed air, and the temperature of the hole tested with a thermometer before it is recharged.

Cooling of the hole is essential; a fatal accident recently occurred from dropping sticks of dynamite in a hot hole. When more than one stick of dynamite is used in a hole, care is needed to see that there is a good contact between the sticks. In connecting up with the lead wires be sure that the splice between the wires from each hole does not touch the salamander, so as to cause leakage of current and a misfire. Misfires are especially dangerous in a salamander, and particularly so if the holes have not been thoroughly cooled with water. In event of a misfire in a hot hole, from displacement of the detonator, leakage of current, or a weak detonator, the heat will explode dynamite sooner or later. For this reason, thorough cooling of the hole rather than inclosing the sticks in an asbestos tube, and care in connecting the detonator, firing, and lead wires, is essential to avoid misfires.

FIRING THE BLAST.

The firing lines must be kept entirely separate from the lighting circuit or firing machine until the moment of firing, preferably by stationing one helper at the end of the lead wires, with instructions to hold them in his hand. All people should be kept at a safe distance, guards being placed at all passages that might lead to the danger zone. When the holes are loaded a whistle may be blown continuously until after the explosion to give warning. No explosives except the amount required for each shot should be kept near the furnace. The top of the furnace should be covered, or better still, a barricade of railroad ties and brushwood placed around and on top of the salamander before shooting. At least 30 seconds' warning should be given either by blowing a whistle or by the lookout shouting "Fire," before the shot is fired.

After the blast has been fired, the ends of the lead wires should be immediately disconnected from the firing machine, or electric circuit, and the lead wires examined to see that the insulation has not been injured or stripped by flying material. When such defects are found they should be repaired with insulating tape. Any uninsulated spots in the wire may cause leakage of current and misfires.

CONNECTING SHOTS IN SERIES OR IN MULTIPLE.

The following discussion of methods of connecting shots and the relative safety of the two methods, prepared by H. H. Clark, electrical engineer of the Bureau of Mines, is from Technical Paper 111:[a]

> There are two general methods of connection that may be used when several shots are to be fired at one time. These methods are known as series connection and parallel or multiple connection. As a general rule, shots should be connected in series when they are to be discharged by hand, or by spring-operated magneto generators, because these generators have not sufficient current capacity to fire shots in multiple. Where the shots are discharged by current taken from a power circuit, either series or multiple connection may be used. If shots are connected in series, a minimum potential of about 1½ volts per shot will be required if the source of potential is constant, and if it is variable, as is the case when hand or spring operated magneto generators are used, a somewhat larger voltage will be desirable. The source of power used to fire shots connected in series should be capable of supplying at least 1½ amperes.
>
> When the shots are connected in parallel, the source of power should have sufficient capacity to supply about an ampere for every shot to be fired, and should be capable of supplying sufficient potential to force the total current through the conductors that connect the shots to the source of power.
>
> When the shots are connected in series, it is easy to make a preliminary test to determine whether or not the circuit is open; and, barring the presence of defective detonators, all shots will be discharged if the proper voltage is supplied to the firing circuit. If the charges are connected in multiple, there will be no easy means of testing for poor connections and for faulty detonators, and consequently the presence of either is more likely to pass unnoticed. On the other hand, if the connections are in wet ground (or on a metallic surface such as a salamander) and the wires are not properly protected from moisture and leakage to the ground, it is quite possible for some shots to fail when connected in series that would surely be discharged in multiple. It is practically impossible to state which of the two methods of connection is inherently safer if a sufficient supply of voltage and current is available.
>
> Faulty connections are probably the most prolific cause of failures of shots, and the best advice is to emphasize that connections should be made very carefully; that wires should be prevented from leaking to ground and short-circuiting with each other; that firing machines should not be overloaded with shots or used for firing in mutiple when they are designed for series only; and that all conductors used in connection with shot firing should be of ample size to carry their required current without appreciable decrease in voltage.
>
> The danger of misfires may be minimized by the use of a testing galvanometer. This simple instrument employs a very weak current, which is insufficient to fire the charge but is enough to deflect a needle and thus indicate a perfect wiring connection. When the test is made the lead wires should be long enough to prevent any mishap. If such tests are made before tamping, wiring defects can be corrected.

PRECAUTIONS IN CONNECTING SHOTS.

The following rules given in Bureau of Mines Bulletin 80[b] should be followed:

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[a] Bowles, Oliver, Safety in stone quarrying: Tech. Paper 111, Bureau of Mines, 1915, pp. 21–23.

[b] Munroe, C. E., and Hall, Clarence, A primer on explosives for metal miners and quarrymen: Bull. 80, Bureau of Mines, 1915, pp. 47–48, quoted by Bowles, Oliver, Safety in stone quarrying: Tech. Paper 111, 1915, p. 24.

CONNECTING LEGS TO LEADING WIRES.

An electric igniter or an electric detonator should be so loaded into a bore hole that while it is in perfect contact with the charge the legs of the igniter or detonator reach at least 6 inches out of the completely stemmed and tamped hole. Both legs should be bared of their insulation for about 2 inches from their ends, and the wires so cleanly scrapod that a good electrical contact can be made with them. Each leg is then firmly connected with one of the leading wires by about five turns.

SPLICES SHOULD NOT BE OPPOSITE—WRAPPING SPLICES.

It is bad practice to have the two splices directly opposite each other, because when the leading wires are pulled the splices may touch one another and thus make a short circuit, which will prevent the electric igniter or electric detonator from being exploded. A better plan is to wrap the bare-wire splice with tape made for the purpose, which completely insulates them.

CONNECTING LEADING WIRES TO FIRING MACHINES.

After the legs are spliced to the leading wires (and not till then), the wires are connected to the firing machine from which the electric current is to be obtained. This last connection should never be made until all the men are at a safe distance from the place where the blast is to be fired.

ALL CONNECTIONS TO BE MADE FROM BORE HOLE TO FIRING MACHINE.

The rule should be made and never broken that when bore holes are charged the "connecting up" shall move from the bore hole back to the firing machine. The work in all blasting operations should be so organized that it can never be possible for the leading wires to be coupled to the firing machine while anyone is about the place where the holes are being charged and where the blast is to be fired.

PRECAUTIONS IN STORING AND HANDLING EXPLOSIVES.

The following rules regarding the use and care of explosives are taken from Miners' Circular 6,[a] of the Bureau of Mines.

STORAGE OF EXPLOSIVES.

Don't store detonators with explosives. Detonators should be kept by themselves.

Don't open packages of explosives in a magazine.

Don't open packages of explosives with a nail puller, pick, or chisel.

Packages should be opened with a hardwood wedge and mallet, outside of the magazine, and at a distance from it.

Don't store explosives in a hot or damp place. All explosives spoil rapidly if so stored.

Don't store explosives containing nitroglycerin so that the cartridges stand on end. The nitroglycerin is more likely to leak from the cartridges when they stand on end than it is when they lie on their sides.

Don't repair a magazine until all explosives are removed from it.

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[a] Hall, Clarence, Permissible explosives tested prior to January 1, 1912, and precautions to be taken in their use: Miners' Circular 6, Bureau of Mines, 1912, pp. 3–6.

THAWING EXPLOSIVES.

Don't use explosives that are frozen or partly frozen. The charge may not explode completely, and serious accidents may result. If the explosion is not complete, the full strength of the charge is not exerted and larger quantities of harmful gases are given off.

Don't thaw frozen explosives before an open fire, nor in a stove, nor over a lamp, nor near a boiler, nor near steam pipes, nor by placing cartridges in hot water. Use thawers such as are furnished by the manufacturers of explosives.

Don't put hot-water or steam pipes in a magazine for thawing purposes. Where large quantities of explosives are used, a special thaw house should be built large enough to hold the quantity of explosives needed for a day's work.

HANDLING DETONATORS AND EXPLOSIVES.

Don't carry detonators and explosives in the same package. Detonators are extremely sensitive to heat, friction, or blows of any kind.

Don't handle detonators or explosives near an open flame.

Don't expose detonators or explosives to direct sunlight for any length of time. Such exposure may increase the danger in their use.

Don't open a package of explosive until ready to use the explosive; then use it promptly. All explosives are injured by exposure to the action of the air.

Don't handle explosives carelessly. They are all sensitive to blows, friction, and fire.

USING DETONATORS AND EXPLOSIVES.

Don't crimp a detonator (blast cap) around a fuse with the teeth. Use a cap crimper which is supplied by any manufacturer of explosives.

Don't use a metal tamping rod. Even a copper-tipped rod is not to be recommended. Use only wooden rods.

Don't return to the face until at least five minutes after a shot has been fired. All explosives tested to date, including black blasting powder, yield inflammable and poisonous gases.

Don't return to the face for at least one-half hour after a misfire. Hangfires and misfires are most likely to happen when squibs or fuse are used.

Don't attempt to draw nor to dig out the charge in case of a misfire. Drill and charge another hole at least 2 feet away from the hole that misfired.

Don't use an ordinary dry-cell battery for firing electric detonators unless it is carried in a box having a safety key or button contact. The battery should be so constructed that if the leading wires or the legs of an electric detonator should come in contact with its poles the current will not be discharged until the safety key or button is used.

PART III. BLAST-FURNACE SLIPS.

PURPOSE OF INVESTIGATION.

The mechanism of blast-furnace slips, the chemical and physical conditions causing them, and the various theories to account for irregular movement of stock in the furnace have been an attractive field for discussion and speculation among blast-furnace and technical men for many years, because the subject related to the attainment of regular practice, and consequently to tonnage, quality of iron, and costs, and also to the safety of the men and of the equipment. The dangers incident to blast-furnace slips are comprised in the ejection of coke, limestone, ore, and scrap from the furnace top over the yards, tracks, roofs, and pathways, and in the added danger about the base of the furnace from bursting tuyères, burned blowpipes, chilled iron notch, and filled tuyère stocks. Occasional bosh failures, blown-off tops, or ruptured furnace shells have added to the apparent hazard, and, in addition, a marked tendency toward slipping tends, at many plants, to increase the accident risk of other operations about the furnace.

The safeguards against slipping are extremely few. Essentially the best safeguards and precautions against slips, with their attendant damage or hazard, lie in suitable furnace construction, design, and practice, and in the wise selection of the materials used for the burden. The number and severity of slips at blast furnaces in this country have been considerably lessened by progress along these lines. This reduction has been accomplished chiefly through many costly experiments and trials in the contour of linings, methods of distribution of charges, and changes in design and practice to suit the various ores and cokes American plants have had to use, covering a considerable range of years, rather than to any improvement which may be attributed to contribution of fact or theory by technical authorities.

In undertaking to review the theory and prevention of blast-furnace slips no attempt has been made to suggest a panacea for supposedly poor practice or unduly hazardous conditions at American furnace plants, for such a condition does not prevail. A summary of the more important contributions to the technical literature, with a tabulation of information obtained from various sources by word of mouth, has been prepared for the benefit of those in charge of and working about blast-furnace plants. Every slip of any consequence is indicative of deranged conditions in the furnace which need correcting. It is believed that the review of causes and remedies here presented may be found suggestive in overcoming the abnormal conditions which lead to slipping.

OCCURRENCE OF SLIPS.

A blast-furnace slip is always preceded by "hanging" of the charge in the furnace, where, because of irregularity in practice, or the augmenting of some abnormal or detrimental factor, a part of the charge becomes wedged. Sooner or later, from a few minutes up to several hours, this material gives way and falls inside the furnace; this sudden descent of material has been given the name of a "slip."

At many plants small slips are of daily occurrence, usually frequent, and many large furnaces apparently work only by slipping. As a rule, such slips cause little disturbance to furnace processes and offer little hazard to life or property. If this were not so, practice and work about blast furnaces would have become intolerable many years ago. However, chronic and excessively heavy slips may develop and may disarrange the working of a furnace for weeks at a time, in addition to introducing a considerable hazard to the force. After a furnace has hung for some time the charge may descend quietly, or gas and dust may be blown from the bleeders or explosion doors for several seconds, or the stock may fall so heavily that the impact shakes the furnace foundations or bursts the bosh. Sometimes the slip becomes so violent that it resembles an explosion. There is a deep, heavy roar, and dust and gas are ejected from the bleeder 150 feet or more in the air and from the explosion doors; a sheet of flame plays about the furnace stack, hiding the top; and large quantities of ore, coke, and limestone are ejected over the yards and tracks. Sometimes the top is blown off and the side is split, and in some instances blast furnaces have been completely destroyed. This explosive type of slip is prolonged rather than instantaneous, as it lasts 5 to 15 seconds.

INDICATIONS OF HANGING.

APPEARANCE OF THE GAS.

A furnace will nearly always give some indication that it is hanging. The most ancient and time-honored indication of good or bad furnace working has been the "top flame." This practice dates back scores, perhaps hundreds, of years, to the time when furnaces had open tops, and later on, to the time when the usual excess of gas was led off through a bleeder, where it might or might not burn. A furnace working regularly will usually emit a gas characterized by a gray fume, more or less heavy. At night there is usually a continued emission of pyrophoric sparks about the bleeder. If the gas burns, it usually gives a blue-red smoky flame.

As a furnace begins to hang, the gas becomes clearer and more translucent and the flame bluer. When the furnace becomes very tight and cold, the gas burns with a dust-free and translucent flame, or the gas may completely disappear, when the furnace is said to be "dead on top." With tight tops and bleeders the appearance of gas on top has ceased to be a conclusive indication of furnace working. However, as considerable amounts of gas leak from about the bleeder seats, it is still watched closely by experienced and practical furnace men. The use of washed gas at stoves removes any possibility of judging the working of the furnace by the appearance of the gas at this point; moreover, the characteristics of stove gas have never been a reliable indication of furnace conditions. The connection between hanging and the top flame is not necessarily constant, as different rates of driving, the use of coke blanks, the character of slag, or the wetting of the ores, affect the appearance of the flame and it may mislead those who are not familiar with the variations introduced in blowing and charging the furnace. When a furnace hangs for some time, becomes tight, and the gas apparently disappears or is "dead on top," the starting of the slip is often indicated by small jets of yellow flame about the seats of the explosion doors, several seconds to a few minutes before the furnace slips.

RATE OF CHARGING.

A second indication of hanging is the slackening in the rate at which the furnace takes the rounds of burden. A furnace taking 96 charges in 24 hours should, when working regularly, take on an averages 12 charges in each three-hour period; the number of charges is, however, above this average during the night and below it during the day. Wider variations in rate of taking burden are not uncommon, as on some days the furnace may take many more charges than corresponds to the average daily rate of driving and tonnage, or, on the contrary, charging may be so retarded that the tonnage for the day represents a gain of 5 to 15 per cent over the equivalent amount of iron charged. This may take place with regular driving and absence of marked slips. As a rule, however, a deficiency of more than two rounds from this established average over a three-hour period is an indication of hanging. Of course, the rate of charging is checked at much less than three-hour intervals, because hanging may develop in as short a time as 30 minutes.

A furnace may be working "stiff" and after taking two or three rounds stick for possibly 10 or 15 minutes, move, take another round, stick, move; then take three rounds, stick, and move, and so on. Many furnaces work this way day in and day out with no serious effect. At other furnaces the charge descends regularly at short intervals by a series of slight drops, and the beginning of a "stiff" movement is likely to develop into serious hanging. A furnace, whether working "stiff" or regularly, may "stick" very suddenly, or may work into this condition only by becoming gradually stiffer. As a general rule, subject to some leeway, when a furnace which is working stiff stands still for periods of more than 20 minutes, in which no charges are taken, it may be said to be sticking or hanging.

TEMPERATURE AND PRESSURE OF THE GAS IN THE TOP.

Closely connected with sticking, or a reduction in the rate of charging, is the temperature of the top gas, as in this case the temperature invariably rises, sometimes as much as 200° F. in 20 minutes. This rise in temperature is primarily due to cessation of charging, and may indicate nothing more than a "spell" (fillers stop charging to rest or eat) in the stock house. It may, however, if charging is stopped on account of the furnace sticking, indicate abnormal conditions inside the furnace. If the furnace is an isolated one, a marked drop in the pressure of the top gas, as its temperature increases, indicates hanging of the charge.

At a few plants where instruments for recording the composition of the gas are installed, a relation can be traced between "hanging" and the carbon dioxide content of the gases. Such information is, however, more of historical interest, after hanging has developed, than of immediate value in anticipating the condition.

BLAST PRESSURE.

Another indication of hanging is variation in the blast pressure. For example, the pressure almost invariably increases after casting, when the furnace hangs for a while, especially if the blast has been off the furnace for over two or three minutes and if the furnace has been running on blast temperatures of more than 1000° F. In normal practice and with an average blast pressure of 15 pounds, the pressure may rise to 17 or 19 pounds immediately after casting, and then gradually fall in the course of 40 minutes to the normal average pressure. The blast pressure may similarly rise when sticking and hanging develops between casting periods. A furnace running at a blast pressure of 15 pounds between casts may start to hang on top without any increase in blast pressure, or, on the contrary, the blast pressure may suddenly increase without any immediate sticking or decrease in rate of charging.

RELATION OF VARIATIONS IN BLAST PRESSURE AND RATE OF CHARGING.

An increase in blast pressure is almost invariably accompanied, however, by a decreased rate or the cessation of charging. Whenever hanging assumes serious proportions, increase in blast pressure and irregularity of charging are closely related, although a period of chronic hanging and heavy slipping may be brought about by a "hanging" in which the blast pressure remains normal until the slip has occurred. The most unmistakable, and the earliest, indication of hanging is to be found not in the blast pressure or character of the top gas but in the rate of charging. If the furnace is watched closely and at the first indication of hanging, steps are taken to anticipate its developing into protracted sticking and heavy slipping, the more serious and chronic hanging can usually be avoided, this depending somewhat on whether the hanging is at the top or bottom of the furnace.

CONDITIONS CAUSING TRANSITION FROM "HANGING" TO "SLIPPING."

Whenever the charge sticks or hangs a cavity is formed at some point in the furnace. For every 40,000 cubic feet of air blown per minute there is burned approximately 730 pounds of coke, which will occupy a space of about 25 cubic feet.

Accompanying the coke is 1,460 pounds of ore, occupying about 9 cubic feet, and 400 pounds of limestone, occupying 4 cubic feet, making a total of 36 cubic feet of material which is smelted per minute and disappears into the hearth as slag and iron or out the gas offtakes as gas and steam. In 40 minutes, therefore, a cavity about 1,200 cubic feet in size would be formed beneath the hanging part, representing in the shaft an average depth of about 5 feet or at the top of the bosh about 3 feet. Above this cavity, which is filled with gas, is a column of stock weighing, roughly, 25,000 pounds per foot of height, so that the total weight of the column of stock above the cavity ranges from 1,500,000 pounds when hanging takes place at the bosh to 250,000 pounds at 10 feet below the top.

The downward movement of the stock column having been arrested, it remains at rest until the cause of its sticking or hanging is removed. Uhling[a] has brought out the fact that as the friction of rest is greater than the friction of motion, then, if the latter is sufficient to bring the descending stock column to a standstill, the friction of rest must be more than enough to hold the material stationary. Also, the column is supported by the gas pressure against its base, and this pressure becomes greater as the top of the descending part of the column is lowered, because the shorter the descending part becomes the less resistance it offers to the ascending gas, and, therefore, the greater the temperature and pressure of the gas passing into and from the cavity. Finally, if the sticking has occurred in the upper part of the furnace, the arrested part of the column probably grows in bulk from the moment of retardation, and thus wedges itself still more tightly against the lining. Eventually, when the pressure of the gas in the cavity is high enough to dislodge the overlying column, or is so reduced by slackening the blast that it will not support the hanging material, or when the gas-filled cavity gets hot enough, or is abnormally cooled, the arrested column drops into the cavity, or "slips."

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[a] Uhling, E. A., Fundamental principles involved in blast furnace practice; Jour. Franklin Inst., vol. 159, 1905, p. 117.

THEORIES OF THE CAUSES OF BLAST-FURNACE SLIPS.

The effect and appearance of a slip depends upon where the hanging started, the depth and shape of the cavity, the pressure of the gas beneath, the weight of the suspended column, the physical character of the ore, and the manner of falling. There being no means of direct observation, one can only surmise the mechanism of a slip from the correlation of a great deal of experience, data, and observation of various men. Some slips have been so violent that they have been termed "explosions," and many furnace men believe that a true explosion may take place under certain conditions, this theory being based on the fact that the so-called explosion has been observed to follow the slip by an appreciable interval. A knowledge of the various theories of hanging, slips, and explosions is important in order that the most effective steps to prevent their occurring too persistently and excessively may be taken. Hanging of the charge probably takes place throughout the depth of the stock column, but to distinguish the different causes it may be considered to be divided into two varieties, "top hanging" and "bottom hanging." The causes are quite distinct in each case, but as the location of the hanging or sticking approaches midway between the mantle and the top the causes probably become due to a combination of irregular working, both at the bottom and the top.

CAUSES OF BOTTOM HANGING AND SLIPPING.

"Bottom hanging" or "deep hanging" may be considered to take place from the lower bosh to the top of the zone of slag formation and to be due to excessively refractory slag caused by variations in the materials charged or in the blast, fluctuations in silicon content of the iron, or by running regularly on too limy a slag. Although the theories of fusibility of slags are not exact, it is well known that an excess of lime or alumina renders a slag refractory. The point of fusion rises with increasing percentages of lime or alumina, and the excess of these constituents over the percentage represented by a slag of normal fusibility is apparently dissolved, the slag being a solution of a spinel, basic silicate, or aluminate in a silicate or aluminate which makes up the body of the slag. For example, a slag analyzing 33 per cent silica, 15 per cent alumina, and 48 per cent bases would roughly correspond to a solution of one part $2CaO.SiO_2$ in two parts $2SiO_2.Al_2O_3.3CaO$. The most fusible slag-forming constituents are considered to combine first in the upper bosh or lower inwall, the excess of basic constituents entering in as the materials sink farther down in the furnace and become hotter. Under these conditions it is evident that throughout the descent of slag-forming material some of the cinder-forming constituents are at the limit of their fusibility, and any violent irregularity which causes chilling of that part of the furnace where slag formation is under way, or any sudden increase in basicity, must cause these parts of the slag to congeal or freeze.

The effect of the lowering of the temperatures on slag formation is described in the following:

In normal work the first portions of the slag may be assumed to melt in contact with gas at 2,400° F. at 3 feet above the bosh line, the remainder of the basic gangue remaining in a pasty state until a temperature of 2,800° F., say, at a point 1 foot below the bosh line, is reached, when it goes into solution with the slag first formed. If the furnace suddenly works cold, the temperature may be 2,400° F. at, say, 1 foot above the bosh line, whereas the last stage of melting will be at 3 feet below the bosh line and the portion of slag held in the upper zone will now tend to freeze out the more basic part. That portion of the cooling slag away from the walls of course continues to descend, but the part next the walls may stick to them. As the working of the furnace again becomes normal, the slag stuck to the walls should melt, and probably does more often than not. Under certain conditions, however, the slag frozen to the walls may contain a large amount of fine coke dust and carbon dust, which makes it sufficiently refractory to withstand the normal temperature at the place where it has formed. When such scaffolds are analyzed or examined, they are usually found to consist of a conglomerate of very limy slag, iron oxides, lime dust, coke, and coke and carbon dust, cemented together into a solid refractory mass by slag and cyanide of potassium. These scabs may be augmented by fresh deposits of congealed material, or by somewhat plastic material plastered against the scab by compression and weight of the stock column above. In a way such action of a furnace in producing accretions at or below the working temperature, which the same or a higher temperature can not smelt is an anomaly, but it is a condition that has been repeatedly and thoroughly demonstrated.

Scabbing may also be caused by the furnace working erratically and excessively hot. A furnace running on 1.25 per cent silicon in the iron, with 1,000 pounds slag volume containing 34 per cent silica, 14 per cent alumina, and 48 per cent bases, will on making iron with 2.25 per cent silicon withdraw 48 pounds of silica from the slag, making the analysis 30.6 per cent silica, 14.7 per cent alumina, and 50.4 per cent bases, which represents a much more refractory slag and one which may conceivably lower the zone of fusion and cause scabbing of the furnace walls. Other factors, such as an excessively limy burden, flat boshes, high or overcooled bosh, too hot a blast, varying coke structure, irregular weighing of charges, variations in moisture content of the blast air, and, in fact, any defect in design, construction, or practice, may similarily change the location of the fusion zone, or cause the more infusible parts of the slag to freeze, and thus start scaffolds. Any one of these conditions, although possibly insufficient in itself to cause serious scaffolding, is aggravated by large variations in the temperature, volume, or moisture content of the blast. When scabbing or accumulation of material on the lower inwall or bosh becomes chronic, and the deposits refuse to come off, scaffolding is introduced. The accumulations of material project out into the furnace, diminish the cross section, increase the velocity of gas and hence its supporting effect on the descending burden, and afford a foothold on which the charge may rest and arch across. In one instance a plate had been cut by a slip, and the blast was taken off at casting, and upon the plate being pulled part of an arch could be seen on the opposite side of the furnace, on the upper part of the bosh.

Most cases of bottom hanging doubtless start in the fusion zone from one or more of the above causes, approximately as indicated. When such hanging develops into chronic slipping, other factors may be introduced, such as tight jamming of material in the lower inwall and bosh when the stock column slips down, bringing down of cold or unreduced, uncarbonized, or unsmelted material to the fusion zone, or the formation of a shelf or shoulder by erosion of lining. The primary difficulty and the one to be corrected is, however, the formation of an incrustation of slag, coke dust, and lime conglomerate on the lower walls of the furnace.

CAUSES OF TOP HANGING.

CARBON DEPOSITION.

"Top hanging" is entirely different both in occurrence and seriousness from bottom hanging and is more difficult to account for. The most attractive theory has been that hanging is due to the deposition of finely divided carbon on and in the interstices of finely divided and easily reducible ore. Van Flotten[a] was the first to advance this theory strongly, although Bell had demonstrated its plausibility and furnace men were familiar with the presence of carbonaceous dust in the charge, as in repairing a furnace it can be noticed that a fine voluminous mass of carbon fills every interstice of the charge, and when a furnace is blown out the walls are covered and the cracks filled with carbon, and also when a furnace slips or "rolls" a cloud of black dust is frequently ejected. The theories of carbon disposition may be summarized as follows:

1. The reaction (*1*) $2CO=CO_2+C$ occurs between 750° and 1,800° F., being strongly from left to right at 750° F., with consequent separation of carbon but diminishing rapidly from 1,300° F. up. The velocity of the reaction is increased in contact with catalytic or accelerating agencies, such as finely divided iron, manganese, or nickel.
2. When the carbon dioxide formed by the decomposition of carbon monoxide comes in contact with the finely divided iron, the iron may be oxidized to ferrous oxide, according to equation (*2*) $CO_2+Fe=FeO+CO$.
3. Above 1,200° F., inside temperature, a blast furnace works ideally, and reaction *1* can not take place to any marked degree because the partial pressure of carbon monoxide in the gaseous mixture of monoxide and dioxide is smaller than the equilibrium pressure of the reaction.
4. If from any cause the temperature falls below 1,200° F. in the zone in which the formation of metallic iron sponge from the reduction of ores has begun, then the equilibrium pressure of the reaction *1* becomes proportional to the temperature; and the partial pressure of carbon monoxide in the gaseous mixture being greater than the pressure corresponding to equilibrium, decomposition of the monoxide follows, with separation of carbon.
5. This is because the effect of increasing the pressure is to accelerate reaction, which may take place with decrease in volume. The decomposition of carbon monoxide takes place with lessening volume from two to one.

With these physical laws in mind, one can build up a theory to account for hanging at the top of the furnace. If temperatures in the furnace above 1,200° F., which will comprise zones where the reduction of iron oxide is virtually complete, are ignored, it is evident that at temperatures of 1,200° to 500° F. uninterrupted reduction of ore is taking place, and with theoretically ideal conditions there should be little or no deposition of carbon. This is in spite of the fact that metallic iron may appear at a temperature of 750° F. and is usually present in marked amounts at 1000° F., and is therefore in contact with the carbon monoxide and carbon dioxide gases. However, a sufficient change in the temperature of the gas will, according to physical laws, promote splitting up of carbon monoxide and reoxidation of iron, with resulting carbon separation. Suppose that a charge of extremely fine and reducible ore is introduced. On account of the accelerated rate of reduction and heat absorption the temperature drops. The pressure of the gas consequently drops below the equilibrium pressure for carbon monoxide and carbon dioxide in contact with finely divided iron and iron oxide, the partial pressure of carbon monoxide causing its decomposition according to equation *1* ($2CO=CO_2+C$).

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[a] Van Flotten, W., Das Hangen der Gichten in dem Hochofen: Stahl und Eisen, Jahrg. 22, February, 1892, p. 114.

By the increase in concentration of CO_2 and its higher partial pressure iron is reoxidized according to the following equation (*3*) $Fe+CO_2=FeO+CO$, and the CO thus produced again forms CO_2 and C, according to equation *1*. The catalytic effect of iron sponge increases the velocity of the reaction and results in carbon accumulating in such amounts that it fills all the spaces of the charge and an extraordinary increase in volume takes place which causes the stock column to wedge between the walls, especially when they are only slightly sloping, and the column remains hanging. The increased density and imperviousness of the charge retards the flow of gas and pressure builds up between the bottom of the hanging part and the top of the descending column beneath, which also tends to prevent the wedged part from moving.

The net effect of the two reactions is to give off heat according to the following equations:

$$(1)\qquad 2CO=CO_2+C.$$

$$(3)\qquad Fe+CO_2=FeO+CO.$$

Adding (*1*) and (*3*)

$$(4)\qquad Fe+CO=FeO+C.$$

$$-29{,}160=+65{,}700 \text{ British thermal units.}$$

Therefore the net addition of heat is 36,540 British thermal units. That is, the ultimate effect of the process of carbon separation will be to increase the temperature of the charge at the point where carbon separation is taking place until eventually the reaction ceases and may even be reversed, and the carbon dissolved again. The above process is assumed to be taking place under ideal working conditions in the furnace, which presumably never occur in actual practice, and if every reduction factor were correct carbon deposition would never occur in amounts sufficient to cause difficulty. At every temperature throughout the height of the furnace there should be, theoretically, that amount of carbon deposited which corresponds to equilibrium between CO_2, CO, Fe, FeO, and C, but owing to the high velocity of the gas current, equilibrium is probably seldom reached, and the content of CO in the blast furnace gas is always higher and that of the deposited carbon lower than would correspond to equilibrium at the various temperatures of the different zones.

INVESTIGATION BY METZ.

Carbon deposition is, for this reason, held lightly by many furnace and technical men and is not regarded as the cause but rather as an effect of hanging. In an investigation by Metz made on a blast furnace in Luxemburg,[a] the following observations were made which bear on this theory:

> The charges did not slide gradually, but intermittently. The masses alternately lessened and compressed, and just as intermittently do the gases stream upward, not so much through the mass as through accidentally formed chimneys. Since the time of Bell the separation of carbon has been regarded as the cause of scaffolds. If we regard as correct the results of the investigation, then a separation of carbon is possible only at temperatures below 1265° F., that is, only in the upper part of the blast-furnace stack and at a height of 9 to 13 feet under the throat; then the ores stay about 3 to 4 hours in the zone. As has been mentioned, the charges slide intermittently in the uppermost zones of the furnace; the carbon, which settled on the surface of the ores, is therefore partly blown off by the violent gas streams which arise during the descent, partly pressed against the surface of the ore fragments, and perhaps partly used in the direct reductions so that the increase in volume brought about by the separation of carbon becomes very small after the charges continue their descent for a short while. I am of the opinion that the separation of carbon is not the cause of scaffolds. It is, on the contrary, the effect of scaffolds and it only influences unfavorably the condition thereof.
>
> If for any reason an obstruction has been formed, then the gases flow only through channels of the mass. In the remaining parts thereof the velocity of the gas is but slight, and the temperature falls off gradually and carbon separates. The mass grows thereby considerably in its volume and is still more maintained in its hanging position. Here the conditions are favorable for a rapid, almost gas-tight obstruction.

EJECTION OF CARBON DUST.

Another argument against carbon deposition being a cause of hanging is found in the behavior of furnaces when they are hanging periodically on top. When the charge slips, often a cloud of black smoke is ejected, which consists mainly of carbon. By the falling of the stock owing to the collapse of the arch the entire contents of the furnace in the upper zones undergo considerable shaking up, and any carbon deposited should be blown out and cease to cause further trouble. Yet in spite of the burden material being free from carbon deposit after a slip, hanging may persist and grow worse after each check and slip. This is taken as indicating that the carbon, often ejected in astounding amounts, must have formed after hanging developed, and that its appearance should not be confused or distorted into being the cause.

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[a] Metz, N., Studien über die im Hochofen zwischen den eisenerzen und Gasen obwaltenden Verhältnisse: Stahl und Eisen, Jahrg. 33, 1913, pp. 98–104.

AMOUNT OF CARBON DEPOSITED.

The experiments of Laudig[a] showed that the volume of carbon deposited on Lake Superior ores may twice exceed the volume of the ores, and that the weight of carbon deposited may be 50 per cent more than that of the oxygen extracted from the ore. Such an increase in volume is out of all proportion to the usual batter of furnace walls, and it goes without saying that such an action can not regularly take place in any furnace even with the usual defects in operation, as it would make operation impossible or reduce practice to the expedient of checking the furnace every hour or every 40 minutes. As all furnace operators, when not confronted with scaffolds at or about the mantle, or with eroded inwalls, manage to get along with only infrequent or minor slips or checks, except for spells of irregular work, it must be that the amount of carbon normally deposited is small. Although carbon separation takes place to a considerable extent in the interior of an ore fragment when carbon monoxide diffuses in, and may persist until reduction is completed, the amount separated on the surface can not be equal to the amount theoretically or experimentally determined, because the gas at a given temperature is in contact with the ore during only a small fraction of a second. From the way furnaces work when running regularly the amount deposited can not be much in excess of the amount required for carburizing the iron, say 90 pounds; for absorption by CO_2 of limestone, say 150 pounds; for direct reduction, say 200 pounds; and some indeterminate portion which may travel down with the burden to the lower zones of the furnace, where at a higher temperature a larger amount of carbon monoxide remains in equilibrium with carbon dioxide, and the carbon dioxide again dissolves the carbon. Altogether the quantity deposited according to this hypothesis should be not more than about 450 pounds per ton of iron, whereas, according to experimental tests in combustion tubes, nearly 1,000 pounds of carbon would be deposited per ton of iron.

OTHER THEORIES OF THE CAUSE OF TOP HANGING.

This theory that excessive carbon deposition is the effect of hanging and not the cause requires its followers to furnish another cause for hanging, which is not easy. Top hanging can not be attributed to sticking in the bottom of the furnace, because charging may be stopped for an hour or more while the furnace is working normally at the tuyères and making cinder, nor can it always be attributed to erosion or overcooling of the middle inwall from 15 to 30 feet above the mantle, because hanging and top slipping occur in uncooled furnaces not eroded above the mantle. Hanging has been attributed, leaving carbon deposition aside, to the formation of scaffolds or accumulations on the sides of the furnace. One instance of such a scaffold was seen where the charge, after continued slipping, hung two hours and then slipped to 50 feet below the top, blowing the top off. In examining the furnace from the top a distinct scaffold was seen on the 45-foot line, sloping at an angle of 45° over the cinder notch and leaving an opening of about 10 feet. The material was analyzed after the furnace was torn out, with the following results:

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[a] Laudig, O. O., Action of blast-furnace gases upon various iron ores: Trans. Am. Inst. Min. Eng., vol. 26, p. 269.

Results of analysis of material from scaffold.

Constituent.	Per cent.
SiO	4.11
Fe	9.30
Al_2O	1.38
Lime	2.26
KCN	2.80
C	49.29

A similar scaffold analyzed as follows: Ignition loss, 52.87 per cent; SiO_2, 8.60 per cent; Fe_2O_3, 16.35 per cent; Al_2O_3, 6.56 per cent; and bases, 8.97 per cent.

Another cause of hanging may be found in the tendency of the stock to travel faster in the center of the column, and for fine coke, stone, and ore dust to accumulate on the sides. If the tendency becomes too pronounced there may be lodging or wedging of the interior part of the column on the slowly descending ring next the walls, as such accumulations of tightly packed ore, coke, dust, and lime have been found next the walls.

A theory infrequently advanced is that potassium cyanide sublimes in the bosh and is deposited in the cooler part of the furnace, where it may, if accumulated in excessive amounts, cement portions of the burden together and cause them to congeal on the wall. Deposits of this salt have been found as high as 40 feet above the mantle. Another possibility is that if the gas channels up through, under a gas offtake for instance, there are formed lumps of partly carburized iron, partly reduced ore, and slag, coke, and lime dust, which may adhere to the side of the furnace. If alkaline salts, heat, and abrasion cause deep erosion of the inwall, this may afford a foothold for an arch to form on, and thus easily induce hanging.

PROBABLE CAUSE OF TOP HANGING.

The most apparent cause of top hanging is a combination of carbon deposition in excessive amounts and of mechanical wedging independent of carbon deposition. It is possible that if, on account of irregular charging, or variations in the amounts charged, or in materials, the reactions do not proceed properly and equilibrium is destroyed, then carbon may be deposited in excessive amounts. This deposition may conceivably occur in but one limited segment of the charge, but be sufficient in itself to wedge the adjoining material firmly in the throat of the furnace. Mechanical causes, such as markedly uneven descent of charges, improperly designed gas offtakes, channeling, and obstructions or depressions in walls can also introduce conditions productive of hanging, with consequent carbon deposition. Obviously, remedies for this condition are to be found only in improvements in design, construction, and practice.

COLD WORKING.

Top or bottom hanging should not be confused with hanging and slipping due to cold working. This condition is simply the result of the furnace getting too cold from lack of heat in the blast, overburdening, too refractory a charge and insufficient reduction, or "cold-bottom" iron. In such a case the cause as well as the remedy is well known and consists primarily in reducing the blast volume, which usually goes hand in hand with increase of temperature or lightening the burden. Neither should top or bottom hanging be confused with the condition that arises when on account of insufficient blast penetration, badly arranged charges, or improper dumping of the bell, or improper size of bell, a column of material forms in the center of the furnace which does not smelt as fast as the material nearer the furnace wall. This column at first may consist of coke, but if it persists it may clog with cinder and dust and become quite refractory. When such a condition occurs, unusual hanging may take place. In these cases the use of a cold or light blast, light burden, coke blanks, or checking the furnace, does not have the desired effect, but the furnace conditions steadily grow worse. Luckily, such occurrences are rare. The time-honored expedient used to determine whether this condition exists is to rapidly drive a 1½-inch bar across the furnace, through a tuyère, let the bar remain about 20 seconds, rapidly pull it, and then place a chalk mark at the limits of the cold center. The practice is in some disrepute, as it only rarely indicates the anticipated condition. The writer has seen this test result satisfactorily only once in seven years' work.

HANGING NEAR THE CENTRAL PART OF THE STACK.

Hanging at the central part of the stack is usually due either to excessive water cooling of the inwall or to excessive erosion of brickwork at this place. The erosion is due, probably, to abrasion by the poorly distributed stock as it slides along the walls, softening of the walls from heat, and the attack of incandescent lime dust and alkaline salts on the brickwork. The most that can be done is to lessen by good bricklaying, better distribution, and shaft cooling the destructive action of these agencies on the ring of fire brick midway up the furnace.

CHARACTER OF BLAST-FURNACE SLIPS.

After hanging has developed, sooner or later the stationary part breaks away and descends or falls into the cavity formed beneath it. There is little or no relation between the violence of a slip and the duration or location of the hanging that preceded it. A furnace may be hanging in the bosh for two hours and then settle quietly, or may slip so violently that the escaping gas from the bleeder makes a terrifying roar. Hanging at the top may develop into a slip which manifests itself only by "rolling" of the stock and large quantities of dust carrying over into the gas, or by the sudden ejection of stock accompanied with a loud report resembling an explosion, or by a momentary cessation of gas escape at the bleeder, followed by an ejection of stock and gas, moderate or excessive in character, or may partake somewhat of the characteristics of two or more of the above types. Sometimes a slip may occur with no further hanging, or, on the contrary, it may be followed by increasingly protracted hanging and more heavy slips, until finally the charge sticks so firmly that ordinary measures will no longer suffice to make the furnace move.

MECHANISM OF TOP SLIPS.

If a furnace has been hanging on top for 60 minutes, at a point, say, 11 feet below the stock line, it is evident that a cavity 4 to 6 feet deep will be formed beneath the stationary port. Beneath this cavity is a deep column of somewhat dense stock, permeated with gas under pressure, the pressure increasing toward the tuyères. As the stock column below the cavity becomes lower and shorter the temperature and pressure of the gas in the cavity will increase as long as blast pressure is maintained; but if the blast pressure is reduced by checking the furnace, the pressure in the cavity will slowly drop until it is insufficient to hold up the overlaying wedged column, and the latter falls. If on account of the conical slope of the furnace lining the material falls in a mass like a plunger, the gas below is highly compressed by the weight and velocity of the falling charge. As the compressed gas can not escape to any extent or quickly enough through the lower column, the pressure will continue to build up until the gas bursts up through the weakest part of the falling column, carrying with it any overlying material. This may happen within two or three seconds, the escape of gas resembling an explosion. The uprush takes place almost instantaneously, and, finding easy relief at bleeders or explosion doors, the pent-up gas bursts out at the top, lights with a roar, and coke and ore are vomited up vigorously.

It is only rarely that the wedged column can fall as a solid mass. To slip in this manner, the bottom of the mass would have to hang tightly together and the brickwork would have to be smooth and of equal periphery. As a rule the hanging part starts to fall on one side, or piecemeal, from the bottom to the top. The former condition is often indicated by the stock line being lower on one side than the other after a slip, indicating that the gas has probably broken through along the walls and in its escape has taken most of the ejected material from the depressed side. When the hanging part gradually breaks away from the bottom up the resulting ejection of stock is less, owing to the more gradual release in gas pressure.

As a general rule, the higher the scaffold or hanging, or the higher the temperature of gas in the cavity, or the more suddenly and completely the stock falls, the greater the expulsion of stock. Also, the finer the ores and the greater the amount of coke, limestone dust, and fines the greater is the expulsion of stock. This latter statement explains why in former days with use of lumpy ores these ores allowed gas to escape throughout the cross section and hanging, and explosive slipping on top was not so prevalent as with the use of finer ores. Because of these various factors it is evident that the time and duration of a slip is somewhat variable. For example, it is well known that a furnace may slip 10 feet without "spitting," whereas at another time a slip of 10 feet will eject huge quantities of ore and coke.

MECHANISM OF BOTTOM SLIPS.

Bottom slipping is indicated by burning of gas from leaky blowpipe and tuyère-stock joints, by incandescent gas and coke surging back into the hot-blast main and filling the blowpipes with coke, a distinct shock or shaking of the furnace base or foundations, and usually by a more or less sudden expulsion of gas and dust from the bleeder. The pressure in the cavity beneath a scaffold near the bottom of a furnace is partly relieved when the hanging material falls, as the gas escapes to some extent up through the charge, and the blast mains and stove afford a considerable reservoir to lessen the back pressure. As the escape of gas must be up through a considerable height of burden, the friction muffles or delays any tendency toward an excessively violent outrush in the upper part.

When an arch formed at the bottom falls, it drops into a cup-shaped cavity, in contrast to the falling of an arch higher up where the lining increases in inside diameter with each foot of fall. Therefore, although an arch near the top may possibly hang together in part when it falls, an arch in the bottom must crack and collapse before the overlying column can fall. However, when the yield point is reached in a bottom arch, the great weight of burden above makes the material fall so suddenly and heavily into the empty bosh space that the increase in pressure of the gas, and the impact on the sloping hearth walls may burst the bosh or crack it, especially if it is badly eroded and weakly banded or jacketed. Top slips never come heavily on the bosh, whereas bottom slips do.

PRESSURE GENERATED BY SLIPS.

When the high pressure from top or bottom slipping ejects tremendous quantities of material, if explosion doors are provided, or as has occurred, throws out or displaces bells and hoppers, and resembles an explosion within the furnace, it is sometimes difficult not to believe that an explosion has occurred. Since 1901, when furnaces were first built without explosion doors and of sufficient strength to withstand the maximum blast pressure at any point, many furnace men have come to believe that explosive slips are merely the result of physical forces, that is, the release of pressure pent up by the falling column of stock.

METHODS OF CALCULATING PRESSURE.

The force which may theoretically come into play from a heavy slip is as follows: Assume that a 90-foot furnace with the stockline at 14 feet is hanging, and that after 1½ hours or more a cavity 9 feet in depth has formed at the 34-foot line. The overlying column is therefore 20 feet deep, with an average diameter of 17 feet, thus containing 4,540 cubic feet, or 490,320 pounds of burden material.

The problem resolves itself into finding the maximum pressure produced by the column falling into the cavity like a solid plunger or cylinder. Any pressure drop from the escape of gas due to the slope of the walls and leakage up the sides and through the burden as the cylinder falls is disregarded in order that the maximum pressure from the velocity of fall and weight of material compressing the gas may be calculated.

The weight per cubic foot of burden material may be calculated as follows: Three hundred pounds of ore equals 2 cubic feet, 150 pounds of coke equals 5.9 cubic feet, 75 pounds of stone equals 0.7 cubic foot, whence 64 pounds of burden equals 1 cubic foot. The initial gas pressure in the cavity may be taken as 3.5 pounds per square inch. As the cylinder falls the point of maximum pressure will be where the work done by the cylinder produces a force equal to the resultant upward force exerted by the compression of the gas in the contracted cavity. Consider an area 1 square foot in section. The work done by the falling weight equals the work done by the gas.

$$\text{Work done by the gas}=\frac{P_2V_2-P_1V_1}{n-1}.$$

P_1=initial pressure.
P_2=pressure resulting from fall of cylinder.
V_1=initial volume.
V_2=volume resulting from fall of cylinder.
n=adiabatic constant=1.407
W=weight of column A per square foot at bottom.
X=distance of fall,

$$V_1=1\times 9=9$$

$$P_1=3.5\times 144=504.$$

$$P_1V_1=4536$$

$$n-1=0.4$$

$$V_2=9-X$$

$$P_2V_2=P_2(9-X)$$

$$P_2(V_2)^n=P_1(V_1)^n$$

$$P_2=\frac{504\times 9^{1.4}}{(9-X)^{1.4}}$$

$$\frac{WX=P_2(9-X)-4536}{0.4}=\frac{\frac{504\times 9^{1.4}X(9-X)}{(9-X)^{1.4}}-4536}{0.4}$$

$$0.4WX=\frac{504\times 21.674}{(9-X)^{0.4}}-4536$$

$$0.4WX=\frac{10923.70}{(9-X)^{0.4}}-4536$$

The total weight in the column is 4,540×64=290,560 pounds, or 1,142 pounds per square foot at the base of the column. Hence

$$456.8X=\frac{10923.7}{(9-X)^{0.4}}-4536.$$

By trying various numbers, 4, 5, 6, etc., the value for X, the distance fallen, can be closely approximated, the true value being such that when it is substituted in the above equation both sides are equivalent or equal. Six feet is a close approximation of the distance at which the resultant forces acting upwards and downwards should bring the plunger at rest. To find the pressure in the cavity at this point the following calculation is necessary:

Let X=distance of plunger from bottom of cavity.
Y=pressure in pounds per square inch.

Then $X^{1.4}Y$=constant, at any point in the fall.

If $X=9$ and $Y=3.5$ at the instant the plunger starts to fall, then $X^{1.4}Y=75.859$=a constant, during the fall.. Values for X and Y are given in the following table:

Values for X and Y.

X.	Y.	X.	Y.	X.	Y.
9	3.5	6.0	6.17	3	16.29
8.5	3.79	5.5	6.97	2.5	21.03
8	4.23	5	7.97	2	28.75
7.5	4.52	4.5	9.24	1.5	43.00
7.0	4.98	4	10.89	1	75.86
6.5	5.52	3.5	12.13	.5	200.2

From the above table it is apparent that the maximum pressure will be at a point 3 feet from the bottom, which corresponds to a pressure of 16.3 pounds.

The same method can be applied to a bottom or bosh slip as follows: Assume that the charge is hanging at the top of the bosh. The volume of the column is 14,000 cubic feet, corresponding to a weight of material of 840,000 pounds. The weight per square foot at the bottom will be 2,211 pounds. With the blast checked, the pressure in the cavity may be considered as reduced from 20 to 12 pounds per square inch, or 1,728 pounds per square foot.

Then:

$$V_1=14$$

$$P_1=1728$$

$$P_1V_1=2192$$

$$n-1=0.4$$

$$V_2=14-X$$

$$P_2V_2=P_2(14-X)$$

$$P_2(V_2)^n=P_1(V_1)^n$$

$$P_2=\frac{1728\times14^{1.4}}{(14-X)^{1.4}}$$

$$WX=\frac{P_2(14-X)-24192}{0.4}=\frac{\dfrac{1728\times14^{1.4}\times(9-X)}{(14-X)^{1.4}}-24192}{0.4}$$

$$0.4WX=\frac{1728\times40.23}{(14-X)^{0.4}}-24192$$

$$884.4X=\frac{69,477.4}{(14-X)^{0.4}}-24192.$$

By a series of approximations it is found that when X, the distance of fall, equals 5 feet, that both sides of the equation are nearly equal. Calculating the pressure at this point by means of the equation $X^{1.4}Y=\text{constant}$, it is found that with X taken at 9 feet from the bottom, Y is equal to 22.3 pounds.

The same method may be used to calculate the ideal or theoretical pressure under any combination of circumstances. It is to be noted that the pressures thus indicated are of necessity higher than the actual conditions can possibly cause. The calculated value may be taken as that beyond which the pressure may not be expected to mount under any combination of circumstances for a given point of hanging in the stack, height of arrested column, and depth of cavity. It is to be noted that the maximum pressure in the two cases illustrated is more than the pressure per square foot at the bottom of the wedged column. This explains adequately why sometimes the entire furnace contents from the point where the hanging develops is shot out of the top. This, as far as is known, has occurred only where the bell and hopper have been insufficiently tied in, or where an excessive area of explosion doors is provided at the gas offtakes. If an easy vent is provided, then the entire furnace contents may be lifted bodily by the excess pressure. If the gas can escape only through one to three bleeders of limited capacity and through the downcomer, then the throttling effect gives time for the gas to escape and for the pressure in the cavity to drop without an excessive amount of the burden being blown out by the gas.

FURNACE DESIGN FOR RELIEVING PRESSURE FROM SLIPS.

From the records of burst boshes, split furnace shells, and blown-off tops at slips, it is evident that if the physical theory of explosive slips is correct, a burst bosh or a split shell results from the fact that at the point of maximum pressure the bosh reinforcement, or the shell and its lining, offers less resistance to the escape of the built-up gas pressure than do the columns of overlying and underlying stock, and consequently the pressure bursts this barrier rather than the one offered in the columns of burden material. As regards the blown-off tops, if the bell and hopper weigh 15 tons, and have an area of 200 square feet sealing the top, then a pressure of 1¼ pounds per square inch will be sufficient to counterbalance the weight of the bell and hopper, and a slightly higher pressure should suffice to blow it out if it is not bolted or tied in. If the pressure of the gas under the bell, as the result of a slip, should be as much as 10 pounds per square inch, a total lifting pressure of approximately 130 tons would be exerted on the bell and hopper. Originally the bell and hopper were simply set in the brickwork at the top of the furnace. From the fact that they were frequently displaced by even small slips, and sometimes fell into the furnace on account of failure and collapse of the brickwork from erosion at the stock line, various expedients were resorted to, the four chiefly used being as follows: (1) Tying the bell and hopper in by rods and washer plates embedded about 16 feet in the brickwork; (2) placing I-beams across the furnace top and resting the bell and hopper on them, these being bolted to the beams or held down with washer plates; (3) riveting brackets to the shell, bolting a top or bull ring to the bracket, and bolting the hopper and bell to the bracket ring, washer anchor plates being sometimes used in addition; and (4) contracting the top two rings of the furnace shell or jacket, riveting or bolting a cast-steel ring to the top of the shell, and securing the hopper and bell to this ring. The fourth method is now standard construction. With this construction, many still think it necessary to provide one or two bleeders at the top gas offtakes to relieve excessive gas pressure, and this is a reasonable precaution. A few plants, however, have bleeders of the butterfly-valve type which will not open automatically in event of a slip, but must be opened by hand if relief is desired. A few have bleeder valves which are forced against their seats by any rush of gas, no relief being afforded whatever.

Mitigation of the dangerous results of blast-furnace slips can be provided for only by adhering to the physical theory of the origin of explosive slips and constructing the furnace to conform with this belief.

THEORIES OF THE EXPLOSIVE CHARACTER OF SLIPS.

Those unable to find an explanation in the physical theory for the violence of some slips have attributed the apparent explosion to intensive oxidation reactions. These theories will be discussed briefly.

COMBUSTION OF FINELY DIVIDED COKE AND ORE.

Van Flotten[a] attributes the explosion to the intense carbon monoxide evolution caused by the intimate mixture of finely divided glowing ore and carbon according to the equation, $Fe_3O_4+3C=3FeO+3CO$. He reproduced the conditions inside the furnace by putting fine ore and coke dust separately into two tubes. These were inserted in a larger tube, which was sealed and brought to a bright red heat. The materials were then mixed by placing the large tube in a vertical position. When this was done a gage attached to the tube showed a pressure of 60 pounds within a few seconds, and on a stopcock being opened there was ejected a thick dark dust cloud as at furnace slips.

A pressure of 60 pounds is, of course, sufficiently high to explain the destructive results which have attended some slips. Van Flotten refers to the theory of carbon deposition and the wedging of burden in the furnace, which he considers sufficiently well proved by circumstantial evidence, to account for the conditions that may give rise to accumulations of red-hot fine ore and carbon dust in the furnace. As these materials stick in the furnace, they absorb heat and may become bright red, and when a slip occurs and they are precipitated into the lower and hotter part of the furnace reaction sets in. Van Flotten considers that the use of fine or friable ores, or friable coke, is disadvantageous because these promote the formation of scaffolds as well as the violence of the reaction.

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[a] Van Flotten, W., Die Explosionen beim Stürzen der Gichten im Hochofen: Stahl und Eisen, Jahrg. 28, 1908, p. 1015.

COMBUSTION OF DEPOSITED CARBON BY CARBON DIOXIDE.

Reaction between carbon and carbon dioxide, with formation of carbon monoxide, according to the equation, $C+CO_2=2CO$, obviously takes place with double increase of gas volume and is supposed to be accelerated by some catalytic agent. Simmersbach[a] possibly first advanced this theory. If it be assumed that a furnace is hanging and that the stock in the upper part of the furnace is impregnated with carbon, then when the furnace slips the finely divided carbon falls with the burden into a deeper and hotter zone, where it will jar loose in great part and, as a separate body, quickly attain the surrounding high temperature, become intimately mixed with carbon dioxide, and react to form carbon monoxide, thus generating two volumes of gas from one. This, Simmersbach holds, is the cause of explosions at slips, reasoning back from the phenomena he had observed at several slips. The objection to each of the oxidation theories is that they comprise reactions which take place in any normally working furnace, are incident to the process, and take place every minute without explosive violence. In fact, if this were not so it would be impossible to blow in and run a furnace without its exploding. Both reactions take place with an absorption of heat as follows:

(1) $Fe_3O_4+C=3\,FeO+CO$

$-270{,}800=+3\times 65{,}700+29{,}160$ calories.

Net heat requirement=44,540 calories.

(2) $C+CO_2=2CO$

$-97{,}200=+2\times 29{,}160$ calories.

Net heat requirement=38,880 calories.

As a general rule, an explosion can not take place by chemical means while heat is being absorbed, and the above reactions can take place only when heat is led in from the outside. The conditions attending an explosion of coal dust in a mine or of grain dust or wood dust in a mill are different from those in a blast furnace, because in the former the dust combines with oxygen, with an evolution of heat, which fulfills the conditions necessary for an explosion, because a chemical reaction takes place in which the formation of gas is simultaneous with liberation of energy, the gas phase transforming heat energy into volume or work energy. In addition, there is a tremendous acceleration in propagation velocity, a progressive rise in flame temperature, and thereby an excessively rapid increase in gas volume, which accounts for the destructive results of such explosions. One can not find any characteristic reaction in the blast furnace approaching these requirements. An accelerated explosion temperature, or, in fact, any generation of heat to give an explosion temperature, is absent, because the heat of explosion is the heat set free divided by the specific heat of the products, and in both of the above reactions no heat is liberated. This should suffice to contradict the theory of accelerated oxidation.

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[a] Simmersbach, Oscar, Ueber die durch das Hängen der Gichten veranlassten Hochofenexplosionen: Stahl und Eisen, Jahrg. 23, 1903, pp. 922–925.

ADMISSION OF AIR TO SPACES ABOVE THE HANGING STOCK.

The fact that just before a slip occurs there may be a decided diminution or stoppage of flow of gas from the bleeder, or even an apparent sucking of air, followed by an explosion, has led many to attribute such explosions to mingling of air with gas or fine carbon. In one instance an explosion door 42 inches in diameter was kept open at the top of a gas offtake of the "castle type" pending a slip and this allowed air to be sucked in when the slip occurred. The mixture of air and gas exploded on top of the stock line and displaced the bell and hopper. This explosion was almost simultaneous with the furnace slip and could very easily have been confused with it. A gas explosion on top of the stock line can not expel ore, is very different from so-called explosions at slips, and is likely to be so much more serious in results that it is safe to say that when such an explosion occurs the operators will recognize it. The theory has been much elaborated, the assumption being made that when a furnace hangs at the bottom, and slips, the entire stock column does not fall, but that the upper part remains hanging. Beneath this stationary part the falling part tends to produce a cavity with a vacuum into which air is drawn or sucked down through the overlying column. As soon as the hot combustible gases force their way from beneath the falling column into this air-filled cavity an explosion takes place with its usual effects. For obvious reasons this theory has never received any credence.

A less incredible theory assumes that with pronounced sticking the required amount of gas does not come back into the furnace to fill the empty space created when the charge slips, and that air rushes in through crevices in the top and about loose valves to form an explosive mixture under the bell. The assumption is also made that this gaseous mixture will diffuse with the scanty gas and may penetrate down deep into the stock, because it should not ignite until an ignition temperature of 1,100° F. is reached, and in this assumption explanation is found of the throwing out of ore and the sometimes almost incredible force of the explosion. Explosions of gas and air in the top of the furnace may and do occur, but that such an explosion can eject tons of ore and coke into the dust catcher, or out of explosion doors, is difficult to believe, because the force ejecting the stock must originate beneath the surface of materials in the furnace. It does not seem reasonable that an explosion of gas on top of the stock can explode gas beneath the top, nor that air can diffuse down into material through which gas is rising, or which is wedged so tightly that gas under pressure can penetrate it only with difficulty.

ADMISSION OF AIR TO SPACES BENEATH ARCHED STOCK AT BOTTOM OF FURNACE.

Occasionally one sees, hears of, or reads of a scaffolded furnace in which an arch has formed in the upper part of the bosh. The space beneath a scaffold at the top of a 16 by 22 by 13 foot bosh is 3,720 cubic feet, and the burden in this space would contain about 100,000 pounds of coke, which at the rate of 750 pounds per minute would completely burn out in 2¼ hours, assuming that the material remained bridged at the top of the scaffold and that the furnace was blown at an undiminished rate. If such conditions should occur and the bosh be entirely empty, or contain only a small amount of coke at the bottom through which the blast channels, this combination of circumstances might give rise to an explosion. It has been assumed that wedging at the top of the bosh is partly due to an excessive accumulation of fine carbon which traveled, undissolved, from the top of the furnace to the bosh, and that when by a slip this pulverent carbon was carried into the bosh, being full of air and incandescent, it burned explosively, thus producing all the results which give rise to so-called explosions at slips. A few instances in which the explosion blew the bell and hopper off and emptied the furnace down to the top of the bosh or lower seem to give credence to this theory. Against its plausibility is advanced the trouble experienced in anthracite practice, when the coal decrepitated and filled the furnace from top to bottom with fine carbon, ran out of tuyère openings when the tuyères were changed, and caused much trouble from scaffolding and hanging. So-called explosions, however, did not occur even with this excessive carbon dust. An experience may be cited where the existence of an arch across an empty bosh was conclusively known, and on blowing cold air for nearly an hour the furnace slipped very easily, with little lifting of explosion doors. If it is considered that in such a bosh cavity there can be not over 300 pounds of air, or 69 pounds of oxygen, which would at best burn explosively only 50 pounds of carbon, that all experience unites in recording only a pronounced thud or jar when the scaffold falls, and that eliminating excessive area of explosion doors and consequently throttling the gas has stopped ejection of stock to the 40 and 50 foot line, to say nothing of emptying to the bosh, then the plausibility of this theory of explosion becomes remote. That there is oftentimes free oxygen beneath bosh scaffolds and arches is true, but that it does more than to burn out carbon or coke in the bridged stock, induce smelting to begin in the arch, and open up cracks by which it eventually drops, is not likely. Some of the most severe and explosive slips seen have apparently had their origin at points 20 to 30 feet above the mantle, or 40 feet below the top, and it is difficult to imagine how any free oxygen could find its way up to this point in the furnace.

SKAPPEL'S THEORY.

Another theory concerned with explosion phenomena is as follows:[a]

> For some cause hanging develops, the under charges melt down, and then a gas-filled cavity develops, which may be considered to contain 26 per cent carbon monoxide. If now the charges fall, the gases will be strongly compressed and mixed with catalytic substances, thereby the reaction velocity of $2CO = CO_2 + C$ will increase greatly, in virtue of the reaction heat the temperature increases from the pressure, and consequently the reaction velocity again. Further, the separated carbon works as a catalytic agent acceleratingly on the reaction, so that it augments itself into an explosive character. These effects add themselves to those of mechanical compression of gas under the falling charges, thus generating a tremendous gas pressure which under certain conditions may blow the furnace shell open; that a black cloud is noted at blast-furnace explosions (slips) agrees most favorably with the above explanation.

The above theory is diametrically opposite to that of the formation of carbon monoxide from carbon and carbon dioxide. It finds its basis in the fact that the oxidation of CO to CO_2 and C takes place with evolution of heat and largely ignores the fact that it takes place with volume decrease and that it must stop at a certain temperature and increase in concentration of CO_2. It is included as illustrative of the divergent conclusions arrived at when theory is too enthusiastically relied upon to account for facts.

CONCLUSIONS AS TO CAUSES OF SLIPS.

The explanations for explosive slips discussed in the preceding pages are largely of historical interest, and are a product of the period through which furnace men struggled, when with increasingly finer ores and demand for greater tonnage slips assumed an increasingly destructive character. That furnace men were not satisfied with an explanation based upon simple physical facts but continued to search for obscure reasons for violent slips is possibly difficult for those unfamiliar with furnace phenomena to understand.

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[a] Skappel, H., Discussion of Van Flotten's papers on "Die Explosionen beim Stürzen der Gichten im Hochofen": Stahl und Eisen, Jahrg. 29, 1909, p. 214.

Those who have had experience with stubbornly hanging furnaces and the accompanying slips will find no difficulty in realizing the state of mind of their predecessors in the profession and their earnest efforts to establish an acceptable theory on the basis of which they could proceed to make improvements and changes in design, construction, and practice.

The physical theory was first formulated by Van Flotten, in 1892, who said, "If the upper hanging column falls, the gas in the cavity will be strongly compressed; it can not escape from beneath quickly enough, and in breaking through the weakest part of the arch and in creating a passage the overlying parts come out with it." Not until 1901 were furnaces built upon this theory. A furnace in Illinois was designed and built by Julian Kennedy with a tight top, and since that time his design has been increasingly widely followed. Some furnace men still are inclined to believe that explosive slips may be due to the sudden generation of gas. They point to the fact that an explosion occurs at a breakout of iron into water, which introduces reactions requiring heat, and that although reactions which are supposed to produce explosions take place normally in a furnace, they proceed too slowly, on account of their heat requirements, to produce a pressure increment. This failure to react explosively under ordinary conditions is because the heat necessary for the reaction is led in so slowly that it never becomes available quickly enough to cause explosive combustion; but if this necessary heat becomes available almost instantly, as it is claimed to occur at a slip, then the explosion will and does occur. Dynamite, burning quietly by ignition but exploding by shock, is also cited as an analogous illustration of how an explosive tendency may be accelerated when a slip occurs. As it has not been demonstrated that certain types of explosions can not occur, the subject will be concluded with the statement that development of design and construction based upon the physical theory has afforded almost complete freedom from blown-off furnace tops, and has also given increasingly more regular work, less terrific and less frequent slips, and a marked decrease in the hazard from injuries from these causes.

PREVENTION OF HANGING AND SLIPPING.

Where the subject of hanging and slipping is treated from purely physical and mechanical aspects, leaving aside theories of chemical reactions causing explosions at slips, one immediately begins to find reasons for improvements effected and the paths along which future improvements may be expected. Keeping in mind the facts of hanging due to wedging, arching, and bridging on account of mechanical obstructions or excessive carbon deposition induced by local irregularities, the principal items which cause hanging, and consequently slipping, are as follows:

1. Construction and design.
    - (*a*) Relation of size of bell to stock line.
    - (*b*) Distribution of materials in the furnace.
    - (*c*) Contour or lines of furnace.
    - (*d*) Preservation of lines and linings.
    - (*e*) Inferior brick and bricklaying.
    - (*f*) Location of gas offtakes.
    - (*g*) Distribution of blast.
2. Materials.
    - (*a*) Physical condition of ores.
    - (*b*) Structure of coke.
    - (*c*) Size and cleanliness of coke and flux.
    - (*d*) Irregularity in materials.
    - (*e*) Insufficient slag volume from materials.
3. Practice.
    - (*a*) Insufficient, excessive, or varying blast volume.
    - (*b*) Tuyères of unsuitable length or diameter.
    - (*c*) Improper charging.
    - (*d*) Excessive water cooling of lining.
    - (*e*) Unsuitable slag composition.
    - (*f*) High heat.
    - (*g*) Water leaks or excessively varying moisture content.

It is not possible to prepare a description of design, construction, selection of materials, or practice covering such a variety of factors as the above, which can be of universal application over the wide range of ore mixture, cokes, fluxes, and grades of iron that characterizes the blast-furnace industry of this country. The following discussion primarily applies to practice on Lake Superior ores.

DESIGN OF BELL AND RELATION TO STOCK LINE.

The relation between the diameter of the bell and that of the stock line is held within close limits at present. Not many years ago there was a wide variation, as is shown below.

Variation in relation between diameter of bell and of stock line.

Furnace.	Diameter of stock line.		Diameter of bell.		Furnace.	Diameter of stock line.		Diameter of bell.	
	Ft.	*in.*	*Ft.*	*in.*		*Ft.*	*in.*	*Ft.*	*in.*
A	16	0	9	0	G	15	6	11	1
B	16	0	9	11	H	16	6	11	0
C	16	0	9	11	J	16	0	12	
D	15	6	8	0	K	14	0	10	6
E	15	6	12	6	L	17	0	12	0
F	15	6	9	0					

Some plants used a narrow bell with the idea of preventing the accumulation of fine material next to the lining, while those using a wide bell claimed that such a bell did place the greater proportion of fines next to the walls and thereby prevented channeling next to the lining, and cutting of the brickwork by excessive localized reduction and melting. The main objection to placing fines next to the walls was that the fine ores, stone, and coke dust would become semifused in the course of their descent and stick to the lining, giving rise to scabs and scaffolds. To support this belief was the fact that many furnaces using a relatively large bell experienced severe slips, accumulation of scabs, and cutting of lining.

Smaller bells, however, gave no marked improvement, and so-called distributor devices were introduced. These consisted of deflector rings, cylindrical or conical, placed outside of and extending below the hopper extension to a point below the lowest position of the bell when open. Some variations of the distributor were an annular casting which could be raised or lowered at predetermined charges in order to distribute the ore alternately against the walls or in a ring away from the walls, or cutting out of alternate segments of the deflector ring to allow certain parts of the discharge from the periphery of the bell to go to the walls while the adjacent parts were deflected. In most instances, such modifications of the bell afforded relief from excessive and heavy slipping, erosion, and scaffolding, but at the expense of some other desirable feature of practice, usually in higher fuel consumption and decreased tonnage, even if the walls were kept clean. Two instances are known where a distributor of the above type was replaced, during the blast, with a large bell, and the furnace upon being started got hot faster than burdens could be put on to keep the silicon content in the iron down.

The best work on Superior ores is now being done with a relatively large bell, 12 feet to 12 feet 4 inches on a 16-foot stock line, which places an annular heap of ore about the walls with a central conical cavity or depression. The fines accumulate in larger proportion at the outside, with the coarse ore running down the face of the pile to the center. The difficulties which seemed connected with the large bell have largely disappeared with changes in the design of furnace lines, modification of means employed for the preservation of linings, and improved distribution of the charge on the bell before it is dumped. Attributing difficulties to the accumulation of fines next to the walls is not logical, for experiments with small-scale model furnaces have demonstrated that fines tend to move toward the center, and with the central part more permeable to ascending gases, this tendency would be accelerated by the faster dropping or movement at the center. This keeps the more intense action to the center of the furnace, in contrast to the action where coarse material accumulates in excess at the sides.

The method of dropping the bell has considerable effect on the relation of fines going to the outside to coarse material going to the center, and the condition of the ore also causes variations in the relation. With a bell opening slowly, or with a charge of wet or clayey ore, the tendency of the ore will be to drop nearer the center than when a bell opens quickly or when the charge consists of lumpy ore.

A final factor is the height at which the stock line is carried. With a 16-foot throat and 12-foot 3-inch bell the stock line is usually carried at 14 feet below the top, or about 5 feet below the bottom of the bell when it is dropped, and 7½ feet below the bottom of the bell when closed. It can be carried at 12 feet to advantage when the gas offtakes are not led off at too great a distance below the top. As a matter of fact the height of the stock line in relation to distribution of ore from the bell is at most plants seemingly more of theoretical than of actual importance, for a stock line supposedly at 14 feet varies from 12 to 19 feet. One of the advantages claimed for the Kileen distributor was that irrespective of the variations in the height of the stock line the distribution of material as discharged from the distributor was approximately constant. With no distributor the height of stock line has a considerable influence upon distribution.

DISTRIBUTION OF BURDEN.

The correct distribution of the charge on the bell is perhaps a chief factor in promoting regularity in furnace operations and one in which it is most difficult to obtain theoretically correct results. In getting the ore from the bins to the large bell there are from two to four drops, at each of which the coarse material may roll more or less to one side. The direction of fall being usually always the same, there is likely to be a pronounced concentration of coarse material toward the side of the furnace opposite the skip incline. This induces channeling by the blast and cutting back of the lining, which causes hanging and slipping. Upon the extent to which design of the charging apparatus obviates this detrimental sizing of the charge depends largely on the uniformity of movement of the furnace.

The majority of furnaces use a stationary double-bell top. At these a great deal of experimenting is necessary in determining the angle and speed of dumping of the skip car. Overthrow of material from the skip is also a point to be watched. If the angle and speed are correctly fixed to give the best possible distribution of material about the periphery of the large bell and the distribution still falls short of the desired result, defects in the design of the receiving hopper or in the capacity of the throat are usually responsible. In some cases lengthening the throat to equal the capacity of the skip has almost eliminated sticking and slipping. The stationary top has given regular and good practice on exceptionally low coke consumption, but the possibilities of defects in the distribution are nevertheless very great with this type. Even if the top adjustment and the dumping of the skip are correct for one mix of burden, with a progressive or sudden change in materials, the furnace may develop a hot spot through irregular heating within six weeks to six months after it is blown in, or sooner or later hanging and slipping may become a menace to safety and good practice.

REVOLVING DISTRIBUTORS.

The present tendency is to install a mechanical distributor. There are two classes of distributors—those revolving on top and those revolving at the base of the skip incline. Those revolving at the top of the furnace are all alike in principle; that is, the charge is deflected from an angle of about 60° to one of about 90° before it is dumped onto the hopper of the large bell. They differ in the way the charge is dumped onto the large bell and, to some extent, on the small bell. Some are open to the objection that if the operating mechanism of the distributor is deranged, the furnace must be stopped until repairs are made. Distributors revolving at the bottom of the skip incline are represented by modifications of the Neelend bucket. With this device the revolving mechanism is removed from the top of the furnace, the distribution can be inspected easily, and in event of a breakdown the furnace can still be filled satisfactorily.

DESIGN AND POSITION OF BELL.

No distributor or design of receiving hopper and throat can exert its maximum benefit unless the large bell is centered in the furnace. In addition the bell should be balanced before placing in order to avoid side thrust and swinging of the bell when it is lowered. The bell, for the large percentage of Mesabi or fine ores now used, should have a slope of 53°, and machining the surface is advocated at many plants. The resulting regularity of practice repays many times the time expended and the cost of constructional charges and design. The life of the lining is invariably reduced by localization of coarse ore in one segment of the furnace. Hot working will always occur at this point, the abnormal heat softens and fluxes the lining, and erosion and hot spots are the result. Spraying with water may prevent damage to the furnace shell, but there is no remedy for the hole gouged out, as material tends to build up and peel off, and the hole to constantly grow larger.

LINES OF FURNACE LINING.

Alteration of the lines of the furnace has often been essential, in addition to correct distribution, to freedom from slips. It is a matter of record that furnaces of relatively small height are freer from slips than high furnaces. Although this fact has been attributed to less crushing of coke, the principal factor contributing to this condition is that such short furnaces have a more tapered stack, which offers less opportunity of a foothold for material to freeze onto.

CONTOUR OF UPPER PART OF FURNACE.

Improvement in lessening slips has been obtained in high stacks by extending the straight brickwork down from the top of the furnace for as much as 22 feet and by running the inwall straight up from the top of the bosh as much as 9 feet. Shortening the cone segment of the inwall in this manner has greatly lessened the tendency toward slips, because the greater slope of the inwall affords less foothold for arches to spring from and for wedging of the stock. Wedging may be considered to take place in the upper part of the furnace, whereas arching usually is confined to the lower part. The wedging is due to expansion of the stock from heat in its descent, carbon deposition, channeling, and formation of scabs. Arching is more usually due to the cementing of slag-forming materials at the top of the bosh.

The extent to which straight brickwork may be carried at the top of the bosh and from the top of the furnace down is problematical. If a height of 5 feet at the top of the bosh and a depth of 20 feet at the stock line is taken as typical of the extent of straight lining in the stack, the fact that freedom from slipping is being attained in furnace practice with wide variations from these contours indicates that the requirements for satisfactory lines are not rigid. There is some point down to which the straight brickwork at the top may be carried that will present the maximum advantages of a steep batter of the lower inwall, without overreaching the point where the increase in bulk from carbon deposition and disintegration of the ore would cause wedging of the charge. The depth of straight brickwork at the top is a problem to be settled only by experience at individual plants or groups of plants using closely identical materials for burdens. Knowledge of the depth at which reduction, carbon deposition, disintegration, and swelling of the charge begin is not available. In view of the improvement that might be effected by definite data along these lines, it is to be regretted that more is not known, as such information would possibly enable the lines to be designed with more precision and with greater expectation of more regular working, decreased flue-dust losses, less off-grade iron, and fewer slips. Knowledge of the gas-ore reactions for various mixes is totally lacking for American blast-furnace practice.

CONTOUR OF LINING AT THE BOSH.

Slipping caused from scaffolds is confined largely to the bosh and slightly above it. Here furnace men are also confronted with uncertainty as to the exact conditions. One may, by a study of the lines of furnaces when blown out, form a theory of the location of the zone of slag formation and of the nature of the physical changes taking place there. It is known from various researches abroad that there is a point where swelling of the ore from its reduction ceases and its further descent through the furnace is accompanied with only such small expansion as would be caused by the increased heat. At a certain point well localized under conditions of uniform practice the various solid materials begin to soften, become pasty, and then to liquefy by reason of the combination of the various entities, such as sand, clay, lime, and oxides. Simultaneously with the formation of slag the liquefaction of such iron sponge as has arrived at this point and the reduction of residual iron oxide begins. The range of this action is evidently not confined to a narrow belt, and for a distance of a few feet there must be a zone of more or less pasty material in the transition stage between the solid and molten phase. The transition from the solid uncombined state, in which the material bulks large by virtue of the spaces between the particles, to a molten condition is accompanied by the following changes: A marked contraction in volume and a change of state from a solid slowly descending body, through a condition of agglomerated pasty material, to the relatively great fluidity at which the slag and iron quickly drop to the hearth.

If this zone of slag formation can be held where the area of the cross section is at a maximum—that is, at or just above the bosh—it is clear that the transition from solid to liquid phase should not interpose any resistance to the passage of the gas upward. If this zone, however, should be embraced in the contracting walls of the bosh, the effect of its diminished cross section would be to compress the pasty or agglomerated mass into a plug of relatively impervious material through which the blast could penetrate only with difficulty. Exposure to the cooled bosh would congeal adjacent portions of the slag, unreduced iron oxide, and carbon dust into a scab of refractory material, thus further decreasing the area of the cross section in which slag formation is taking place. The combined effect of resistance to blast pressure and freezing of material to the sides can easily be sufficient to arch the charge and prevent further descent, when the furnace will commence to hang.

Building the inwall straight for a few feet above the bosh provides some leeway for the inevitable rise and fall of the semifused plug of material in the fusion zone, but such a cylindrical section above the bosh is without benefit unless the height of the bosh is kept down. If the burden material should pass the cylindrical section and enter the water-cooled bosh before fusion and shrinkage takes place, the furnace will either work irregularly—that is, the iron will be alternately too hot or too cold, and frequently hanging and slipping will result—or the furnace manager will have to accommodate his practice to the furnace lines at the expense of lower blast heat, higher fuel consumption, and probably more rapid deterioration of the furnace lining.

The location of the top of the bosh must be dictated by the study of previous "blow-out lines" and furnace practice, in the absence of authoritative information as to intervening furnace temperatures and pressures, and points of slag formation. Keeping the bosh at a moderate height necessitates enlarging the hearth, or making a relatively flat bosh. The most satisfactory construction when working on Bessemer and basic iron has been found to be a low bosh, a steep bosh angle, and a wide hearth. A flat bosh accumulates material, wedges the descending coke, and causes high blast pressure, whereas the steeper bosh gives the least likelihood of material hanging to the side, keeps the column of coke open and thus promotes smoother working. The location of the top of the bosh, and consequently the hearth diameter and bosh angle, is somewhat dependent upon the limits of blast temperature which it is possible to carry, upon the sulphur content of the coke or ore, and consequently the basicity of the slag, and the character of ore smelted and pig iron made. Smelting of hard Clinton ores, or magnetites, is a very different proposition from the smelting of easily reduceable Lake Superior ores. High-silicon iron introduces a variation, the definite limits of which are fixed with difficulty.

PRESERVATION OF LINING.

After the operator has fixed upon the distribution that guarantees the best arrangement of charges in layers of uniform permeability for a given cross section, with the least possible channelling, and upon furnace lines which assure uniform and unobstructed descent of material with the least distortion of distribution, the preservation of these conditions is necessary to insure uniform and safe working. Failure of the lining to stand up under the work is detrimental because much of the benefit from good distribution is lost when the lining becomes worn in one or more spots.

PRESERVATION OF THE LINING AT THE STOCK LINE.

At the stock line the lining has to resist the continued impact of material discharged from the conical bell as well as the disintegrating effect of carbonaceous gases and varying temperatures. The abrading action of coke, scrap, and limestone as they are deflected outward from the bell hopper against the lining cuts an irregular channel or recess in the brickwork. This cutting action may extend some distance above and below the normal position of the stock line, according to the varying heights of burden in the furnace when the bell is dropped. The disintegration of the brickwork may reach such a stage that whatever advantage is obtained by relative distribution of coarse and fine material or contour of the top of the stock line is lost. If it be assumed that the lower inwall retains its shape, the inferior distribution and arrangement of material at the stock line may result in slips, because the increased diameter allows a thin stratification of less impervious material at some point or points. As this defective spot lowers in the furnace it induces channeling; the relatively thicker section receives insufficient gas to preheat it; and some portions of the charge arrive at the bosh insufficiently reduced or relatively cold, where they may introduce hanging and slipping by chilling the furnace, lowering the fusion zone, or plastering the sides of the furnace.

The effect of good distribution is also lost when the lining becomes eroded below the stock line. With proper distribution the movement of stock should be faster in the center of the furnace, on account of the greater coarseness of the core. Its more rapid descent creates a tendency in the surrounding material to work toward the central part. So long as this movement is regular about the surrounding ring of fines, no detrimental distortion of material is introduced, and the penetration by the gas tends to become more uniform by reason of the column of stock becoming of more uniform permeability throughout its cross section. If the lining should erode at some spot, the stock tends to spread out and fill the increased space. This promotes thinning of the layers in the charge at this place and permits an increased local upflow of gas. Any such effect is slowly cumulative, because the increased velocity and heat of the gas causes the adjacent materials and wall to become pasty and agglomerated. The recess formed in the wall affords a skewback, upon which an accumulation of the conglomerate material builds and pushes its way out into the descending stock. This material may continue to build out and form a permanent scaffold, may alternately build and slip off, on account of the heavy weight of the column of stock lodged on it, or may be melted off by cleaning blanks or by "blowing down" at low heat.

Various means are employed to preserve the shape of the stock line. In one type of construction the brickwork is replaced by a cast steel or iron shell which is riveted to the furnace jacket on top of the inwall brickwork and is either spray cooled or air cooled. The construction is somewhat complicated because expansion and contraction of the metal shell must be provided for, and the shell can be relied upon to carry only its own weight, so that much of the top work must be carried by other construction. Large wearing plates or rings of steel, embedded in the brickwork and tied in with rods, have caused trouble where used on account of their warping and displacing the brickwork. Tile of hard-burned brick have given fair satisfaction, but are not an absolute guaranty of preservation. The use of cast-iron brick invites trouble from variation in size when exposed to varying temperatures. Such action has resulted in shearing of the rivets on horizontal seams of furnace shells on top. The use of cast-steel plates, which are sometimes water-cooled and are hung and tied in front of the lining, or of high-carbon (or cast-steel) plates of small area anchored in the brickwork or set in concrete, at the stock line has proved to be the most efficient means of protecting the stock line from abrasion. When small plates are used any warping that takes place is in small local segments, which does not greatly affect correct distribution.

Most furnace men regard stock-line protection as an important means of attaining regularity in furnace working. A few anticipate the need of renewal and the necessity for shutting down at some time during the blast to repair the stock line and do not provide mechanical protection. As a rule, however, most furnace men consider the difficulties of access, the hazards from gas, the trouble in starting up, and the slipping and irregular working from the failure of the stock line prior to repairs as sufficient incentive to provide maximum mechanical protection at this point. Only a minority consider the preservation of the stock line as a minor point.

PRESERVATION OF THE LINING ABOVE THE BOSH.

Preservation of the brick lining in the inwall, from the bosh up to about 40 feet above the mantle, is a most difficult problem. Although intensive water cooling in the bosh, to the inside face of the lining, has proved sufficient to preserve an ample thickness of lining over a blast of several hundred thousand tons, in the inwall the lining must be largely left to its own resources and to the attack of heat, gas, and fluxes. These destructive agencies act in different ways, but all, under certain conditions, will effectually disintegrate refractory linings. Alkaline salts have been known to enter the body of the brickwork and to flux out the fire clay; lime dust at high temperatures readily combines with clay; furnace gas on penetrating fire-brick construction and coming in contact with iron oxide undergoes a decomposition by which carbon is deposited in the interstices of the brick lining, and this may disrupt and completely disintegrate the brick. The wear upon the lining by abrasive material in the stock may be a fairly insignificant factor except at points where there is local heating sufficient to soften the brick.

However, as the concentration of coarse abrasive material in any vertical segment of the stock column induces channeling by the blast, with consequent higher temperature and increased velocity of gas travel, there is likely to be intermittent, and at times constant, periods when the wear of the lining by abrasion is markedly accelerated by partial softening or incipient fusion of the lining from the increased temperature and velocity of the gases.

THICKNESS OF LINING, AND WATER COOLING.

The means taken to preserve the lining in the inwall are comprised in the use of thick, moderate, or thin linings, and in the extension of water cooling to the inwall construction.

USE OF THICK LININGS.

An excessively thick lining is characteristic of older practice, and some furnaces built within the past three years have thick linings. Good work has been done with these linings, but because of the high cost of construction, and the expense and time required for removal and replacement at relining, the natural tendency is to use them to destruction. Long before the lining has worn back to the shell, however, the furnace lines may become so irregular from local failure or erosion of the lining that it is difficult to prevent protracted hanging and heavy slipping. There is usually a period, preceding the blowing out, in which the furnace crew have to contend with severe scaffolding and hearth trouble. Water sprays are used on these thick linings when the shell becomes hot, but as a rule sprays can not counteract failure at any point until the lining has become eroded to a detrimental depth. Repeated observation shows that although brick in the upper inwall can withstand the heat and friction, the brickwork 6 to 30 feet, or more usually 15 to 30 feet, above the mantle wears out relatively quickly on thick linings. This is the zone of maximum erosion and is roughly the point at which the material is still coarse and firm, although at a heat sufficient to somewhat soften the brick. When the face of the brickwork has worn away so that the shell gets red hot, a spray is put on. By this time, however, there may not be more than 6 inches of brickwork left of an original 60 inches. The water spray may succeed in holding this thin wall, or if the wall crumbles the spray may preserve the shell for long periods, although incandescent coke is against the lining. If the spray is put on as soon as the paint on the shell starts to darken and before the shell becomes red hot, a thicker inwall will be preserved. Steam is sometimes introduced between the lining and the shell, a ½-inch steam line tapped into the shell protecting about 100 square feet of surface. These steam jets are more useful for patching cracks than for checking erosion, as in the latter case they are of doubtful benefit. Tonnages on linings 48 to 72 inches thick amounting to 1,800,000 tons of iron have been obtained, although the last 150,000 or more tons has usually been made under very trying conditions of slipping and with high consumption of coke.

USE OF THIN LININGS.

The other extreme is to use a thin lining, 9 to 13 inches thick. The difficulty with thin linings is that the life of the entire lining is dependent upon that of the weakest part. When local failure occurs the entire lining may fail on account of lack of stability, owing to the thin wall and falling in of adjacent brickwork. Failure of the lining is followed by heavy slipping, caused by the adhering and building of material on the cold shell, a condition which necessitates blowing out after too short a run to make this construction feasible. Also, the temperature of the cooling water must be carefully watched in order to prevent scaffolding, even when the lining is intact. One instance is known where a thin-lined furnace being blown in was very nearly put out of blast by the formation of a scaffold at the mantle, a shutdown being averted only by cutting off the water on the first two cooling rings of the jacket above the mantle, with much trouble from hearth difficulties and cold iron. Thin linings have been replaced at some furnaces with 22½-inch linings.

USE OF LININGS OF MEDIUM THICKNESS.

Moderately thick linings have given good results and tonnages of 600,000 to 900,000 tons to a lining. A lining 27½ to 36 inches thick has a seeming advantage for the reason that the brickwork on a thick lining will cut back to a thickness where exterior cooling checks further erosion of the brick. On a moderately thick lining this cooling action will be almost immediately effective, also a water spray can be used to immediate advantage when local failure of the lining develops, thus obviating deep erosion before the spray is effective. Consequently the contour of the walls can be expected to remain more uniform, without the excessive chilling caused by a thin lining.

PLACING OF COOLING PLATES.

The practice of water-cooling linings has been very successful, although it has been brought into disrepute repeatedly by the practice of placing the nose of the cooling plate flush with to 9 inches back from the face of the lining or too close together.

The plates have been placed so near the face in order to preserve the lining at approximately its original lines. That this is unnecessary is indicated by the usual experience that a furnace starts to work most smoothly three to five months after it has been blown in, when it may be assumed the contour has reformed by the cutting back of the original lines. Although the deviation may be small, it is a sign that rigid holding of contours is far from necessary. Placing the plates near the face of the lining has been a failure, either because they could not survive long, owing to cracking and erosion of the nose, or because the water had to be turned off on account of the scaffolding and slipping induced by excessive chilling of the face of the lining. The best practice is to place stock-cooling plates 14 to 18 inches from the face of the lining, spaced 3 feet to 3 feet 6 inches on vertical centers and about 5 feet on horizontal centers, each row being staggered. Such cooling is not intense enough to cause scaffolding or disposition of cyanides, but chills the face sufficiently to check abrasion by the grinding of descending stock against softened brick and to retard fluxing action. Thick linings which had previously shown hot spots and commenced to work irregularly at the end of 12 to 18 months have worked smoothly on successive linings with stack-cooling plates inserted for upward of three years. The height of the top plate is about 30 feet, to prevent the formation of a recess or shelf on top of the top row from erosion, as may occur when the height is but 20 to 25 feet. Several plates may be lost without apparently affecting the work of the furnace, and the effect of possible shelf formation between the rows, although theoretically detrimental, has not been marked.

INFERIOR BRICK OR BRICKLAYING AS A CAUSE OF FAILURE.

Failure of the lining, when not due to faulty distribution or the contour of the lining, is frequently ascribed to the brick itself. In the inwall defective brick have a wide opportunity to cause failure. Among the chief factors contributing to inferior brick for the inwall are irregular size, spalling propensity, and insufficient hardness. Excessive porosity, excess iron oxide, sand, or other impurities, deficient resistance to fluxing, insufficient refractoriness, and shrinkage are possible faults, although not as a rule outstanding or of frequent occurrence. The open joints left by irregular brick require an excessive quantity of fire-clay grouting, which is comparatively easy to flux and is permeable to gases; any spot in the inwall of insufficient hardness begins to abrade at an early stage of the blast, and spalling may accelerate any failure due to other tendencies; but, as a general rule, it is the opinion of furnace men that fire-brick construction for the inwall is fully abreast of the perfection to which other factors affecting regular working have been brought.

Given satisfactory brick, good bricklaying is essential. The expense of grinding, washing, weathering, mixing, molding or pressing, and burning demands equally painstaking work in placing brick in the furnace. If the shell is heavy—¾-inch plate, double and triple riveted—the brick can be placed against the jacket. Such construction prevents the packing space yielding and permitting cracking of the brickwork by expansion strains. The formation of cracks, and a hot jacket are not uncommon where a large packing space is used. For jackets lighter than ⅝-inch, a packing space filled with some yielding material is usually essential. There are several examples of split shells or sheared rivets on furnace jackets when insufficient or no packing space has been provided. It is, of course, common opinion that thin joints should be made in laying the brick, and that accuracy in making the joints should be exacted. After the lining has been placed, it is essential to dry it thoroughly. Blowing it with a green damp lining handicaps the durability of the lining by the generation of steam in the interstices of the structure, thus introducing the possibility of the formation of crevices and rupturing of the fire-clay bond. Two weeks is none too long, in the opinion of most furnace men, for drying out the lining.

GAS OFFTAKES AND DISTRIBUTION OF BLAST.

Other structural defects which induce slips are improper blast distribution and improper location of the gas offtakes. The gas offtakes should be high enough above the stock line so that opportunity will be afforded for any stock carried upward by a rush of gas will fall back before it is swept over into the downcomer, and so that stock dropped from the bell will not be carried over into the gas offtake. The offtakes should be large enough in diameter and of sufficient number so that the velocity of gas at each opening will not be excessive. With fast driving, there is a tendency for the gas to channel up through the stock column, cut the walls, and cause hanging if an excessive amount of material is lifted and carried over directly from beneath the location of the gas offtake. A similar tendency may be noted when a gas offtake is directly over the juncture of the bustle pipe and hot-blast main, or over the tapping hole or cinder notch. Accordingly the best location of offtakes is off center from the location of the blast main and notches.

Improper blast distribution is seldom the cause of slips, but occasionally a defect in blast allows more air to enter on one side of the furnace than at the other. This causes one side of the stock column to travel faster than the other, the top of the stock becomes lopsided, and the distribution is distorted, with channeling of the gas up one side; then cutting of the walls, and scaffolding, and heavy slips may result. This is avoided in construction only by having the hot-blast main join the bustle pipe at right angles to the tangent of the bustle pipe at the point of juncture. The hot-blast main should preferably extend from this point in a straight line back past the stoves.

MATERIALS USED FOR THE BURDEN.

FLUX.

Limestone or dolomite occasions the least difficulty and produces the least irregularity in furnace work of any of the materials of burden. Dolomite, when used exclusively for a flux, may cause considerable trouble with some practices and grades of ore, but the trouble is very easily remedied by introducing calcite into the mix. The difficulty is due to indeterminate causes, involving the fusion point, fluidity, and sulphur-saturation point of the slag. When a more suitable proportion of lime and magnesia in the slag has been determined, other factors which may cause slipping are too large a proportion of dust or dirt, too refractory limestone, crushing the stone to excessively large or small sizes, and irregularity in analysis. Elimination of fines by screening and washing the crushed limestone has been specified at several plants, and the additional cost has been offset, it is stated, by improvement in practice. Further elimination of gangue is obtained by specifying a lower silica content in the limestone. With increasing amounts of gangue in the burden to be fluxed, the use of a limestone with lower silica content decreases the amount of stone used, and thus reduces appreciably the variation from changing the amount of available lime in the burden. This is a source of irregularity which is aggravated by a large proportion of silica in the stone and by the use of a large amount of flux. The size of the stone should be dependent on its refractoriness, as the temperature at which carbon dioxide is completely expelled from various kinds of limestone varies. So long as excessive amounts of limestone persist in coming down to the tuyères, there is need of finer crushing. On the other hand, it should not be crushed so fine that all the carbon dioxide is expelled before the limestone has passed through the zone of reduction. Charging finely crushed stone, similar to road-building material, has been found to cause slips. The usual specifications are for limestone to be crushed to pass a 4½-inch ring, and not through a ½-inch ring.

COKE.

The character of the coke may be the keystone of uniform practice and freedom from excessive slips, as even with satisfactory ore and flux, and with every attention to details of construction, inferior coke is productive of irregular practice. The chemical analysis of blast-furnace coke is becoming less an object of scrutiny, because the working limits of ash and sulphur content are approximately fixed, and excessive percentages of these constituents are kept down either by the use of coals which contain the proper percentages within limits, or by washing the coal before coking. If the percentages come within the desired limit, the usual variation in practice appears to be negligible in coke from the same operation. However, from the fact that many furnaces do not draw coke from one operation or set of ovens, there is usually a sufficiently wide variation in analysis to demand attention. It is common experience that variations in coke, as shown by analysis, do not bring about the irregular furnace practice that a similar variation in the ores or fluxes do. The fact that probably the greater proportion of the ash is incorporated in the slag at the tuyères accounts for this. Variations in slag-forming constituents which combine at the bosh should introduce a more direct effect upon the regularity of descent, because a wide variation in fusibility of the slag in the smelting zone is immediately reflected in increased pressure, hanging, and slipping, whereas a considerable range in acidity or basicity may exist in the hearth without any marked effect on the working of the furnace.

High-ash content is in itself frequently detrimental, in that it introduces cleavage planes into the coke structure and may thus characterize physically weak coke. The physical requirements for coke are a certain uniform degree of hardness, toughness, porosity, density, and possibly absence of dense graphitic coating on the surface. Excessive softness and brittleness invite crumbling during drawing, transportation, and charging, with the formation of an excessive amount of dust. This introduces a handicap to uniform practice and freedom from slips by reason of the less open character of the column of stock. To eliminate this drawback, the coke is largely screened at the bin, and this, in addition to causing a more open stock column, also lowers the ash content and involves little or no loss in heat value, inasmuch as fine coke is evidently dissolved to a large extent in the top of the furnace by carbon dioxide. A more serious consequence of very soft or brittle coke is its liability to be ground to pieces by the abrasive action of the furnace walls and of the burden in its descent through the furnace, thus choking the furnace, with detrimental results to uniform working.

The effects of variations in the size and density of the coke are especially serious when the coke is charged by volume. By reason of the variation in weight charged, for the same volume, there is a tendency from day to day, or period to period, for the fusion zone to rise or fall. Excessively large pieces of coke permit the fragments of ore, especially if it is fine ore, to roll or heap together, so that the ore may be insufficiently reduced or even fused prematurely, and on arriving at the fusion zone in this condition cause chilling of the furnace. Coarse coke also presents a difficult handicap to the accommodation of charging practice to the lines of the furnace, if the coke is too dense or not porous enough to permit prompt combustion of the large pieces in a restricted zone at the tuyères. The burning must necessarily extend high up, thus making most difficult the concentration of combustion and localization of intensity of heat necessary to keep the slag-formation zone at the desired point. This introduces a possibility of irregular working, because the contraction in volume of material from combustion may not take place in a manner suited to the batter of the bosh. Excessively small coke, even if of the desired porosity and toughness, introduces irregularity by reason of its lessened ability to keep the stock column open enough to allow an approximately uniform and easy upward flow of gas throughout the section of the stock column. The blast is forced to tear a local channel through the impervious heap of material, which is not conducive to proper preheating and reduction of the other parts, thus giving rise to slips.

Variations in the porosity, density, cell structure, and surface of the coke may also cause irregularity and slipping. The increase in speed with which a furnace works when more combustible coke comes down to the tuyères is often beyond control by any increase in blast temperature. In this case the heat of the blast is insufficient to meet the demands from the quicker movement of the burden down to the hearth and chilling takes place, or slag may form lower down and chill on the walls, and the charge may arch and hang. Corrective measures, such as using smaller tuyères, plugging the tuyères, lightening the burden, or reducing the blast, are often no sooner applied than it is found that the character of the coke is changing to a normal or even excessively dense character.

Irregularity in the size, density, structure, and surface of coke is common where furnaces are furnished coke from several operations or sets of ovens, even from the same coal. Variations in temperature and time of coking and in methods of drawing and quenching cause variations in the characteristics of the coke that are difficult to cope with, even with coke from the same ovens. The development of tests that will indicate the probable behavior of coke in a furnace from a preliminary study of its physical characteristics and a rational set of specifications based upon physical qualities for coke would afford furnace men much relief from irregularity in practice and the attendant difficulties and dangers incident to the hanging and sticking of the charge.

ORES.

The tendency in blast-furnace practice throughout the country is of necessity toward the use of ores of increasing fineness and of decreasing metallic content. This movement has been retarded from time to time by the utilization of new deposits previously unknown or unavailable by reason of geographical location or prejudicial composition and by the fact that, as the richness decreases and the difficulty of smelting the ore at a profit increases by reason of the growing fineness, leanness, and cost of the charge, hitherto excessively expensive processes for concentration and agglomeration of the metallic material become an accepted and profitable feature of practice.

EFFECT OF CHARGING COARSE ORE.

The handicaps to safe practice introduced by the character of the ores charged is dependent upon their size and uniformity. Large lumps of ore may pass through the furnace without becoming completely reduced. The effects of charging occasional lumps of ore the size of a man's head are almost impossible to detect, but occasionally some of the hard ores are insufficiently crushed for furnace use or are charged in excessive amounts. Experience is common where a proportion of 15 or more per cent of a hard "old-range" ore has immediately caused difficulty. The presence of lumps of partly reduced ores beyond the point where indirect reduction takes place and where the direct action of carbon must take up the unfinished work may result in frequent chilling of the hearth and in scouring cinder. The loss of equilibrium at the fusion zone in cases where direct reduction is in high proportion results in insufficiently carburized and smelted iron and also partly reduced and relatively cold ore being precipitated into the hearth. Such material demands a higher heat to be restored to normal condition than does partly plastic iron sponge and may easily result in ironed-up tuyères, loss of bronze cooling equipment, and chilled holes with their attendant danger and difficulty. Considerations of correct practice no less than of safety require further crushing of any ore so dense or difficult to reduce that it comes down through the furnace without adequate indirect reduction.

EFFECT OF CHARGING FINE ORE.

Similar conditions obtain when the burden is made up of excessively fine ore. Even the most ideal distribution of burden, proper lines of furnace, proper grades of coke and limestone, and skillful practice do not suffice to overcome completely the erratic behavior of such burdens. Several factors contribute to this. Too high a proportion of fine oxides induces deposits of finely divided carbon as a result of any marked variation in the distribution or the rate of descent, thus causing the column to swell and stick in the upper inwall. The black fume given off during such practice is an indication of this carbon deposition, and is sometimes observed when a large proportion of flue dust or exceptionally fine Mesabi ore is used. This results in frequent slipping at the top, which causes irregularity in charging, a condition which aggravates and magnifies any tendency toward irregular working to which the furnace is prone. Hanging is also induced, farther down in the furnace, by the fact that the fine materials cause an excessively dense stock column by reason of their easy reducibility and their consequent increase in fineness. As the gas can ascend through such a charge only with difficulty it forces irregular paths through the column. The tendency of the fine material to pack and fill the interstices of the charge precludes much possibility of the gas currents being diverted from their segregated paths until the column is shaken up by a slip. The channeling permits the gas currents to reach the top of the charge without giving off their sensible heat and those parts of the charge adjacent to the streams of rapidly ascending hot gas become plastic. Where this fine plastic material comes in contact with the walls, it congeals and adheres to the relatively cool surface, and results in hanging, scaffolding, and slipping. Such agglomerated material, if dislodged, and portions of the ore charge which have packed and become impervious to gas currents, may arrive at the melting zone insufficiently reduced. Lack of reduction of fine ores gives rise to all the hearth difficulties and slipping which characterize difficultly reducible ores, or ores crushed to too large a size.

From the extreme of irregular and often dangerous practice characterized by an extreme proportion of fine ores, the practice improves as their proportion is reduced, the furnace finally exhibiting only occasional and infrequent symptoms of hanging. It is often difficult to mark the point at which poor practice begins, or the percentage of fines which definitely causes poor working, as the effect of the use of such material is usually cumulative, possibly not immediately producing a slowing up in production, offgrade iron, or heavy slipping. Although indeterminate, there is a point where further inclusion of fines in the burden is detrimental to both practice and safety, and among investments calculated to promote efficiency and improvement in practice and working conditions, equipment for the proper treatment of the ore, often only a small proportion of the total charge, holds out great promise not only of effecting the desired object, but also of returning interest on the investment.

SINTERING AND BRIQUETTING ORES AND FLUE DUST.

Many processes of sintering, nodulizing, and briquetting ores and flue dust are in use, both in the United States and abroad, which give products of varying strength, reducibility, density, purity, and cost. Their introduction among American works has been slow, largely on account of the possibility of using ores of rich iron content and good physical characteristics, and also on account of conflicting claims of promoters. Definite information would be of value to furnace men in enabling them to apply the most suitable of the several available processes to the treatment of flue dust and fine or concentrated ores. Irregularity in the chemical and physical composition of ores presents hindrances to uniform practice which have to be obviated to prevent slipping in furnaces. Physical variations introduce the most discouraging handicaps, as the abrupt irregularities such changes of burden introduce are extremely difficult of control, and the variable practice induced by variation in physical constitution must often be faced without opportunity for an adjustment of burden which will be of immediate avail. Irregular working due to changes in burden can usually be controlled between surprisingly narrow limits provided the furnace crew are watchful and prompt to take action. If variations in the appearance of the slag and iron, the brightness and liveliness of the tuyères, fluctuations in the pressure, movements of the stock line, and appearance of the gas are disregarded, the furnace will inevitably become beyond control, and can be saved only by rapid changes in burden, blast pressure, and volume, or the use of heavy coke blanks. In fact, given the best in design, construction, and material, the fundamental factor of success and safety lies in the interest and attention that the furnace crew manifest in their work.

FURNACE OPERATION.

VOLUME AND PRESSURE OF THE BLAST.

INSUFFICIENT VOLUME OF BLAST.

Insufficient volume of blast is seldom the cause of slips nowadays, it being more a feature of former practice where the blowing capacity of the engines did not keep pace with the increase in size of the furnace. A few years ago, when "slow driving" was for a time emphasized as a means of saving coke and promoting regularity in furnace practice, numerous plants experienced a period of heavier slipping than had been experienced for several years previous, when fast driving had been the slogan. Instances are sometimes encountered, however, of deficient volume of blast, for example, in blowing in a furnace where the burden ratio is increased faster than the rate at which the volume of wind is stepped up, or where a furnace sticks tightly and the blowing equipment does not deliver the desired amount of wind against the higher pressure. Serious difficulties are always introduced by this condition, as the low volume of air and gas in its ascent through the charge always selects the easiest path and it does not penetrate the more difficultly permeable parts. As a rule, this causes hot working near the walls, resulting in scouring cinder from an inner core of raw unreduced ore, which requires direct reduction and an excessive amount of coke, erosion of the inwall, and eventually a cold hearth, off-grade iron, and slipping are the results. If the ore descends faster at the center, with annular rings of unreduced and insufficiently prepared material coming to the smelting zone next the walls, this material may, in addition to the above results, attach itself to the walls and thus promote scaffolds.

EXCESSIVE VOLUME OF BLAST.

An excessively large volume of air in proportion to the hearth diameter or to the combustibility of the coke is probably responsible for a certain proportion of furnace troubles from hanging and slipping. Theoretically, with an increase in the blast the zone within which the combustion of coke is confined is higher, the melting zone is raised, and whatever erosion of lining normally occurs takes place higher up and forms profiles, which may prove detrimental to the resumption of normal practice. Scaffolding is made more pronounced at such times because with a higher blast volume and lower heat than on normal practice a limy slag is sometimes used in an attempt to keep the fusion zone down. The consequence may be that under the normal irregularities constantly encountered, the excessive basicity of the slag magnifies any tendency it may have to build on the wall or to erode the lining, erosion being especially promoted by the abnormal increase in temperature of the lower inwall section. A large volume of blast may also definitely retard the descent of the charge, and, added to any incipient tendency insufficient in itself to cause sticking, may introduce slipping, which would not develop with a normal blast volume.

IRREGULAR VOLUME AND PRESSURE.

Variations in the rate and pressure at which the blast enters the furnace have from time to time been seized upon as explaining the constant irregularities to which furnaces are prone. As regards constant volume, this is closely attained by the use of modern types of engine governors and of recording tacnometers and revolution counters to check the quantity of air blown and regularity of delivery. As a rule the only variations are chiefly introduced by varying moisture content and temperature of the air and by air leaks at the engine and mains. The effects of leaks are perhaps not serious when kept at a minimum, but if they are excessive, as indicated, say, by a consumption hour of 62 cubic feet of air per pound of coke, then if the pressure goes up because of hanging, the leaks increase the proportional amount of air lost. Consequently the furnace receives less air and generates less heat, although more heat is required in the hearth and there is a constant loss by radiation and tuyère water, and therefore the furnace gets cold very quickly. Before a furnace is blown in, the tuyère stocks should be stopped with blank flanges, an engine turned over slowly with the snort valve partly open, and then each stove be put on the line. Inspection of hot and cold blast valves, doors, bolted flanges, and keyed junctions will usually disclose a surprisingly large number of leaks. Hot-blast and chimney valves should be inspected once a week for air leaks. The delivery efficiency of the air tubs can be checked by cards and temperature observations.

After air leaks and piston slippage have been eliminated and the engines are running at uniform speed there remain moisture and temperature variations of the air at the blowing tubs. The use of air that has been dried eliminates these variations, but as furnace men are slow to adopt this method, the problem becomes one of obtaining the advantages of dry blast by other means. The expedients resorted to are to draw the air from outside the blowing room, obviating the high temperatures and moisture contents found in the blowing room, and increasing and decreasing the revolutions 2 per cent for every change of 10° F. in the temperature of the air. Both practices prevail to a limited extent and solve approximately the problem of delivering a constant weight of oxygen to the furnace. The effects of varying moisture, usually unimportant in cold weather, at times become detrimental to uniform operation in the summer. Theoretically an increase of one grain per cubic foot in the moisture content of the blast should cause a change of 25° F. in the combustion temperature and a change of 100° F. in blast temperatures should introduce a change of about 45° F. in the combustion temperature. The obvious remedy for variations in moisture content is to adjust the blast temperature to meet them. However, as other variations in furnace conditions may have an effect opposite to that of the moisture variations, introducing a mathematically calculated correction for moisture may cause more serious variations than the one which is supposedly being eliminated. Such an adjustment has been tried, but its use has not lasted long, as an experienced blower or hot-blast man can judge the necessity of changing the heat more quickly and accurately than can be accomplished by adhering to a calculated schedule.

DISTRIBUTION OF BLAST.

Severe slips, loss of ore dust, and similar troubles are often caused by the irregular distribution of blast at the tuyères. Apart from incorrect design of the hot-blast main and bustle pipe, irregular distribution of air may be caused by the accumulating of dirt in the bustle pipe, by channeling of descending stock by the blast, or by a tuyère opening being directly at the juncture of the hot-blast main and bustle pipe. Cases in which one side of a furnace is receiving more than its share of blast are characterized by vertical shearing of the column of stock, owing to faster driving on that side; the top of the stock column becomes sloping, being lower over the tuyères which are driving fast, thus increasingly disarranging the distribution on top; this distortion of distribution, the folding and shearing of the column, and channeling by gas cause the furnace to work stiffly and irregularly. The combined effect of fast and slow driving and of poor distribution must be combatted, the resultant sloppy or ironed-up tuyères, scaffolding, scouring cinder, slipping, and off-grade iron being alternately indicative of almost every possible cause of irregularity. A portable pyrometer should be used at regular intervals to detect this condition, as a higher temperature at any tuyère indicates that more air is entering that particular tuyère. If two or more tuyères are taking regularly and persistently an undue proportion of the blast, the size of their openings should be promptly reduced in order to equalize the blast distribution. The bustle pipe should be cleaned regularly, as large accumulations of dust result in a few months' operation when dirty gas is used at the stoves.

CHANNELING FROM THE BLAST.

Channeling over one tuyère is not infrequent and is easily detected from the appearance of the material in front of the tuyère; often unmelted scrap and lumps of unreduced ore and uncalcined limestone come down to the tuyère persistently. Channeling over one tuyère is not as serious as over two or more tuyères, nor are these conditions to be confused with the normal variations in the brightness and liveliness of different tuyères from hour to hour. Persistent channeling at some definite side has to be corrected to avoid erosion of the lining. When such channeling takes place directly beneath a gas offtake, it is sometimes due to excessive lifting of stock at the top, and to check it plugging the tuyères on that side for a couple of casts must be resorted to.

BLAST PRESSURE.

Variations in pressure are usual in normally working furnaces, even between furnaces of the same design working on the same burden and practice, and the pressure can vary over a somewhat wide range without apparently materially affecting the working or tending to cause slipping. The pressure of the blast is often held to have a direct bearing on the penetration of air into the center of the hearth. This is because, if the air is assumed to be delivered in constant quantity, the delivery of a given volume through a smaller area should result in a higher back pressure within the blast main and also in a higher velocity of discharge. That a change in tuyère diameter resulting in greater velocity has such a marked or beneficial effect as is indicated by the frequency with which this means is resorted to is doubtful. Penetration of the blast to the center of the hearth is of course essential, but rather than its being due to the pressure of a jet of air, it is more probably due in part to the temperature of the blast and of the hearth, as the higher the temperature is the more eagerly will the oxygen combine with the carbon and the further it will travel, and in part to the blast being supplied in sufficient volume and against sufficient pressure to fill all void spaces, tending to flow laterally before it disperses upward through the less permeable overlying material. A glance into a tuyère suffices to show that the incoming air does not drive direct to any one part of the hearth, nor in a straight line, but that it expands and flows into the cavities or voids in the coke filling the hearth.

No matter at how high a pressure a furnace is blown, if the burden is not properly charged and distributed the gases will find the channel of least resistance, and if this is up through the column of burden rather than through to the core of the hearth there will be little chance of uniform work. Forcing the blast pressure up to 16.5 pounds by inserting smaller tuyères can give only a semblance of the conditions at another furnace doing regular work on a larger tuyère area, where the blast pressure also averages 16.5 pounds. The essential condition for penetration by the blast is to have sufficient back presssure inside the furnace, and this is attained by correct charging. There is naturally a limit to the size of tuyères, a certain initial pressure of the blast is necessary, and an increase in this initial energy or velocity will increase the area of horizontal section reached by the blast. This is brought out by experience with wide hearths. However, in instances where insufficient penetration is regarded as the cause of irregular working of a furnace receiving the normal amount of air it is often found that the difficulty is in the arrangement of materials on top.

SELECTION OF TUYÈRES.

In order to effect penetration by the blast long tuyères have been used, 18 inches being the length usually substituted for the standard length of 12 inches. When a furnace tends to work toward the walls such expedients are useful in temporarily overcoming this condition. The cause is essentially some defect in charging, however, and the continued use of a long tuyère may easily introduce serious obstacles to smooth working, in that it produces a flatter bosh. Such distortion of the bosh contour affords a base on which the column of descending stock more easily comes to rest. The important bearing the length of tuyères has on the smoothness with which a furnace works may be illustrated by the fact that in one instance substituting 6½ by 9 inch for 6½ by 12 inch tuyères transformed a 16-foot 6-inch hearth furnace from a very stiff working to a regular working furnace.

This indicates also the marked effect the bosh shape has on smoothness. A steeper bosh affords less foothold for arching and scaffolding and probably does not jam the coke coming down to the tuyères as tightly as would a flatter bosh, thus permitting easier penetration of the blast to the center.

CHANGING THE TUYÈRES.

A few furnace men believe in a cycle of tuyère changing—for instance, changing from sets of 6 by 12 inch to 6½ by 9 inch tuyères, thence to 6½ by 12 inch to 6 by 9 inch and back to 6 by 12 inch. In this way, so it is claimed, the furnace does not settle down to a uniform method of working, in which hanging and slipping occur at approximately definite intervals. Seemingly such a procedure has considerable merit, because distribution is never perfect, and, consequently, there should always be a proneness to faster work in some perpendicular circle or ring. If the charge tends to work away from the walls excessively, changing the tuyères as indicated above would tend to make the furnace work hotter near the walls at times and prevent scaffolding. Conversely, if the tendency is to work on the walls, changing the tuyères may decrease the temperature near the lining often enough to check detrimental erosion. Obviously such tuyère changes have to be made with some appreciation of the needs of the furnace, as shown by its driving or smoothness of operation.

A method of combating stiffness which is decreasing in popularity is to blank half of the tuyères on the side opposite from where the presence of a scaffold is indicated. This practice is still followed at a few plants, and the time over which the tuyères remain blanked varies from four hours up to three days. Good results are claimed, but many furnace men believe that the practice introduces as many irregularities as it is supposed to eliminate.

There is naturally a tendency to resort to tuyère changes to correct irregular working and slipping, because there is still much to be accomplished before technical control of furnace practice is perfect, and those variables most under the control of furnace men are most often changed to cure indefinite and obscure troubles. On this account the crew often resorts to tuyère changes so frequently that the superintendent is inclined to avoid them, except under a definitely indicated necessity. For this reason a majority of furnaces are blown with the same size tuyère from about the second to the last month of the blast, the method of blanking opposite tuyères being resorted to, as a rule, only in event of irregularities amenable to treatment by variation in tuyère area.

PREVENTION OF LEAKS IN THE WATER-COOLING SYSTEM.

Leaks in the bosh-cooling system or at the stove valves are a somewhat frequent cause of irregular working and slips, and are often most difficult to detect. Leakage of water into the furnace may be indicated only by scouring cinder, cold gas in the bosh, sloppy tuyères, or slipping; at the same time it is difficult to detect gas by the discharge water. It is easier to detect gas if the pressure of water at the circle pipe supplying the bosh is carried at about 4 pounds less than the blast pressure. Under such conditions gas from inside the furnace will force itself into the plate and tuyère in event of a leak, and gas will show at the water discharge almost immediately. A small gage should be attached to the circle pipe so that the pressure carried can be checked.

That some defects in old linings are due to minute leaks on the cooling bronze is indicated by the fact that when plates from an old lining are tested, preparatory to use on a new lining, often as many as 50 per cent may be rejected owing to small cracks and pinholes, the existence of which could not be determined while on blast. Testing of all bronze, new as well as old, at 75 to 100 pounds pressure is essential to eliminate leaks. Possibly the best way to detect minute leaks is to immerse the plate or tuyère in water and apply an air pressure of about 75 pounds. The frequent analysis of the scrap used, as failure and especially cracking develops, is a valuable check on the probable behavior of the metal. The use of pure copper has proved of advantage in causing freedom from cracking and in increase of life, and analyses will sometimes develop the fact that deleterious alloying has crept in through mistakes in the use of material or scrap. Accurate alloy analyses are more difficult than the usual analyses in blast-furnace laboratories; and if more than traces of bismuth, lead, antimony, arsenic, or iron are found the analysis should be verified before the metal is condemned.

Certain sections of the cooling system, such as the tuyères and the first two rows of plates above the tuyères, are subject to the maximum effect of the combined action of the blast, slag, and iron, impinging on the surface, in addition to the temperature, and are more subject to cracking, burning, and leaking than other parts of the system. Some plants test the discharge water from these tuyères and plates for gas every afternoon in order to detect any small leaks.

The use of screens in the bosh-water supply line is of marked value in preventing bronze being burned from slackening or stoppage of the water supply by débris. Regular flushing of each plate, cooler, and tuyère with high-pressure water also helps to eliminate leaks, especially in the bosh plates, because the deposits of mud are stirred up and ejected. The discharge pipe over the trough should be in such position that the men do not have to go on the bustle pipe to see the water, but can see it easily from the cast-house floor. The employment of every means which facilitates watching, testing, and keeping the water supply and discharge going is urgent on the basis of efficient furnace practice as well as freedom from slips.

EFFECT OF WORKING AT HIGH TEMPERATURES.

The use of high heat—that is, blast temperatures upward of 1,050° F.—is frequently productive of continued hanging and sticking. This was originally attributed to the use of Mesabi ores. As much of the future supply of ores must come from the Mesabi district, and good practice and tonnage could be obtained with lower heats, 800° to 1,000° F., there has not been, until recently, any widespread efforts or facilities for applying higher heats. By changes in the lines of the furnace, in materials, and in slag, several plants have succeeded in operating at temperatures of 1,100° to 1,350° F. without continual slipping and hanging.

Recognition of the fact that the sulphur content of the coke and the basicity of the slags, or the lines of the furnace, either by design or by the use of low heats, or both, are unsuitable to readjustments in heat practice, will do much to eliminate excessive slipping due to high heat. Although examples are not lacking of furnaces hitherto on low heat being worked into the use of high heat, it has invariably been the result of gradual increase in heat for weeks and months, accompanied by a progressional increase in burden and decrease in slag basicity, with repeated setbacks and stiff furnace conditions. Attempts to rush the furnace to higher temperatures may be set down as ineffectual, as the use of high heat involves such changes in practice, considerations of profile of lining, and improvements in distribution that, unless these are obtained, pronounced slipping will result.

COMPOSITION OF SLAG.

Unsuitable slag composition is a prolific cause of hanging. There may be, and probably are, several complex reasons for this, but one evident reason is variations in temperature. If the furnace is run at a higher heat the coke consumption per ton of burden is lower. Consequently there is less ash to be fluxed and a considerably decreased volume of slag per ton of iron. With this smaller volume persistent variations in the charge will produce a wider variation in the slag than with the former large volume, so that it may easily occur that a swing toward increased basicity or acidity will produce an exceedingly variable composition of slag, which a higher slag volume would take care of without showing as great a change or without difficulty or detrimental results. This liability to the formation of excessively basic slag when a limy slag is carried in low volume on high heat is a deterring factor in the use of limy slag. An insufficient volume of slag to cope with the variations in the burden material will, of course, cause sticking and slipping, even at moderate heat, but even more so on lower slag volume and high heat. Such a condition is not infrequent, and is characterized by the statement that the furnace is too "dry." Trying to keep the slag volume down, when high-sulphur coke is charged, by the use of limy slag is likely to cause hanging, if a variation in materials or charging increases the proportion of basic material in the slag. There are sometimes many perplexing factors to be adjusted in burdening a furnace, especially where the situation is complicated by such problems as abnormally high or low alumina content, excessive sulphur content, the proper proportioning of available fluxes, the relation of the basicity or acidity of the slag to the silicon and sulphur content desired in the iron, or production of slag for commercial purposes. Usually recourse must be had to previous practice, or, if that be lacking, to a series of cut-and-try experiments. The attainment of the correct slag is not always assured, and such uncertainty is not conducive to good work, especially when the slag does not seem to be all that might be desired, and consequently there is naturally a tendency to make changes. Some of the difficulties incident to the use of certain ore mixtures or to the adjustment of practice to operating handicaps is undoubtedly due to the unsuitable composition of the slag. It must be admitted, however, that at times the slag is made the scapegoat for various troubles, such as error in charging, deficient distribution, etc., to which it has not the least relation. To correct such irregularities the slag is made limy or sharp or the alumina or magnesia content is altered, when any of these factors actually are at the most only remotely responsible for the behavior of the furnace.

STOCKING AND CHARGING.

Much can be done toward insuring uniform and regular operation, with freedom from slipping, by care and thought in stocking and charging. The widely varying analyses and fineness of charges from which many of the Mesabi mixes are built up, the fluctuations to which the analysis of so many "old-range" ores are liable, and the certainty that wide variations in quality will be encountered in the ore during a year's run, require as much care in handling ores as in other items of practice.

Furnace troubles are usually more numerous in summer than in winter and at plants having inadequate stocking space for winter consumption. In using ores direct from the cars during the summer months many of the difficulties are attributed to proved variations in the shipments. Plants formerly dependent upon direct shipments have effected marked improvements in practice by providing stocking space sufficient for winter and also summer requirements, the ores being handled through the stock pile. This eliminates the annoyance of frequent changes of burden, spreads cargo variations over a wider period, and removes the necessity of steaming, thawing, and dynamiting in winter. Sufficient space should be insured to avoid any necessity for mixing of ores. The practice of stocking together ores that are similar in physical character but differ in composition or, more frequently, are similar in composition but differ in texture, is often made necessary by the limited stocking facilities. In so doing, the expense and advantage of selected mixing of ore at the mines is lost, and occasional serious irregularity in furnace operation may result.

Narrow stockyards, where capacity is obtained by extension beyond the limits of the space occupied by the stove and furnace, require stocking by dumping on the apex of the heap. When ores are stocked in this way the pile contains a high proportion of coarse material near the floor, grading up to the finer portions on top. Where possible each shipment should be dropped over the surface of the stock pile evenly, so that the grade of ore will be more uniform throughout the season, and also from hour to hour. After uniformity in stocking has been obtained the ore should be taken off evenly and uniformly from the top, rather than removed in deep local sections or cuts.

WEIGHING AND LOADING.

The stock house presents many opportunities for the attainment of the best practice, or for an utter loss of such a possibility. In hand-filled furnaces accuracy in the weighing of charging buggies should be required, on account of the large number of weighings per charge. The use of larry cars may reduce the number of weighings per charge to two, and the less the better, as the chance for error in weighing is reduced and the furnace may be more easily kept full. Utilization of the full capacity of the bins seldom requires urging. When the maximum amount of bin space in proportion to the percentage of ores used has been provided, the superintendent should insist on drawing in turn from the various bins of each grade. Possible sharp changes in quality are thus spread over a longer period and the change made more gradual.

The car operator is more likely to do this work correctly if he has the proper facilities, and in view of the importance of his work every detail should be carefully worked out. The ore and limestone should be under perfect control while being withdrawn from the bin. Chutes should be closely spaced to preclude tendency toward sticking in the bin, and the contour of bins and chutes should be designed to promote movement of the entire contents. Few things are more discouraging to correct weighing than difficulty in getting the ore to run, while the furnace is continually keeping ahead of the stock-house force. Efficiency in weighing is promoted by equipping the plant with larry cars with automatically recording scales, from which the weight of each component of every charge can be checked. It is unreasonable to expect a man to do his best under unfavorable conditions. A dark, cold, windy, dirty, wet, or dusty stock house is not conducive to careful work in weighing.

CHARGING COKE.

The essential feature in charging coke is to obtain the minimum breakage from the cars to the furnace and to avoid charging dust and breeze. For this reason the location of one or two large bins in proximity to the skip is preferable to many small bins from which the coke must be dropped and conveyed to the skip by a larry car. The accumulation of fine coke dust and breeze in the coke pockets, owing to the profile of the bin not conforming to the angle of repose of the coke, is more obnoxious than pockets of fines in ore bins, because delay in spotting coke drags on the trestle, with consequent drawing on the dead pockets, with their accumulation of fines, is certain to cause furnace trouble. Regularity in spotting and dumping coke drags contributes markedly to regularity in furnace work, as was shown at one plant when an attempt to reduce labor costs by eliminating night unloading of coke in pockets which were of inadequate size and poor design led to most unsatisfactory work, and return to smooth furnace work was obtained only by frequent unloading at night as well as by day and by keeping the height of coke in the bin above a fixed limit.

SIZE OF COKE UNIT CHARGED.

The adjustment of the size of the coke unit charged depends on the size of the furnace, character of the burden, blast pressure, distribution of stock in the furnace, limitations of the stock-house system, and also at times upon indeterminate factors so obscure as to lead to the belief that individual furnaces possess characteristics and perversities of their own. The practice of furnaces working on cokes of open structure and rapid combustibility has been at times improved by increasing the size of the coke unit. Increases in volume of the coke unit of up to 50 per cent have effected an increase in pressure of 2 to 4 pounds, thus counteracting the tendency toward a drop in pressure peculiar to fast-burning cokes. Unsatisfactory work, where fast driving was the practice, has yielded in several instances to an increase in the coke unit. Moderate driving, in general, has been most successful with coke units of such size that approximately 85 to 90 units are required per day; in fact, the proportioning of the most advantageous unit of burden may depend more upon the selection of a weight that requires nearly constant charging. than upon any other factor.

The size of the coke unit at furnaces producing 500 tons of iron per day, and of about 22,000 cubic feet capacity, varies from 7,500 pounds up to 16,000 pounds, and satisfactory work is done over this range. The extremely large coke units are in a general way confined largely to furnaces which are more notable for production records than for fuel economy. A coke unit of 9,000 to 11,000 pounds seems to give the best fuel economy and smoothest work on furnaces of the capacity mentioned.

SEQUENCE OF CHARGING COKE, ORE, AND FLUX.

Except with a certain type of distributor, it is not found productive of regular work to dump the coke, ore, and limestone with one drop of the large bell.

In this case a large coke unit is used, about 15,000 pounds, but it is split as follows: Two skips of coke, one of ore, two of coke, one of ore, and two of limestone, making seven skip loads in all on the large bell, which is then lowered. A more common practice at present than dumping all the materials of a charge at one lowering of the large bell is to split the charge as follows: Seventy-five thousand pounds of coke is charged and the bell lowered; next 15,000 pounds of ore and 3,300 pounds of limestone are charged and the bell lowered; then 7,500 pounds of coke, 15,000 pounds of ore, and 3,330 pounds of limestone are charged and the bell lowered. Other plants make two drops of the bell, charging the coke in with one drop, and the ore and limestone in with a second drop. This method of filling places the fuel and the burden in separate strata, and the coke, being comparatively open in comparison to the ore, should allow the gas currents to equalize themselves in each successive layer, and should prevent the formation of chimneys up through the ore. Channeling by gas is detrimental, and the fact that experimental changing from two drops to one drop of the large bell usually results in irregular work and slipping lends a certain plausibility to the theory. As the volume of the charge of ore and stone is roughly one-half that of the coke charge, the minimum size of coke unit is reached when the small volume of ore will not cover the fuel. If the ore charge tends to work toward the wall, the limit in decrease in size of ore unit is reached much sooner than if it were spread out in a level or nearly level layer over the top surface of the coke. On a 16-foot stock line, 16,000 pounds of ore and 4,400 pounds of limestone is about the smallest charge that will work satisfactorily.

Theoretically, the thicker the layers of ore and coke the better stratified are the coke and ore charges. The reason that the limits are narrow for a given furnace is, probably, that a thick ore charge may tend to channel in the deep-ore blanket, or that with an excessively large weight of ore charged as a unit, the annular heap next the wall becomes too thick and dense in comparison to the depth in the center. After the most advantageous weight of ores for a charging unit has been fixed upon, the method of dumping the components of the charge offers many possibilities, and, similarly, the sequence of charging the various ores. The combination of these two factors affords a considerable range and inasmuch as they are among the most obvious means of correcting unsatisfactory operation they are frequently resorted to, often with little effect and to the further detriment of furnace conditions. In general after the operator has determined by previous practice, or by trial during the early period of blast, the best method of operating the bells and the best sequence of ore, coke, and limestone, there is little to be gained by further experiments, and the necessity for the change should be evident before it is made.

The sequence of ore, coke, and limestone does not seem to have any definite relation to furnace operation over a large number of furnaces, although for a given furnace the sequence is seemingly very vital. For instance, at one plant the coke charge is followed by one-half the limestone, then the ore, and then the other half of the limestone; the limestone and ore being dumped by one drop of the large bell. After a period of irregular working, the limestone was all charged last, following the ore, but this sequence caused such irregular work that the operators went back to the original procedure. One large plant charges the limestone immediately after the coke, following this with the ore. As it was suspected that this method of dumping was causing trouble, the limestone was dumped with the coke, and the ore by itself. Inasmuch as no improvement could be noted, and in fact, the furnace required more coke, the operators went back to the original plan.

As to whether the limestone should be charged ahead of the ore, or after the ore, both being lowered into the furnace with one drop of the large bell, there is the most divergent opinion, the majority of plants charging the limestone after the ore. There have been several instances where dumping the ore, coke, and limestone with one dump of the bell resulted in very poor operation, whereas at another furnace in the vicinity the charge may be split and two or three dumps of the large bell made, each lowering of the bell charging in, say, 5,000 pounds of coke, 12,000 of ore, and 3,000 of limestone, with good results. At most plants each round contains all the components of the ore burden, as follows:

Example of typical charge containing all components of burden.

Constituent.	Number of pounds charged.
Ore A	4,000
Ore B	10,000
Ore C	6,000
Ore D	2,000
Cinders and scale	2,000
Total	24,000

A variation practiced by a considerable number is to alternate "D" ore and "cinders and scale," 4,000 pounds of each being charged in sequence every other round. Another variation is found in setting the scales to charge 7,200 pounds of "C" ore for 5 rounds; on the sixth the "C" ore is omitted and 12,000 pounds of "cinders and scale" is charged. A practice which is somewhat frequent is, assuming that a siliceous ore makes up about 10 to 15 per cent of the burden, to charge the siliceous ore every second, fourth, or fifth round. Obviously a considerable number of similar variations can be worked out for the same mix, and although such variation may seem a very minor factor, or, on the contrary, opposed to regular operation, considerable significance is attached to these small adjustments by the men whose ideas and practice they are. A wide deviation from theoretically uniform charging is represented by the following charge:

Charge showing wide variation from theoretically uniform charging.

Rounds.	Materials.	Number of pounds charged.
First	Coke	12,000
	Ore A	16,000
	Ore D	8,000
	Limestone	6,000
Second	Coke	12,000
	Ore B	24,000
	Limestone	6,000
Third	Coke	12,000
	Ore B	16,000
	Cinder and scale	8,000
	Limestone	6,000
Fourth	Coke	12,000
	Ore C	24,000
	Limestone	6,000

This plan is not actually followed, but is representative of one where successful and uniform work is obtained.

The coke unit, being made up of one component, is subject to little variation. However, many experienced furnace men believe that much better work is obtained if, instead of charging 24,000 pounds of ore and 12,000 pounds of coke per round, 25,500 pounds of ore and 12,000 pounds coke is charged for four rounds and in every fourth round an extra skip of coke, 3,000 pounds is charged.

The general idea behind such variable charging is that the furnace is benefited and hanging prevented, by giving the furnace a periodical "jolt." Another frequent means of "livening" up a furnace is to change the size of the charge or sequence of charging. That in this way scouring of the walls is somewhat promoted is undoubtedly true, and in the event of conditions trending toward slowing up of rate of driving or hanging, some adaptation of the above method works toward elimination of the sticking and stiffness developing into heavy slips. Although these expedients are productive many times of smoother work, it is probable that even the better practice resulting is not as good as could be attained were the distribution, the lines of furnace, and the materials used more closely aligned with the requirements.

INSPECTION OF THE BELLS.

The bells should be inspected at least weekly to insure that sticky or wet ore is not adhering or baking on them. Machined bells with an angle of 53° to 55° give considerable relief, but offer no assurance that a scab is not interfering with distribution. Such lumps can, as a rule, be barred off through the gas-seal doors.

When the bell is hung it should be tested and balanced before placing in the furnace. An unbalanced bell may close off center on account of the underbalanced side coming in contact with the lip of the hopper or hopper extension while the heavier side is still in motion upwards. The result is that there is a horizontal displacement of the bell toward the overbalanced side, with a resulting tendency to side swinging when the bell is lowered. A similar thrust or swing occurs when the bell is off center, either by reason of the point of suspension being shifted relative to the center of the furnace or of incorrect alignment of the bell in placing it. The bell should be exactly centered when it is placed in the top. This is an almost rudimentary requirement, but lack of attention to this point is not unknown, and it usually results in many discouraging irregularities and difficulties during the early part of the blast. Attention to balancing and centering the bell so that it closes simultaneously at every point about its periphery repays many times the time required.

HEIGHT OF STOCK LINE.

The distance the stock line may be carried below the bell has a marked effect on the distribution. Carrying it to the maximum height permitted by the drop of the bell results in an annular ring of finer material with a base toward the wall and in the center of coarse material. Such distribution, which is productive of channeling, may also be caused from the use of too small a bell, and then conditions may be improved by dropping the height of the stock line until the velocity of discharge from the bell carries the material farther out toward the periphery of the furnace. The effect of carrying the stock line too low is to cause excessive abrasion of the lining at the stock line by impact of material and also to shift the apex of the stock discharged from the bell to a point at the walls, or even, theoretically, beyond the walls. Too low a stock line, while affording an increased height in which greater opportunity is afforded for dust to settle before being swept over into the downcomer by the velocity of the gas is prejudical to good work, in that the abrasion of the walls with accompanying breakage of coke, packing of stock by reason of concentration of fine material next the walls, and undue permeability of stock at the center presents too much certainty of scaffolding.

Realizing that the velocity with which the various materials are discharged from the bell will determine the apex of their lodgment upon the stock column, and that this velocity will depend upon their weight, character, and freedom of movement, the aim of the operator should be to select such a height of stock line as will bring the points of lodgment in as close vertical alignment as possible. This can be determined only by inspection of the stock line after each lowering of the bell while filling preparatory to blowing in, and it depends so wholly upon the ratio of diameter of stock line and bell and upon character of ore that no rule of practice is possible. Access to the interior of the furnace while it is being filled should always be provided for. The discharge from the bells should be watched, and as the furnace is filled up and the stock line can be reached with a ladder from a gas offtake or down through between the bell and hopper, the distribution of materials should be examined and the contour of the top of the stock column measured. This information goes a long way toward eliminating questions which come up during the blast, where the furnace gets to slipping persistently. It also helps in determining the height at which the stock line should be carried for the best distribution. To dump large amounts of ore, coke, and limestone into a blast furnace without knowing how it lies in the furnace is leaving too much to speculation and is poor practice. The height of the stock line is sometimes held within such narrow limits by the relative size of the bell and throat of the furnace, the distance between the center line of the gas offtake and the bottom of the bell, the speed of lowering the bell, and the character of materials, that little leeway is allowed, but, nevertheless, there should be a definite idea of the distribution.

USE OF STOCK-LINE INDICATORS.

Keeping the height of the stock line constant is important. The fact that fluctuations in height will be encountered affords no excuse for irregular filling. The location of a stock-line indicator, preferably of the recording type in the stock house, gives the least excuse for intermittent charging. The latest type of rodding devices are worked by air, steam, or water pressure, and these are to be recommended, as they remove any disinclination toward frequent rodding on account of the effort required by older devices. It is a good plan to place one of these rodding devices in the cast house also, so that the blower can keep himself informed as to the condition of the furnace and the regularity of charging. Intermittent charging, varying heights of stock column, and haphazard knowledge of the condition of the furnace not only lead to slips but may make them more serious in results.

SPEED OF OPENING BELLS.

The speed of opening the bells should be kept constant, as the velocity of discharge, the distance to which materials are thrown from the bell, and hence the distribution, depend upon the rate with which coke or ore is made free to move.

IMPORTANCE OF PROPER DISTRIBUTION.

Considerable space has been given to the factors governing distribution of materials in the furnace. This is because correct distribution is a chief factor in the elimination of slips. If the proper volume of blast is used and the contour of the furnace lining is designed for the materials and heat carried, then most of the time, provided the charge is properly distributed, the furnace will work properly, but if the distribution is poor, the furnace will not operate satisfactorily very long at a time.

USE OF COKE BLANKS AND "DOSES."

When a furnace gets to slipping heavily or persistently a widespread practice is to charge a blank of coke. This blank may be accompanied with other materials which are calculated to scour the walls. Examples of various blanks, or "doses," for a 500-ton furnace are given below.

One blank extensively used in the Pittsburgh district is as follows:

60,000 pounds of coke,
10 regular rounds,
10,000 pounds of gravel,
10,000 pounds of converting mill cinder,
5 regular rounds,
50,000 pounds of coke.

Such a blank is in some instances put in a given furnace every four days, whether or not it appears to be needed.

It can be seen that this is an expensive procedure. About 55 tons of coke is charged, which, at $3.50 per ton, gives an additional cost to be carried of nearly $50 per day. There is little iron recovery from the cinder charged in the blank. For the $50 thus expended there is a return in more regular operation, freedom from excessive slipping, and a minimum production of off-grade iron. From comparative practice it is considered cheaper to spend $50 for a blank every four days than to have a "stiff" or slipping furnace, with its accompanying showers of material over the yard and loss in quality and quantity of iron.

A so-called "relay dose" or blank is as follows:

"Relay dose."

Material.	Pounds.
Coke	16,000
Scrap	4,500
Coke	12,000
Heating-furnace cinder	4,500
Scrap	4,500
Coke	16,000
Heating-furnace cinder	4,500

Material.	Pounds.
Sand	2,000
Coke	12,000
Scrap	4,500
Coke	16,000
Heating-furnace cinder	4,500
Coke	12,000
Scrap	9,000
Coke	16,000
Heating-furnace cinder	9,000
Coke	12,000
Scrap	19,000
Coke	16,000
Heating-furnace cinder	4,500
Scrap	4,500
Sand	4,000
Coke	12,000

This so-called "relay dose," which is used in furnaces apparently badly scaffolded, though it appears more formidable than blank 1, in reality contains but 30,000 pounds more coke. In proportion to the coke there is much more unfluxed material, heating-furnace cinder, and sand. The presence of heating-furnace cinder, which is simply a fused silicate of iron, is apt to produce a cast of off-grade iron unless the "dose" comes down very hot.

A third dose is as follows:

Material.	Pounds.
Coke	50,000
Converting-mill cinder, two regular charges	40,000
Coke	25,000
Converting-mill cinder	20,000
White iron (remelt), two regular charges	50,000
Coke	25,000
Converting-mill cinder	20,000
High manganese (6 per cent) ore	20,000

When this "dose" comes down to the bosh, the practice is to lower the heat from 1,100° to 800° F. This "dose" has sometimes proved deficient in coke in the event of its dislodging a heavy scab on the walls.

A dose that is used for a furnace badly scaffolded, and is not used except under very bad conditions of furnace operation, is as follows:

100,000 pounds of coke.
19,000 pounds of siliceous (40 per cent) ore.
One regular charge.
10,000 pounds of coke.
9,500 pounds of siliceous ore (40 per cent).
9,500 pounds of manganiferous ore (4 per cent).
Two regular charges.
35 tons of scrap and remelting iron.

Fifteen regular rounds.
50,000 pounds of coke.
19,000 pounds of manganiferous ore (4 per cent).
Ten regular rounds.
15,000 pounds of coke.
Five regular rounds.
10,000 pounds of coke.

At one plant, when the furnace is very much built up, the following is charged: 33,000 pounds of coke in a blank, followed by 6,650 pounds of coke, 3,000 pounds of ganister, and 3,000 pounds of limestone. This small unit is repeated five times, and is followed by 6,650 pounds of coke, 3,000 pounds of ganister, and 1,000 pounds of limestone. This is also repeated five times, and is followed by 4 regular charges; then 10,000 pounds of coke and 20,000 pounds of scrap and iron are charged, each unit being repeated five times. Then 10 regular charges with 20 per cent less burden, 15 regular charges with 15 per cent less burden, 10 regular charges with 10 per cent less burden, and finally the regular burden are charged. As soon as this blank shows up in the cinder, the engine speed is reduced about 20 per cent and the temperature lowered as much as possible, which is usually about 500° F. The furnace is driven in this way for about six hours; then the blast and temperature are gradually brought up to normal, which takes about six to eight hours' work. Mill or puddle cinder is sometimes substituted for ganister, with good results. This "dose" has proven itself very effective.

Other "doses" that have been used are given below:

Material.	Pounds.
Coke	15,000
Scrap	5,000
Coke	10,000
Converting-mill cinder	4,500
Coke	20,000
Pig iron	5,000
Heating-furnace cinder	4,500
Sand	15,000
Coke	10,000
Coke	10,000
Scrap	5,000
Coke	15,000
Puddle cinder	5,000
Coke	10,000
Scrap	5,000
Coke	15,000
Puddle cinder	5,000
Coke	10,000
Scrap	5,000

Most furnace men use a blank or "dose" from time to time to regulate the working of the furnace, and as each superintendent has his own preferences and carefully worked-out methods of "dosing;" there are almost as many varieties of these as there are plants. Many keep the make-up of the "dose" secret, as they have worked it out by considerable experience, and regard it, properly, as part of their stock in trade. Those "doses" which are presented here are given with the consent of those furnishing the information.

The use of coke blanks is very common, and the amount of coke, at the judgment of the furnace foreman, varies from an additional round up to five rounds of coke in one blank. Many use limestone with the blank to heat up a cold, stiff-working furnace. Sand is not much in favor as a constituent of a "dose," as it is believed to cut the lining. Sandy, rough cast-house scrap is much in favor, but scrap from a granulating pit and steel scrap made up of borings, punchings, etc., is avoided. One furnace man, in attempting to use a considerable amount of wire-nail scrap in a "cleaning dose" had to pull a large proportion of it, unmelted, out from in front of the tuyères. In general, the idea in making up a "dose" is to have it include two materials of different refractory characteristics, so that it will melt over a wide range. Scrap, siliceous and manganiferous cinders, and ore are the essential components. When used as a panacea for all ills about furnaces, the employment of "doses" may be abused, in that their frequent use dulls the need of being continually on the lookout for and determining what actually is causing the hanging and slipping.

Many furnaces operate for months at a time without a dose, as an occasional small coke blank or a frequent adjustment of the burden carries them through without trouble. A stage midway of this type of practice is to substitute twice or three times a day a dose of siliceous manganiferous cinders for the ore burden. Thus, one plant charges, instead of a unit of 20,000 pounds of ore, 15,000 pounds of puddle cinder, gravel, and scrap three times a day. This is simply a development of the idea of charging the siliceous component of a charge every second to eighth round.

As a furnace gets old and the lining cuts out, the number of coke blanks is usually increased. These are often necessary because of slipping, the aim being to replace in larger proportion the coke thrown out by the slip, and also to interpose a blanket of extra coke when filling must be started 10 feet or more below the stock-line level. Often a furnace which has cut excessively above the mantle and sticks and slips persistently has been given a new lease of life by charging an extra round of coke three or four times a day at regular intervals. Instances have been noted where this extra amount of coke enables the furnace to take more burden and a heavier blast, and work with vastly less slipping.

The need for a coke blank or "dose" is not always easy to determine. Often a furnace can be worked out of a period of sticking and slipping by using a cold blast, checking the blast, or lightening the burden for a few rounds. When this can be done, with a good reserve of heat in the stoves, it is undoubtedly better practice than the custom of resorting to a blank or "dose" upon slight provocation, because the use of these may sometimes throw the furnace off. The use of "doses" is perhaps justified under circumstances where excessive slipping must be avoided on account of weak tops or shells, deficient heating, or large explosion doors, or when it becomes necessary to change somewhat frequently from one grade of iron to another.

METHODS OF OVERCOMING SLIPPING AND HANGING.

"BLOWING DOWN."

One old method of removing scaffolds was to "blow down" the furnace until the top of the stock was below the scaffold. A typical method was to start by reducing the amount of ore and limestone charged by about 20 per cent, and at the same time decreasing the limestone content of the burden one-half for every second charge. After 4 to 6 hours the furnace would be allowed to work down slowly. About every half hour a round would be charged in order to keep the top temperature down, though, as a rule, these rounds consisted chiefly of coke. The blast would be reduced gradually until, with about 20 per cent of the pressure off, it was no longer possible to keep the furnace cool on top; then a spray would be introduced. By this means the furnace would be "blown down" till the top of the stock column was alongside of or beneath the scaffold. Sometimes charging would be resumed and the furnace kept at this level until the scab had melted off, or the furnace would be filled with coke, scrap, sand, and cinder and "blown down" again.

These efforts were not always rewarded with success, as the scaffold was not always melted off and several bad explosions have occurred. The method of "blowing down" has not been followed to any extent in recent years, although it crops up at infrequent intervals.

USE OF AUXILIARY TUYÈRES.

A method of combating scaffolds that was formerly much in use, but is seldom resorted to nowadays, consists in inserting an auxiliary tuyère in the lower inwall of the upper bosh. This so-called tuyère consists of a cinder monkey, water cooled, or even a 1-inch gas pipe, which is inserted in a plate opening in the bosh or stack, or in a hole drilled in the shell and lining especially for this purpose. Through this auxiliary tuyère is blown a fairly strong blast of air, which starts combustion of the coke high up beneath the scaffold. The high heat thus generated melts off the scaffold, and if one is fortunate enough to find the right location at the first attempt in placing the tuyère, the furnace quickly cleans itself and starts to drive better. Recently a furnace resorted to this means of removing a scaffold which was apparently causing heavy slipping. A stack cooling plate above the mantle was removed and an auxiliary tuyère was inserted, then the tuyère was moved up gradually to each successive plate. This resulted in the scaffold being dislodged and in better work, but the improvement did not prove permanent, as the scaffold soon reformed and such heavy slipping developed that finally an unusually severe slip dislodged the hopper and bell, which was bolted to the top of the furnace.

Some years ago many plants were provided with about six permanent auxiliary tuyères above the mantle, which were put into use whenever the furnace showed signs of sticking. The use of auxiliary tuyères has been discontinued, but at several plants may be seen patches on the lower ring of the stack jacket where auxiliary tuyères were formerly located.

The reasons that the use of these permanent auxiliary tuyères did not prove feasible are that upkeep was difficult, and that the cooling of the walls at this point, so near the inside face of the lining, possibly caused as much scaffolding as the tuyères themselves were supposed to prevent; improvement in furnace lines and in distribution over former standards is an additional cause.

The fact that the temporary auxiliary tuyère is only infrequently used is probably due to the difficulty of locating the proper place to insert it, the questionable expediency of cutting a series of holes in the ⅝-inch, or thicker, steel shell, the possibility that any relief will be only temporary, and the much lessened tendency toward severe scaffolding.

USE OF EXPLOSIVES TO DISLODGE SCAFFOLDS.

At present the chief use of explosives at blast-furnace plants is in removing hearth bottoms at relining, and this has been discussed in detail on pages 175 to 182. In former years the use of dynamite to dislodge scaffolds was frequently resorted to, and considerable mention of the method is to be found in issues of the Transactions of the American Institute of Mining Engineers as far back as 1881. At that time many of the blast furnaces had no steel shells inclosing the brickwork, but were simply banded, much as the modern bosh is banded. This left a considerable space of uncovered brickwork, through which holes could be drilled where the scaffold was suspected to be, and then a charge of dynamite could be inserted and fired. The dynamite was sometimes inserted in the body of the scaffold, and sometimes in the empty space beneath the arch, and fired. This method was very successful, but the development of the modern thick steel shell has obviated its use almost completely, and in several years of practice with Mesabi ores the writer has never seen it used. The method was in use in Germany up to a recent date, and is described by Schöneweg[a] as follows:

> A tube welded together at one end, 2½ inches in inside diameter and 8¼ to 12 feet in length, is used as a protecting tube; and another snugly fitting tube of the same length and of 2 inches inside diameter is used for the charge. Into this inner tube is drilled a hole for the fuse at a distance of 3 feet from the mouth, when one does not want to use electric ignition. After the inner tube has been closed at one end with a wooden plug and the charge has been introduced into the tube, the upper empty space is filled with sand and the mouth is closed with a wet clay plug. The empty protecting tube is introduced at the proper place into the furnace, where brickwork must be provided with a sufficiently large hole; the fuse is ignited and the loaded tube with the burning fuse is shoved into the protecting tube. Every explosive may be used. If we use dynamite and the protecting tube, on being introduced into the furnace, has become red-hot, it must be cooled by allowing water to flow into it from a hose before shoving in the inner tube. Otherwise it may happen that the nitroglycerin may melt, explode, and throw out the charge. One may also use any of the explosives that are safe to handle, the pulverized as well as the gelatinous. As a charge one may safely take 2¼ to 2¾ pounds, only the charge must be at a distance of at least 1 foot from the brickwork.
>
> A furnace had not been driving well for five days. Through the opened tuyère one could see clearly an arched, closed scaffold, and a space 23 feet in height in the lower part of the furnace was entirely empty. During the stoppage 25 tons of coke had been thrown into the lower part of the furnace. We had eight holes drilled into the bottom of the scaffold. After the firing of the eight shots the furnace began to draw well. After two hours we noticed that the furnace did not work as well in its upper part. It was stopped once more, and we found that the holes had been drilled too high. New holes were drilled; 15 tons of coke and 2,000 pounds of tap cinder were thrown in, whereupon we blasted. This time the scaffold collapsed, the furnace descended uniformly, and after 24 hours it yielded iron for the steel works.

In all, 12 descriptions of the use of dynamite are given in the article from which the above extract is taken.

OTHER MEANS OF PREVENTING SLIPS.

By far the most common means of preventing slips from assuming a violent character or of preventing hanging and sticking from working into protracted and heavy slipping are lowering the temperature of the blast or reducing the volume of blast momentarily

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[a] Schöneweg, Herman, Das Schiessen in Hochofenansätzen: Stahl und Eisen, Jahrg. 34, 1914, pp. 1333–1336.

(checking); other expedients resorted to are increasing the slag volume and increasing the amount of limestone in the burden. Some operators decrease the revolutions about 5 per cent for an hour or so at the first indications of the furnace becoming stiff or tight. The avoiding of soft, punky coke and fine, easily reducible ores is frequently urged as a means of avoiding slips; but from the fact that most furnace operators are allotted a given variety of ores at the beginning of each season the range of adjustment of burden to include more refractory ores is limited. The given allotment must as a rule be consumed during the ensuing year, so that practically the only alternatives are, with certain furnace lines and distribution, to adjust the size of the charge and method of charging, the tuyère length or area, the volume and pressure of the blast, and the blast temperature to the best average practice, and to meet the occasional wedging and scaffolding of the stock by the use of coke blanks, "doses," and the manipulation of blast pressure or temperature in emergencies to bring any abnormal stiffness to an end. These represent the available means, after design, construction, materials, and average practice have been fixed and the problem of regular operation is at hand.

MANIPULATION OF BLAST PRESSURE AND TEMPERATURE.

As a general rule, if hanging develops it is considered advisable not to check the furnace, if this can be avoided. If the furnace will slip of its own accord within 30 to 60 minutes, many prefer to allow it to do so, because if the blast is partly or completely shut off to induce a slip the falling column is apt to pack tightly, and by wedging itself against the walls cause high pressure, repeated hanging, and heavy slipping; but if it falls against a strong ascending gas current its descent is obviously somewhat checked. If in one hour's time a hanging furnace does not settle voluntarily, then it must be made to settle. The longer a furnace hangs the larger is the cavity formed and the greater is the risk of an explosive slip, particularly if the furnace should slip with the blast on. As the cavity enlarges and the pressure of the gas within it increases, there is increasing probability of a violent eruption as the gas bores its way through or about the falling material. Therefore after 30 to 60 minutes steps are usually taken to slip the furnace, it being flushed before check to prevent cinders coming back in the blowpipes, unless there is hanging directly after the cast.

Reducing the blast temperature, say, from 1,050° F. to 900° or 800° F. will often loosen the furnace—that is, cause it to slip. However, it is not always desirable or possible to reduce the blast temperature, and the use of too much cold air, repeated too often, may result in off-grade iron.

If using a colder blast does not cause the furnace to slip, then the furnace must be checked. As a matter of fact, except after casting or in case a furnace is hanging stubbornly in the bottom, checking is resorted to more frequently than the use of cold air. The first check is almost never made sooner than 30 minutes after hanging develops, and then the furnace, if it is necessary to check repeatedly for some hours to keep it moving, is checked at not less than 40 or 60 minute intervals. Checking at 20 or 30 minute intervals almost invariably stiffens the furnace, and if it becomes necessary to check as frequently as this, it is a sign that emergency measures will soon be or are necessary.

Reducing the blast pressure at the tuyères 3 to 6 pounds is usually effectual in slipping the furnace. If it does not slip, the blast is usually turned fully on for perhaps 20 minutes and then checked again, and a second or even a third check may follow in close succession, provided each preceding check does not slip the furnace; the second check is usually all that is necessary, although if hanging persists, a third check, the pressure being reduced 5 or 6 pounds, frequently becomes necessary, or the snort valve may be opened, thus taking the blast off altogether.

The degree to which checking is resorted to always depends upon the judgment of the furnace man, as many furnaces and types of hanging require different treatment and amount of checking. When more than one check has been applied to the furnace the top and tuyères are watched carefully to determine the instant the furnace slips, as it is essential to turn the blast on as quickly as possible to prevent jamming and packing of the material inside the furnace, thus making it stiff. This is especially essential when the snort valve has been opened.

When hanging and sticking prove so stubborn that even opening the snort valve will not cause the furnace to slip, about all that can be done is to resort to cold air at low volume. This usually causes the furnace to slip. Another option is to take the blast off suddenly by first notifying the engine-room force to stand by the engines, then signaling a check, and immediately throwing the snort valve open. This treatment, repeatedly applied at quick intervals has been known to cause the furnace to slip but is hard on blowing equipment.

A practice which is sometimes resorted to is to let the furnace stand with the gas drafted back. This method is seldom successful, the net result being apparently only to make the furnace tighter, and it is usually resorted to only when all other methods have failed. A better final method is to speed the engines up, push the blast pressure up to the limit, and thus break up the hanging column. This method and the use of cold air have been most successful, but the use of high blast pressure is sometimes prevented by weak stove shells, blast mains, blowing equipment, or furnace.

PROCEDURE WHEN CINDER IS AT THE TUYÈRES.

Sometimes neither cold air nor a check can be applied to slip the furnace. This happens when the tuyères are "sloppy"; that is, banked up with a boiling mass of liquid cinder which is kept from running back into the blowpipes only by the full pressure of the blast. In this event, the furnace man is in a dilemma. Sometimes he has to let the furnace go, perhaps for two hours, until it slips itself, when it will usually fill the tuyères and blowpipes. Sometimes he will try to "dry the tuyères" by using all the heat available and flushing continually. Although this makes the furnace stick tighter than ever for the time being, the high heat may cause the cinder to disappear from in front of the tuyères, when the furnace may be checked. Sometimes he takes a chance, checks the furnace, and permits the cinder to run back in the pipes, trusting that they will not be cut or burned. The peculiar feature about this condition is that although the cinder notch may be open, and the furnace apparently drained, cinder may lie persistently at the tuyères.

VARYING THE BURDEN.

Increasing the amount of limestone charged is usually beneficial in old furnaces, both by increasing the slag volume and in increasing the concentration of CO_2 in the top part of the furnace. In one furnace with a badly eroded lining, an increase of 600 pounds of limestone, about 10 per cent, virtually eliminated an incessant series of top slips of long standing.

An extra round of coke and a slight lightening of the burden for 8 to 12 hours is usually efficacious in stopping a proneness to work stiff; that is, slipping about 8 to 10 feet at regular hourly intervals.

Stubborn hanging and frequent checking and slipping can usually be most effectually stopped by a large coke blank, or "dose." At a few plants where the furnace is mechanically strong, has a tight top, and ample stove equipment, there is sometimes a disposition to make the furnace relieve itself, without the use of coke blanks, a lightening of the burden being the only aid given. This is possible only where equipment is in first-class condition, plenty of stove heat is available, and there is little disposition toward being critical as to sulphur and silicon in the iron.

USE OF STEAM IN BLAST.

A few plants have used steam, admitted with the blast, to loosen hanging furnaces, but there does not seem to be much enthusiasm for this method.

EXAMPLES OF ACCIDENTS CAUSED BY SLIPS.

Descriptions of slips and the accidents caused by them follow:

A foreman and helper were on top of a furnace inspecting the condition of the bell when the furnace slipped violently. Flames from the explosion doors and from inside the gas seal burned both men so severely that they were disabled for more than six months.

Two top fillers were caught in a slip which blew the bell out of the furnace and enveloped the top of the furnace in a cloud of burning gas and dust. The time lost was 7 and 13 weeks, respectively.

Two bottom fillers were sent up to oil the sheave wheels on top, and were caught in a slip, the flying dust settling on their faces and arms. Lost time, 10 days each.

Two top fillers and one rodding man were burned by flying dust and flame ejected by furnace slips, resulting in a loss of time of 21, 30, and 37 days, respectively.

A dust man was dumping a dust catcher, when the furnace slipped and gas blew out from the bottom of the catcher, through the open bell, burning his hands and face and resulting in 2 days' disability.

A dust man was standing alongside a car, playing water on the dust which had just been dumped, when the furnace slipped. To escape the falling material he jumped off the wall into the dust car beneath the catcher and burned his feet, ankles, and legs severely with the hot dust. He suffered 26 days' disability.

A dust man was cleaning up dust beneath a downleg which had just been emptied when the furnace slipped. The pressure blew the bell on the leg open, and the dust and flame burned his scalp and neck, resulting in 35 days' disability.

A dust man was beneath a dust catcher stopping up the doors on a hopper car, when the furnace slipped, opening the dust bell and sending a shower of burning dust and flame over his hands, wrist, face, neck, and ears. He lost 41 days' time.

A track cleaner was underneath a dust catcher, the base of which was entirely inclosed except at the track, cleaning the track immediately after the car had been withdrawn. The furnace slipped and knocked the dust bell off the lever, and the man was killed.

A stove tender was adjusting a burner nose shield when a slip caused the burning gas to puff back about the door frame and burner, burning his head and neck, he lost 8 days' time. Another stove tender opened a door to see how the gas was burning; just then a slip occurred and gas ejected from the door burned his face, ears, scalp, neck, wrist, and hand. Another tender was struck on the head by a small piece of falling coke, causing 4 days' lost time.

In three separate instances when cleaners were cleaning the wells of stoves in which the gas had been left burning, gas blew out of the doors on account of the furnaces slipping and burned the men's hair, face, and hands, involving losses of 29 to 48 days.

In four instances boiler-house firemen were tending fires or blowing tubes when gas was driven from the doors by the furnace slipping and caused burns resulting in disability of 2 to 63 days.

In six instances when men were working about the furnace front changing tuyères, blowpipes, or caps, the furnace slipped, blowing gas, coke, and cinder from the opening over the men, causing disabilities of 16, 17, 25, 50, 97, and 210 days, respectively.

Several men were about the front of a tapping hole. The furnace was badly scaffolded and the men were putting an oil burner in to burn out a cavity in the hearth, when the furnace slipped, throwing a shower of cinder, dust, and flame over them.

A cinder snapper received burns of the abdomen in a similar manner, causing a loss of time of 21 days.

Two laborers, engaged in unloading coke from a flat-bottom high-side coke car were fatally burned by being caught in a shower of coke thrown out of an adjacent furnace by a heavy slip.

A boiler cleaner was standing on the combustion chamber of a boiler sealing tubes, when the furnace slipped. There being no roof over the boiler house, some red hot coke hit him, setting his jacket afire. He thereupon jumped off the boiler to the ground, and in consequence of burns and bruises was severely injured.

A slip threw the bell and hopper out at a furnace, killed one man and injured several seriously, the falling ore, coke, and limestone being blown a great distance.

A slip caused the dust catcher to burst, and the escaping gas formed a sheet of flame which fatally burned five men and injured four others.

Several accidents have occurred in former years, none recently, in which a scaffolded furnace slipped and the impact of falling material on the sloping walls burst the bosh. (See p. 22.)

The most numerous type of accidents is comprised in men being struck by falling material thrown from explosion doors and bleeders. Six instances are known in which riggers and millwrights working on elevations or platforms difficult of escape, were struck by material thrown out by a slip, and were injured, losing 2 to 4 days' time. In another instance a man jumped from a hot-blast valve head to escape from a slip, fracturing his leg, and in another case a man leaped from the top of a stove to escape a slip and was killed.

Other accidents from the same cause are cited below:

Coke thrown from the furnace struck a man's leg, burning and bruising it and causing a loss of 2 days' time.

A man was working on trestle and coke thrown from furnace by a slip cut his scalp, causing a loss of 15 days' time.

The furnace slipped and a man running for cover was struck on the neck by hot coke and bumped his head in crawling under a car, losing 17 days' time.

A piece of brick thrown out in a slip fractured a yard cleaner's rib, causing a loss of 30 days.

A laborer was digging a ditch near the furnace, when it slipped. A lump of ore fell from the top and broke his arm, causing a loss of 42 days.

A laborer stepped outside the cast house as the furnace slipped and instead of jumping back, attempted to run to a near-by office. He tripped and hot coke burned his legs and body; loss of time, 42 days.

The furnace slipped, throwing hot coke on man cleaning up cinder on the tracks near the furnace. The man was burned about the scalp, neck, ears, and arms, and lost 68 days.

In four different instances men who were standing near a furnace watching the top to see the furnace slip were injured. In some of these it was necessary for the men to note the starting of a slip, so that they could signal for the blast to be turned on; in others the injured men were not obliged to stay in a hazardous position, but did so out of curiosity. The lost time was 15, 28, 29, and 105 days, respectively.

Eight men were severely burned and bruised by material expelled from a furnace by a slip falling upon them while in an adjoining cast house, from which the roof had been removed.

Two men engaged in cleaning the roof of a cast house were caught by a slip and burned by the hot dust, causing disabilities of 27 and 30 days, respectively.

A slip broke the shell of the furnace and ejected a shower of red-hot coke and ore over seven laborers, causing disabilities of 8 to 245 days.

CHARACTER OF INJURIES.

By far the most numerous class of employees suffering accident from slips is comprised in the laborers about the plant. These men, for the most part unacquainted with the obscure indications of an impending slip, and engaged in tasks singly and in gangs about the plant, in the yard, on the trestle, and at the cast house, make up the bulk of men injured. The possible severity of various injuries and the amount of time lost is indicated in the following tabulation.

Thirteen typical accidents at blast-furnace plants, showing time lost.

Kind of accident.	Period of disability, days.
1. Lacerations of forehead, burns of neck, face, and hands	3
2. Burns of neck	3
3. Burns and cuts on face and neck	5
4. Burns of face, laceration of scalp	7
5. Burns of hand and forearm	12
6. Burns of body	14
7. Burns of face and neck	16
8. Burns and cuts on face and hands	18
9. Laceration of scalp	25
10. Contused and burned shoulder	25
11. Burns and cuts on arms and hand	27
12. Burns on back of hands, also bruises	52
13. Burns on limbs, also cuts and bruises	195

SEVERITY OF SLIPS IN PRESENT TYPES OF FURNACES.

Frequent mention of tops being blown off furnaces, shells bursting, and boshes collapsing as the result of slips may be found in the trade journals of this country up to 1909, and even at the present time in foreign trade journals. These disasters often resulted in considerable loss of life, as many as 3 to 12 men being killed or injured. The lack of mention of such accidents in American journals during the past six years is partly due to the reluctance of those concerned to discuss the trouble or to invite publicity, but more especially to elimination of such accidents. With the exception of a number of furnaces having weak shells and disadvantageous lines and distribution, it may be said that most blast-furnace plants are free from the possibility of such accidents as occur from failures of tops, boshes, or shells.

Terrific slips still take place, however, and the effect of these is confined to the interior of the furnace, if it has a tight top, or shows up at the gas burners and bleeders. The net result of such slips is largely confined to property damage and decrease in tonnage. For instance, at a furnace with a tight top a heavy slip forced open the bell on a dust catcher, and the gas rushed out through a dust chute into the hopper of a railroad car with such velocity that the car rolled over and over until it brought up against a boiler-house wall.

A much-dreaded type of slip is one where the falling column of material shears off a number of bosh plates and admits large quantities of water into the hearth and crucible. When this occurs, unless the leak is discovered immediately, it may require one to three days' hard work to get the furnace in condition again.

Occasionally, although no water reaches the hearth, a pasty mass of unsmelted material fills the hearth and part of the bosh. This also requires a long period of hard work before the mass can be melted up and the tuyères cleaned so that smelting and driving begin again. When the material dropped down into the hearth is loose the cinder-notch cooler is sometimes removed and the material blown out. One case has been seen where many tons of cold material, coke, raw limestone, and unreduced ore was blown out.

Occasionally the iron and slag is just barely fluid and when the tapping hole is opened the sluggish iron runs and chills, builds up, and forms a small crater, out of the top of which iron is flowing and chilling, thus continually building it higher. A case was seen where this crater or accumulation of frozen iron and slag with a liquid core had built up almost to the blowpipe level.

Another condition much dreaded is when the stock slips, falls into the hearth, and jams down into the bath of iron and slag, forcing it up above the level of the tuyères, where it sooner or later runs back into a blowpipe, burns it, compels the blast to be taken off, and then completely fills the other tuyères. Another variety of slip is illustrated by the following incident: A furnace was working stiff but not slipping markedly, and immediately after a cast was finished a large amount of black scaffolding material with much cinder slowly descended to the tuyères. Tuyère after tuyère filled until when only three were available and no cinder could be obtained at the monkey, braces were hurriedly placed against the stocks, the bridle rods were removed, and then the braces were pulled from a safe distance. A large amount of cinder ran over the cast-house floor, but after it was cleaned up three tuyères were easily gotten into blast again and the furnace worked into condition. At the present day, and at most plants, this and similar varieties of slips are the most dreaded.

EXAMPLES OF EXPEDIENTS USED TO CHECK SLIPPING.

Some idea of the work entailed by severe slips, a glimpse of the occasional difficulties of furnace work, and the expedients tried are shown in the following descriptions supplied to the author by blast-furnace superintendents:

The blast on both furnaces had been very regular and the furnace became very hot with a very basic cinder. The burden increased 2,400 pounds in two days on one furnace and 2,800 pounds in the other. The latter furnace became "lime cold," filled the tuyères in a slip, and "skulled up" the cinder notch with a limy iron-bearing cinder. After holding the furnace 10 hours the cinder notch was finally opened and the furnace was flushed, the cinder filling the ladles and running all over the yard. On Sunday the furnace filled its tuyères again from a slip, getting "cold on the lime" and "heavy burden" (too much limestone for flux and too much ore), and making no gas to speak of, a new stove showing only 800° F. By adding considerable extra coke, lightening the burden, and reducing the lime the furnace was gradually worked into good condition.

The other furnace, becoming very limy, was also getting cold and hanging continually. Finally the monkey was taken out to flush out the cold cinder. During the flush the furnace slipped, filling the tuyères and burning several blowpipes and stocks. As soon as four tuyères could be gotten the blast was turned on and attempt made to drill the cinder and iron notch. After several hours the blast had to be taken off, for neither tap hole nor iron notch could be gotten. Preparations were then made to cast through a tuyère about 2 a. m., and an oil burner was put in both the iron and the cinder notch and was set going, taking the blast from the other furnace. The water was then taken from the tuyère, after a runner had been made, and for 15 minutes iron and cinder flowed from the tuyère. The tuyère was then plugged with clay and brick at 7 p. m. in the evening, the iron notch was opened, the wind having been off in the meantime, and four ladles of iron obtained. The notch was plugged with sand, coke dust, and clay, but hardened again, requiring a blowpipe. It was finally opened at 5 p. m. the next morning, the burden lightened, the lime taken off, and several blanks of coke charged, and the furnace was gradually gotten into condition.

Another example is as follows:

The furnace had gotten cold. At 10 a, m. it was reported to have been hanging for two hours, and shortly after it slipped to the 40-foot line (this would be a 25-foot slip). It was filled, and made a slight slip a half hour later. At 4 p. m. it had been hanging two hours and slipped to the 45-foot line, blowing off the top and filling the blowpipes and tuyères. After the hopper and bell had been replaced the furnace was given 300,000 pounds of coke with limestone flux, with the result that the cinder showed up limy and filled the tuyères again. By this time the cinder notch and tap hole were tightly frozen, and a burner had to be put on both. The cinder notch was easily opened with the burner, and after several flushes a runner was made and a cast taken from the cinder notch. At the iron notch, however, the burner was in 6 feet, and no iron could be gotten, so a charge of dynamite was put in, well tamped with clay, and set off. This opened the hole to some extent, but after a while it froze up again. The burner was put on once more, and the notch finally opened, and the roof broke above the hole, making a good cast. By using extra coke and light blast the tuyères were gradually opened and the furnace was worked into condition.

These notes are typical of the work required when violent slipping occurs, and, with the exception of the blown-off top, represent the emergencies for which furnace men must be prepared, even nowadays. The oxygen burner has made the work much easier, but experience is still the requisite in determining the steps necessary to get a furnace into condition again after the tuyères and notches have frozen up.

The general rule is to get at least one tuyère, adjacent to the cinder notch, on blast, and more than two are not considered desirable at the start. In some cases the tuyères have had to be abandoned and the furnace started with a monkey inserted in the bosh.

PRECAUTIONS AND SAFEGUARDS.

Practice, design, and materials may be adjusted to a considerable extent to give a certain freedom from slips. In view of the many handicaps to attaining a minimum amount of slipping and the certainty that slips will occur, protection of the force from injuries by these causes is essential. The most vital step in eliminating slips and the injuries resulting therefrom is an improvement in design.

THE USE OF TIGHT TOPS.

The improvement that has resulted in the greatest diminution of accidents is the application of the tight top to furnace construction. Formerly a man virtually took his life in his hands when he ventured on top of a furnace, or on any elevated place, difficult of escape, in range of explosion doors. The tight top, with no explosion doors and with a positively closing bleeder, or bleeder which allows only fine dust to escape, has practically eliminated the chance of injury by flying material, flame, or hot dust.

On furnaces of modern construction, having heavily armored boshes, three-fourths inch double and triple riveted shells, dome, or strongly braced tops, and strongly constructed downcomers and dust catchers, explosion doors have been eliminated with impunity, and automatic bleeders have been replaced with bleeder valves that opened only by hand, high pressure from slips either forcing them against their seat or balancing any tendency to open. At furnaces of questionably strong construction, so that a slip may burst the shell unless prompt relief of pressure is afforded, explosion doors at the gas offtake can usually be eliminated with no danger of the shell bursting, provided that bleeder valves of ample capacity are installed. By eliminating the explosion doors and providing four bleeder valves instead of the usual single valve, adequate relief is afforded, and placing the valves at the top of bleeder pipes 20 to 40 feet above the furnace top lessens somewhat the amount of material ejected by the rush of gas when minor slipping occurs.

REINFORCING THE BOSH AND STRENGTHENING THE TOP.

Frequently more radical steps are necessary if the furnace shell or bosh construction seems too weak. Heavier and wider bosh bands, wide enough to fill the vertical distance between the bosh-plate boxes or openings, may be provided, and the old bosh bands placed inside the furnace stack in a series of reinforcing bands at 3 to 5 foot centers, a few bands of similar material being used to supply any deficiency. The top may require further strengthening by additional braces from the shell to the top ring, and also better provision against displacement of the hopper and hopper extension. Such modification enables the use of bleeders that afford relief of gas pressure, but not the escape of large pieces of limestone, coke, or ore.

EXPLOSION DOORS.

The presence of explosion doors on gas offtakes has not sufficed, regardless of their size and number, to prevent slips from throwing the bell and hopper out of the top, splitting the shell, and wrecking the base. Furthermore, the difficulty of maintaining tight seats owing to erosion by dusty gas or warping of the door or seat introduces the possibility of air sucking in during shutdowns or slips, thus causing so-called "top shots" occasionally.

The uselessness of explosion doors in preventing danger from explosions, the annoyance they cause by ejection of material on even trivial slipping, the cost of keeping the yard clean, and the danger of ejected material falling on men about the yard and furnace top is sufficient cause for their elimination. Some plants having four to six explosion doors have found it safe to eliminate half of them. The door is left in place and is fastened down on its seat by a so-called "safety" bolt, three-eighths or one-half inch in diameter, which supposedly will break in the event of a violent slip and permit the door to open. At one plant where this arrangement is used none of the bolts have ever broken. Another practice is to provide hinged explosion doors with heavy chains, which prevent the door from opening more than 12 to 18 inches. Sometimes a heavy structural beam is fastened over the top of an explosion door, so that a horizontal door can not lift more than a few inches when the furnace slips. Hinged explosion doors may be provided with a shield in front of them, which prevents material being thrown, sometimes hundreds of yards, over the furnace or adjacent property.

BLEEDER VALVES.

Bleeder valves are of two types—valves seated at the outlet of the bleeder pipe and valves contained within the pipe, the outlet being above the valve and a continuation of the pipe. Both types may be arranged to permit the passage of solid material carried in the gas or to prevent it from escaping with the gas. When possible such valves should be designed to permit only fine material to escape, thus eliminating the showering of heavy lumps on the roofs and yards, while affording relief from gas pressure. Interior valves permit gas to escape without the noise that characterizes the other type and are less liable to burning of the control levers for closing the valves against high pressure.

NECESSITY OF STRONG CONSTRUCTION.

If the use of explosion doors and bleeders that permit free escape of material is required, then safeguards should be provided. However, the fact that the jacket is considered so weak that the throttling of the release of gas pressure would present hazardous conditions, should justify serious consideration of strengthening the bosh, shell, top, and downcomers at the first opportunity. Such weak construction, if it exists, is inimical to the safety of employees and may cause serious losses in production and investment in the event of heavy slipping.

SIGNALS AND WARNINGS.

SIGNALS BETWEEN CAST HOUSE, BLOWING ROOM, AND SKIP OPERATOR'S STATION.

Unnecessary hanging of the furnace may often be avoided by the use of a telephone, speaking tube, or signal bell and light between the skip operator's station and the blower's office and the cast house, or by a system of regular half-hour card reports, showing the height of stock line and furnace condition, whether "hanging," "stiff," "regular," "out of reach," "slipped," "number of charges," etc. The simplest type of signal from the skip and bell operator to warn that the furnace is hanging is a blast from a special whistle, used only for this purpose. More in favor is a signal-light system, of which possibly the simplest type is a red light in the cast house, which in event of the furnace hanging is turned on by the skip operator. This is supplemented by a similar red light at the skip operator's station, which the foreman or blower lights from the cast house when he is checking or stopping the furnace. Sometimes this signal system is elaborated by having two lights, one white and one red; in the cast house red indicates "hanging" and white indicates "slipped," whereas at the skip operator's station red indicates a "stop" or "check," with notice to report the condition of the furnace, and white indicates "blast one" or "start charging." Occasionally a gong is used to supplement the light in order to attract attention. Instead of the foreman having to notify the skip operator that he is checking the furnace, a "buzzer" is sometimes relayed from the engine-room whistle circuit so that the skip operator is automatically notified in case of a check or stop. Two plants have installed an electrically lighted signboard in the cast house, at which the height of the stock line and the words "hanging" and "slipped" are illuminated as required by punching in a plug at a miniature panel in the stock house.

The use of an automatic recorder located in the blower's office, showing the movement of the skip or hoist, bells, rodding device, and the distributor, if any, is an invaluable means of noting any interruption to regularity of charging. A similar knowledge of interruption of filling is afforded by a sensitive top temperature or top pressure recorder. Frequently, if a daily analysis of furnace gas from samples representing an 8 or 12 hour period are made, water leakage may be indicated by the abnormally high hydrogen content.

In order to avoid mistakes in checking the furnace at the blowing room, the whistle should be as dissimilar in tone as possible from other whistles, and the furnace from which the signal is blown should have a light of different color, or its number, displayed in the engine room by the circuit operating the whistle. The use of electric power for operating the whistle is preferable to other means, as a light may be placed at the cast-house switch which is flashed at each contact. At one plant, to insure thorough understanding of signals, each engine is provided with a switch at the engine throttle, whereby the engineer is enabled to relay the blower's signal back to the cast house. One plant, in order to be sure that the electric-signal system is always in working order, has it so arranged that the telltale light on both the cast house and the blowing room are in circuit at all times, and lighted, except when the switch is thrown for the signals.

GENERAL METHODS OF WARNING WORKMEN.

There are three general methods of special warning to workmen about the yard, stoves, trestle, or cast house. Some plants provide a whistle at the skip operator's station, as already noted, or at the cast house. When placed in the cast house it is blown by the man checking the furnace, and is preferable to the other location, in that the blower can often tell when a furnace is about to slip when the skip operator will not have any intimation from the filling or rodding that such is the case. Also, if the blower gives the signal he can time it more closely to the actual danger point than can the skip man, who is in ignorance of when the furnace is to be checked.

Another method, used at but few plants, however, consists in having a number of red lamps at points commanding a wide range of yard or working places. In event of the furnace being due to slip, these lights are thrown into circuit. They really supplement the whistle at the blowing room. Inasmuch as this whistle is blown for a shutdown, for making a cast, or for changing a tuyère, it soon ceases to have any particular significance as regards indicating a slip. If it is supplemented by red lights about the cast house, men in hazardous positions can readily determine whether the furnace is about to slip by glancing at the lights when the signal at the blowing room is heard.

A similar safety provision is in force at four plants, where in event of a slip signs are displayed which read "Danger—furnace slip." These signs are covered when the furnace is running smoothly, and are displayed only in event of an impending slip.

When a signal is needed for men at the stoves, boiler house, yard, trestle, in cars or in ditches, on top of the furnace, or in any exposed place, a whistle is the simplest and most urgent warning. It should be loud and of distinctive sound and not smaller than 2½ inches in diameter. It should be sounded as a warning when the furnace is discovered hanging and repeated before the furnace is checked.

PROTECTION FROM FALLING AND FLYING MATERIAL.

The roof over the cast house should be of at least three-sixteenths inch steel plate, as roofs of corrugated iron are likely to corrode rapidly and permit material to fall upon men in the cast house. The walls of the cast house, if not continuous from roof to floor, should extend down far enough to prevent material from rebounding from stoves or adjacent furnaces and buildings into the cast house. The roof should be continuous about the whole diameter of the furnace. Boilers, hoisting machinery, cinder-crane operator's cabs, hoist houses, and offices and shanties in proximity to furnaces should be similarly provided with strong roofs of fireproof material.

The windows should be provided with strong wire-mesh screens. One plant has installed a series of canopies or coverings over walks and steps about the base of the furnace, where it is frequently necessary for men to walk to and from their work in range of flying material from slips. Where oil, steam, or air cylinders for operating the bell cylinders are in range of flying material, many plants have installed steel roofing over the cylinders to protect repair men from injury. Several plants have installed small shelter houses at the top of the furnaces, generally on the top platform adjacent to the bridge from the stoves to the furnace top, to provide shelter to men caught in a slip while working at the top of the furnace.

PRECAUTIONS TO PREVENT ACCIDENTS FROM SLIPS.

To prevent accidents from slips, aside from the hazards from falling material, which may be quite remote, or absent, or guarded against as described above, certain precautions are necessary in the work about the cast house, stoves, boilers, and top.

PRECAUTIONS IN GOING ON FURNACE TOP.

When it is necessary for men to go on top of a furnace they should report to the blower, regardless of whether routine or special work is to be undertaken. Before the blower permits the men to go he should take all possible precautions for their safety. If the furnace is hanging, men should not be permitted to go on top under any consideration or circumstances. It is best always to check the furnace and keep it on check until the men are off the top. If necessary, the blast should be taken off altogether.

PRECAUTIONS IN WORKING ABOUT THE CAST HOUSE.

If the furnace is known to be hanging, warning should be given by means of the whistle, signs, or lights, so that men about the yards will have time to get out of the way.

Men watching the tuyères when the furnace is hanging should not hold their eyes near the eyesights, as slag or coke may be thrown against the glass and break it; also, they should wear goggles or hold a hand glass between the sight and their eyes.

Before a tuyère, plate, or cinder-notch cooler is changed, or the last shift goes on the drill in opening the iron notch, or the crew begins work on the notch after a cast, or caps are to be removed from tuyère stocks, or the blowpipes dropped to clean them, the blower should be sure that the furnace has settled and that it is not hanging. While work of this character is being done the skip and bell operator must not dump the large bell, as hot slag, coke, or flame may be ejected through the open tuyères or plate openings, as if the furnace had slipped.

PRECAUTIONS IN WORKING ABOUT DUST CATCHERS AND STOVES.

When stove-well bottoms are being cleaned the gas burner should be shut off completely or sufficiently closed so that if the furnace slips gas will not flash out of the cleaning door.

Whenever a signal is given that the furnace is about to slip, the stove cleaners and hot-blast men should keep away from the cleaning doors, and they should never for any reason open the doors of stoves on gas until the furnace has slipped.

Dust men, stove cleaners, and stove tenders should keep from under gas legs, and away from stove burners, and under no circumstances dump a dust catcher when the furnace is hanging.

PRECAUTIONS IN THE BOILER ROOM.

Tube blowers and firemen should keep away from boiler-setting doors upon receiving warning that the furnace is hanging.

GENERAL PRECAUTIONS.

Foremen in charge of men working in places where they are exposed to slips should be on the lookout for any signs of a slip. When a warning is given, any men in cars or ditches or on elevated places should be immediately ordered to a safe place. Other men in places from which escape is more easy should be warned not to remain so far from shelter that it will be difficult to quickly get under cover.

No workman should be permitted about any place where he is in danger from slips unless he is actually employed there. If a man is assigned work in such places, his experience about furnaces should be ascertained, he should be warned of the dangers, and whenever the furnace is hanging the blower should particularly warn him.

One type of accident which occasionally occurs when the furnace slips heavily, is the ignition of oil in the cold-blast main by burning gas surging back from the furnace through the mixing valve. However, it is only through a poorly seated valve, or gross neglect in closing the valve, that such accidents can occur. **Before a furnace is checked, the mixer valve on the by-pass between the cold-blast and the hot-blast main must be closed.** Two minor accidents from fire in the cold-blast main caused by the furnace slipping and burning gas rushing back into the cold-blast main through an open mixing valve on the by-pass have been noted; in one case a cast-steel elbow in the main was cracked, and in the other three cold-blast valves on the stoves were put out of commission.

Inasmuch as it is always possible for a mixing valve to be imperfectly closed or to be forgotten and not closed at all when the furnace is checked, the use of automatically closing valves should be earnestly considered. Such valves are of various designs, either closing by weight of the valve itself when the blast is slackened, or being actuated by auxiliary air cylinders in which the piston opens or closes the valve in the cold-blast mains or mixer pipe, according as the blast is turned on or taken off the furnace. In using such auxiliary valves in the by-pass an indicator should be provided to show the positive working of the valve; also the valve should be inspected at intervals to see that it does not become warped or broken.

Watchfulness is necessary to insure positive and effectual working of automatic valves, for the use of such safety appliances as apparently eliminate the need of personal responsibility invites similar accidents by mechanical failure. The installation of such valves is of the greatest value if they are considered as supplementing the personal responsibility for the prevention of fires and explosions in the cold-blast main, but the stove tender should be held to his former obligation of closing the mixer valve when the furnace is checked, the millwright supplementing the stove-tender's duty in this regard by conscientious care of the nonreturn valve; otherwise such a valve is worse than useless.

CONCLUSION.

The furnace type, or features of construction, or methods of practice that will give absolute assurance of immunity from blast-furnace slips, hot-metal breakouts, or gas explosions have not yet been developed. So far as improvement and progress in these matters is concerned, the past 10 years shows an advance that is to the great credit of operating men and engineers, and if future improvements in this respect can be estimated from the advances of the past, it is apparent that accidents of the above types, although they will never be eliminated, will in frequency and seriousness become an even less serious factor than at present, as regards regular and safe operation.

In the attainment of this end, it is probable that, as in the past, improvements in design, construction, and practice evolved from operating experience and experiments will be of greater value than inventions and technical investigations, and that furnace men will look to small but gradual improvements in the distribution of stock within the furnace, the preparation of the materials of burden, the lines of the furnace, and advances in practice and construction rather than to studies of the chemical, physical, and thermal actions taking place in the furnace. Much advance has been made along practical lines, and the writer believes that it is in further development along the same practical lines by the men engaged in furnace practice that the greatest advance is to be expected in the minimizing of the types of accidents discussed in this report.

NEED OF FURTHER INVESTIGATION.

It is believed, however, that with the improvements effected in progress, and to be expected along these lines, most furnace men would welcome experimental work along certain lines, and that such work, if undertaken in cooperation with men or concerns operating furnace plants and auxiliary operations, would repay the time and effort expended. In suggesting the following studies, it should perhaps be mentioned that although they do in fact approach more nearly questions related to regularity and economy in operation than to safety in operation, it is virtually impossible to separate safety in operation from regularity in operation when the subject of blast-furnace slips or blast-furnace breakouts is being considered. Without qualification, safety is here a corollary of regularity in operation. The possibility of explosions is not related to operation in this close manner; here the personal element enters and the possibility of errors of judgment, omission, and commission play a larger part. But in the work suggested, the successful working out and application of the problems would result in better regularity of practice (which means reduction of the coke consumption per ton of iron produced, a decreased production of off-grade iron, less loss of iron per ton produced, and less interruption to the process from irregularities) quite as much as in the possibility of improvement in decreasing the probability of excessive slipping or dangerous breakouts, a possibility that has already been reduced within the bounds of reasonable if not exceptional security to men employed about blast-furnace plants.

The investigations the writer has in mind are:

(1) An investigation as to the relation of the physical properties of coke to its combustibility in the furnace and its solution loss in the upper part of the furnace, the preparation of specifications for blast-furnace coke to be based upon these properties rather than upon the chemical constituents (see page 223, Part III); (2) an investigation of the resistance of hearth and bosh fire brick to the attack of slag, iron, and the intrusion and disintegration by graphite and alkaline salts (see page 48, Part I); (3) an investigation of the free-flowing temperatures, the temperature of formation, and sulphur eliminating properties of slag (see page 235, Part III); (4) an investigation of sintered, briquetted, nodulized, and agglomerated flue dusts and fine ores, as to strength, state of oxidation and reducibility, porosity and glaze, density, and purity (see page 227, Part III).

GLOSSARY.

Air tub. The cylinder on blowing engine which pumps the blast or wind or air.

Bleeder. An escape valve for gas at top of furnace or along the gas line, to relieve excessive pressure or flow of gas.

Blowing on the monkey. Blowing air through the cinder notch at flushing.

Blowing on tapping hole. Blowing air through the hole at casting, to clean the hearth of iron and cinder.

Boil. Occurs when molten iron runs over a cold or damp spot or object in runner. The sudden generation of steam often causes an explosion, whereby molten iron is scattered about.

Bosh breakouts. Breakouts of the blast, gas, or coke through the bosh brickwork.

Bosh jacket. Reinforcing jacket to hold brickwork and platework of bosh in place. Employed mostly in eastern and southern furnaces.

Bot. A cast-iron or forged-steel plug, that mounted on a long steel rod, fits inside the monkey.

Botting. Thrusting bot into monkey to stop run of cinder during a flush, or when the furnace begins to blow on the monkey.

Bottom fillers. Man who fills barrow with ore, coke, or stone, weighs it and places it on cage, or elevator, to be hoisted to top of furnace.

Breakout. Escape of gas, coke, slag, or iron from the bosh, tuyère breast, or hearth of a blast furnace.

Bridle rod. A wrought-iron rod employed to hold the stock or "bootleg" in place against the blow pipe.

Cast house. The roofed (and sometimes inclosed) space in front of and about a blast furnace in which molten iron is run or cast.

Changing bronze. Changing tuyères, plates, monkey, etc.

Checking. Reducing the volume of the air blast on a blast furnace temporarily in order to make furnace "slip" or "move." Also in sense of first "check," second "check" at casting time to stop iron notch.

Chimney valve. Valve on hot blast stove open at base of chimney or at flue when stove is on gas; closed when stove is on blast.

Cinder breakout. The slag within the furnace comes through the brickwork and escapes. Caused by erosion, softening of brick by heat, or corrosion and penetration by slag.

Cinder notch. The hole, about 5 or 6 feet above iron notch, and 3 feet below tuyères, through which slag is flushed two or three times between casts.

Clay gun. *See* Mud gun.

Cooling plates. Hollow plates in furnace wall, through which water circulates to cool the wall.

"Cracks" of gas. Puffs or explosions of gas.

Distributor. Device for distributing charge when dumped into blast furnace.

Dog house. *See* Fore chamber.

Dose. Special charge used in blast furnace, designed to cure furnace troubles.

Downcomer. Gas main from furnace top to dust catcher.

Downleg. Leg or boot of a dust catcher.

Drilling up. Preliminary digging out of the clay in the tapping hole. This is done usually with hand, air, or electric drill.

Dust catcher. Primary chamber for settling dust out of blast-furnace gas.

DUST MAN. Man who dumps dust catcher or loads the dust.

DUTCH OVEN. *See* Fore chamber.

EXPLOSION DOOR. Door to relieve pressure in gas mains in event of a gas explosion; also valves at gas offtakes at furnace top to relieve pressure when the furnace makes a heavy slip.

EXPLOSIONS FROM MOLTEN IRON. Caused by molten iron escaping and coming in contact with water or wet material.

FORE CHAMBER. Auxiliary construction for gas-fired boilers, provides incandescent surface for lighting gas instantly when turned on after being shut off for any reason. Also called "Dutch oven" and "Dog house," and is of benefit further in enabling combustion of gas to be complete before boiler tubes are encountered.

FURNACE HOLDING IRON. Furnace gives much less than normal amount of iron at casting, although the charging may have been regular. The tap hole runs iron slowly, and amount of slag is somewhat scanty. Compare "furnace losing iron."

FURNACE LOSING IRON. Escape of iron from the hearth of a blast furnace into the foundation beneath, indicated by decreased quantity of iron at casting, and abnormally large appearance of slag at tapping hole.

GASSED. Overcome with gas.

GREEN HOLE. Tapping hole in which clay is not properly set, and through which the drill may break and let iron out unexpectedly.

HANGING. Sticking or wedging of part of the charge in a blast furnace.

HARD HOLE. Tapping hole difficult to open, usually one in which stopping clay is short and drill encounters cold skull of iron.

HEARTH BREAKOUT. Breakout of molten iron through the hearth wall or bottom.

HOLDING IRON. *See* Furnace holding iron.

HOT SPOT. A small portion of the furnace shell that is warmer than the rest. It indicates a thin lining at this place on account of erosion of brickwork by gas and stock.

LOSING IRON. *See* Furnace losing iron.

MANTLE PLATE. Plate that supports the mantle.

MESS. Masses of cinder, slag, iron, etc., about the furnace or yard, from burned tuyère or blowpipe, bad tapping hole, breakouts, burned ladle, or boil.

MONKEY. Small water-cooled bronze casting in cinder-notch cooler through which cinder runs from cinder notch when bot is withdrawn.

MONKEY BOSS. Man in charge of flushing furnace and of claying up monkey and coolers. Helps on tapping holes also, and at cast.

MUCKY HOLE. Tapping hole from which the iron is so pasty that it does not run freely.

MUD GUN. A steam cylinder operating a plunger inside steel tube 6 inches in diameter. Clay is fed into hopper tube as plunger is worked back and forth and is thus forced into tapping hole, at end of cast.

PLATE BOX. Box for cooling plate.

PRICKING BAR. Bar used in opening tapping hole.

RID UP. Cleaning up period after cast, when trough and runners are prepared for next cast.

RIDDING UP. Cleaning up after a cast.

SAND VALVE. An old type of arrangement in gas mains used to shut off flow of gas, superseded by water seal valves.

SHORT HOLE. Tapping hole with a short stopping, may break out unexpectedly when drilled into.

SKULL. Iron that has solidified on the interior of the furnace.

SLAG BREAKOUT. Breakout of slag or cinder through the brickwork between plates, tuyères, and top of hearth jacket.

SLIP. Sudden descent of a hanging or sticking charge in a blast furnace.

SNORT VALVE. A butterfly valve opening from cold-blast main to atmosphere. Allows casting at the furnace without shutting down blowing engines. Operated by large wheel or lever in cast house.

STOCK. Term applied to the mixture of ore, coke, and limestone charged into the furnace, or stored in bins at stock house. Also applies to connection between blowpipe and gooseneck attached to bustle pipe.

STOCK HANGER. Hanger for tuyère stock.

SUBJACKET. An auxiliary jacket about the lower part of the hearth.

TOP FILLER. Man who dumps charges into furnace top.

TOP KICK. *See* Top shot.

TOP SHOT. Explosion or puff of gas at furnace top.

TUBE BLOWER. Man who cleans tubes of boiler.

WELSHMAN. A heavy steel ring about 3 or 4 inches inside diameter, used in withdrawing a bar which is stuck or frozen in a skull of iron. The ring is placed on the bar, a wedge inserted, and the bar backed out by sledging on the wedge.

WILD GAS. Blast-furnace gas that does not burn steadily or properly.

WORKING ON THE WALLS. Eroding of furnace linings or channeling of gas through burden next to the lining.

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INDEX.

A. Page.

Air, cold, slipping caused by 252
  exclusion from gas mains at blowing in 132, 149
  in blast-furnace stoves, regulation of 101
  leakage as cause of slips 205
  trouble from 229
  precautions against 229
  mixture with gas, explosions from 111, 117, 118, 119, 122
  See also Blast.
Air compressors, explosions in, cause of 171, 172
  precautions against 172

B.

Bauxite brick for hearth walls 58
Bell, cleaning of, precautions in 104
  diameter of, in relation to stock line 209, 210
  distribution of charge on 211
  hanging of, precautions in 212, 242
  inspection of 241, 242
  position during blowing in 132, 134, 136–139, 142, 143, 147
    during blowing out 154, 155, 156, 157, 158, 164, 166
  precautions in dumping 264
Blast, distribution of, irregular, cause of 230
    detection of 230
    remedy for 230
  proper, need of 222
  erosion of brickwork by 24, 25
  taking off, during blowing in, danger from 131
    slipping caused by 252
  temperature, during blowing in 133, 144
    lowering of, to prevent slips 250, 251, 252
  volume of, during blowing in 132–134, 136, 137, 138, 139, 140, 141, 144, 148, 149
    during blowing out, reduction of 158, 159, 160, 161, 162
    excessive, as cause of slipping 228
    insufficient, as cause of slipping 227, 228
    variations in, difficulties caused by 228, 229
  See also Air.
Blast breakouts. *See* Bosh breakouts.

Page.

Blast furnace, care in operation of 35, 36
  columns, protection of 68
  construction of, proper 69
  draining of, in blowing out 163
  drying of, in blowing out 140, 142, 147
  early, construction of 5
  fast-driving, hearth conditions in, figure showing 73
  filling of, methods 138, 139
    precautions in 142
  foundation, figure showing 4, 7
  height, effect of, on slipping 213
  jacketed, figure showing 16
  lighting of, methods of 133, 134, 135, 136, 138, 143, 144, 146
    precautions in 147, 148
  lower part of, section of, figure showing 5
  single, blowing in, details of 131, 132, 139
    difficulties in 128
  slowing down, during slipping 252
Blast-furnace gas, composition of 96, 98
  danger of explosion from 95
  explosion limits of 99
  temperatures of 152, 155
  See also Gas.
Blast-furnace stoves, accidents at 101
  explosions at 100, 101, 129, 130
Blast mains. *See* Cold-blast mains.
Blast pressure, during blowing in 137, 143, 144
  during blowing out 157
  during hanging 186
  on brickwork, effect of 25
  reduced, slipping from 252
  regular, control 231
  relation of, to top slips 197
  variations in 231
    difficulties from 228, 229
    relation of, to charging rate 187
Bleeders, automatic, advantages of 259
  definition of 268
  position of, during blowing in 134, 136, 137, 138, 140, 142, 143, 146, 147
    during blowing out 154–157, 159, 160, 162, 164, 166
    during tuyère changing 113, 114
  proper design of 110, 111, 261
  types of 260, 261
"Blowing down" furnace, removal of scaffolds by 248

Page.

- Blowing in, gas explosions at 128
  - hazards in 147
  - methods of 131–146
  - precautions during 71, 72, 147–150
- "Blowing on tapping hole," definition of 268
  - objections to 85
- Blowing out, hazards in 151, 152, 153
  - methods of 153–163
- Blowpipes, breakouts at, dangers from 31
    - details of 34
    - prevention of 31, 32
  - construction of, precautions in 32
- Boiler, explosions, causes of 102, 103, 129, 130
    - prevention of 102
- Boiler room, safety precautions in 264
- Bosh bands, bursting of 20, 21
  - construction of 12
  - method of securing 12, 13
    - figure showing 12
  - reinforced, need of 260
- Bosh breakouts, causes of 3
  - definition of 268
  - frequency of 3, 23
  - hazard from 23
  - prevention of 35, 36
- Bosh jacket, definition of 268
  - figure showing 16
- Boshes, banded, breakouts in 20–22
    - construction of 6, 7
    - objections to 13
  - brickwork, construction of 9, 10, 213
    - figure showing 10, 11
  - combustion in 4
  - construction of, factors governing 69, 215
  - cooling plates in 10
    - leaky, detection of 233
  - details of 3, 4
  - erosion of walls of 7, 8
    - figure showing 8
  - height of
  - jacketed, accidents from 22
    - advantages of 13, 14
    - breakouts, causes of 20
    - brick-lined, durability of 18
    - construction of 13, 14, 15
    - figure showing 15, 19, 21
    - spray for 18
  - overcooling of, danger from 10
  - section of, figure showing 6, 7
  - types of 19, 20
    - development of 4, 5, 6
    - figures showing 19, 20
  - *See also* Roberts Farrell bosh.
- "Bottom hanging," causes of 188, 189
- Bottom slip, calculation of pressure from 201
  - description of 198, 199
  - indications of 198
- Breakout, definition of 2, 268
  - factors governing 266
  - occurrence of 2
  - *See also* Kinds of breakouts named.

Page.

- Brick, defective, faults of 220, 221
  - erosion of, as cause of hanging 197
  - *See also* Bauxite brick; Carbon brick; Magnesite brick; Silica brick.
- Bricklaying, of furnace lining 220, 221
- Brickwork, corrosion of, causes 24, 25
    - melting of, factors affecting 23, 24
    - protection of 26, 27, 28, 29
  - slagging of 24
  - *See also* Lining.
- Bronze, changing of, precautions during 112, 113
  - *See also* Cooling plates; Tuyère.
- Burden. *See* Charge.
- Bureau of Mines, blast-furnace investigations of 1, 2
- Burners, electric, for opening tapping holes 73, 74
  - oxygen 258
  - stove, design of 102
- Burrell, G. A., work of 98
- Butterfly valve, automatic, objections to 127

C.

- Carbon, formation, slips from 203, 204, 207
- Carbon brick, description of 58
  - for hearth wall, advantages 57
    - use of 57, 58
  - heat conductivity of 57
- Carbon deposition on ore, causes of 191, 226
- Carbon deposition, scaffolds from 193, 194
  - top hanging from 192
- Carbon dioxide, formation from carbon monoxide in explosive slips 207
  - formation of carbon monoxide from, in explosive slips 204, 207
- Carbon dust, as cause of furnace explosions 107
- Carbon monoxide, ignition temperature of 94
- Carbon monoxide with oxygen, explosion of, factors governing 96
- Cast house, definition of 268
  - protective devices at 263
  - safety arrangements in 36
  - working about, precautions in 36, 264
- Cast-steel bosh plates, advantages of 217
- Casting time, regularity of 72, 73
  - shutdowns at, procedure during 116
- Charge, furnace, composition of 135, 140, 141, 142, 143, 145, 146
  - fine material in, handling of 210, 211
  - materials in, proportion of 132, 133, 138, 139, 246
    - sequence of 238, 239, 240
  - proper distribution of 211, 243, 244

Page.
Charge, furnace, size of, proper 239
    weight of coke in, proper 238
    See also Coke; Limestone; Ore.
Charging, slackening rate of, as indication of hanging 185, 187
        variations in during hanging 187
    variable, purpose of 241
Check valves, types of 169, 170
Cinder at tuyères, treatment for 253, 258
Cinder breakouts, damage from 3
    definition of 268
    details of 34
    occurrence of 3
    prevention of 30, 31
Cinder for coke blanks, proportion of 244–246
    See also Slag.
Cinder notch, definition of 268
    breakouts at, precautions against 91
Cinder-notch cooler, changing, precautions in 264
    packing, need of care in 91, 92
Clark, H. H., cited 179, 180
Cleaning door, manipulation during blowing in 145
Coke, charging of, precautions in 237
        sequence of 238, 239, 240, 241
    friable, objections to 204
Coke for charge, amount of 133, 135, 143, 145, 146
    composition of, effect of 223
    density of, effect of 224
        high-ash, objections to 223
        proper physical condition of 223, 224
    proper size and weight of 224, 233
Coke blanks, composition of 244–247
    cost of 244
    frequency of charging 244, 247
    use of 244, 248, 253
Coke breakouts. *See* Bosh breakouts.
Cold-blast main, explosions in 167
        precautions against 169
Combustion chamber, boiler, gas explosions in 102
        prevention of 102
Cooler, cinder-notch, cutting of, prevention of 93
        precautions in changing 92
    See also Cinder-notch cooler; Tuyère cooler.
Cooler system, cinder-notch, clamps for 93
        figure showing 98
Cooling pipe, explosion in, repairs for 78
Cooling plates, bosh, arrangement 10
        protection by 10
    beneath tuyère coolers, need of 27, 28
    details of 6, 7
    dimensions of 6, 7
    flushing of 35, 69
    leaks in, detection of 233

Page.
Cooling plates, cast-iron, advantages of 8
        construction of 8
        figure showing 9
    definition of 268
    changing, shutdowns for explosions during 111
        methods for 112, 113
        precautions in 264
    leaky, detection of 233
    proper placing of 220
    tapping hole, explosion of, cause of 82
        objection to 80
    tuyère breast, flushing of 69
        types 25–28
        figure showing 26, 27
    See also, Bosh; Cinder notch; Tapping hole.
Crabtree, Frederick, acknowledgments to 1

D.

Distributor, definition of 268
    purpose of 210
    types of 210, 212
    See also Kileen distributor.
"Dose," definition of 268
    See also Coke blank.
Dougherty, J. W., cited 115
Downcomer, definition of 268
    drafting gas from, hazard in 153
    explosions in, causes of 105, 107
        occurrence of 103
        prevention of 106
Dressler, H., on gas explosion 118
Drills for tapping hole 89, 90
Dust catcher, definition of 268
    drafting gas from, hazard in 153
    explosions at, occurrence of 103
    valve at outlet of 107
    working about, precautions in 264
Dust catcher system, precaution at, during shutdowns 150
Dynamite, for blasting salamanders 175
Dyblie-Larimer valve, description of 171
    operation of 171

E.

Electric burners for hard tapping holes 73, 74
Engine, air leaks at, difficulties from 229
    speed during blowing in 131, 133, 134, 137, 141, 146
        during blowing out 160, 166
        control of 165
Explosion door, definition of 269
    during blowing in, position of 137, 138, 140, 143, 145, 146, 162, 166
    objection to 111, 114, 259, 260
Explosion wave, velocity of 97
Explosions at tapping hole 82
    at top 208
    from hearth breakouts 40, 51, 52, 53, 54, 56
    See also Gas explosions; Iron, molten, explosions from.

Page.
Explosive slips, causes of 204–208
Explosives, firing of, methods for 180
  for blasting salamanders 177
  for removing scaffolds, use of 249, 250
  handling of, precautions in 182
  storage of, precautions in 181
Eyesights, cinder breakouts at 81
    prevention of 81
  construction of 82
  figures showing 33, 34, 35, 36, 37
  types of 82

F.

Fatalities at jacketed-bosh furnaces 22
  at plate-and-band furnaces 21, 22
  from blasting salamanders 179
  from bosh bands bursting 21
  from gas explosions 153, 167, 168
  from hearth breakouts 51, 52, 53, 56, 57
  from molten-iron explosions 173
  from slips 254, 255
  from tapping-hole breakout 81, 82
  from tuyère-breast breakout 83
Fire brick, action of manganese on 46
  for hearths, composition of 60
  description of 59
  heat conductivity of 44
  manufacture of 59
  properties of 59
  resistance to slag 267
  types of 59
  See also Brick.
Flue dust, methods of treating 227
  utilization, need of investigating 267
Flux, charging, practice in 238, 239
  materials for 222
  See also Limestone.
Freyn, H. J., on gas pressure at blast furnaces 126
Furnace. *See* Blast furnace.
Furnace holding iron, definition of 269
  indications of 75
Furnace losing iron, definition of 49, 75, 269
  indications of 49
  precautions against 76
Furnace top, coke fire at, advantages of 150
  cooling of, in blowing out 162
  explosions at, causes of 105, 106, 107
    prevention of 106, 107
  repairs to, precautions in 108–110
  stationary double bell 211, 212
  temperature of, during blowing out 160
  tight, merits of 259
    results of slip in 257
  workmen on, precautions for 264

G. Page.

Gas and air mixtures, explosions of 111, 117, 118, 119, 122, 128
  slips by explosion of 205
Gas at furnace top, indications from 185, 186
  consumption of, at blast-furnaces 120
  during blowing in, regulation of 134–137, 139, 141–146, 148, 149
  during blowing out, regulation of 163, 165
    temperature of 155
  escape of, in bottom slips 198, 199
    in top slip 198
  in blast-furnace stoves, regulation of 101
  in cold-blast main, explosions from 167, 168, 169
  in downcomers, explosions from 153
  See also Blast furnace gas; "Blowing out" gas.
Gas breakouts. *See* Bosh breakouts.
Gas-cleaning plant, description of 120, 121
Gas explosions, causes of 94
  dangers from 94
  investigation of 95
  pressure from, calculation of 98
  processes in 96, 97
Gas mains, care of water seals in 117
    during blowing in, exclusion of air from 132
    filling with steam 143
    fires in 130
  explosions in 103, 104
  gages in 120
  pressure in 115
  reduced pressure in, dangers from 117, 120
    prevention of 119
  repairs to, precautions during 108, 109, 110, 114
Gas offtake, proper design of 110, 111
  proper placing of 221
  size of 221
Gas pressure, low, procedure for 125, 126
  maintenance of, need for 148, 149
  regulation of 127
Gas seal, precautions in entering 104
Gas-seal doors, hazards in opening 103
    precautions in 104
Gas-seal explosions, prevention of 104
Gas supply, insufficient, danger from 122, 125
Gauger, A. W., work of 98
Glossary 268–270
Graphite, action of, on brickwork 39, 48
  on hearth walls, analyses of 39
  separation of, from iron 48
Gravel in coke blank, proportion of 244–246
"Green hole," definition of 269
  dangers from 83

H. Page.

Hanging, causes of 209, 226
 definition of 269
 furnace conditions accompanying 187, 188, 214
 indications of 184–186
 See also Bottom hanging; Top hanging.
Hearth, cooling pipes for 66
  figure showing 66
 figure showing 4
 of fast-driving furnace, figure showing 73
 rising of iron in, danger from 74
 section of, figure showing 6, 7
Hearth bottom, blocks in, figures showing 62, 63, 64
  sizes of 62, 63
 concave, former use of 60
  figure showing 60
 construction of 61–63, 69
 derangement of, precautions for 76
 destruction of, causes of 46, 47, 48
  rate of 48
 hot, blowing in furnace with 147
 keyed, use of 61
 reinforcement of, subjackets for 65
 repairing of, figure showing 68
Hearth breakouts, danger from 2
 definition of 269
 details of 51–57
 frequency of 36, 37
Hearth ditch, care of 70
 level of water in 70
Hearth jacket, breakouts through 40
 conical, defect and merits of 61
 cooling of, methods for 43, 44, 45
 escape of iron at joints of, prevention of 45
 extension below bottom 64
  water cooling of 65
 proper construction of 69
 rolled steel-plate 42
 surface cooled, disadvantages of 44
 types of 42, 43
Hearth walls, construction of 37, 41, 69
 cracking of 39, 40
 destruction of 37–39
  figure showing 38
 graphitic coating on, analyses of 39
 preservation of 39
 proper thickness of 41
Hot-blast main, placing of 222
 valve for, description of 170
Hot spot, definition of 269

I.

Inwall lining, destruction of 217, 218
 preservation of 218–220
Iron in coke blank, proportion of 245, 246
 in furnace charge, proportion of 145
 Page.
Iron, molten, action of water on 172, 173
  explosions from 2, 21, 22, 51–54, 56, 57, 172, 173
   prevention of 174
  in hearth, rising of, dangers from 74, 75
  in tapping hole, chilling of 77
Iron notch, care of 86
 opening of, precautions in 264
 ore in charge, analyses of 144
 water cooling for, danger of 84
 weakening of 78
 workmen near, precautions for 86

J.

Jacketed bosh. *See* Bosh, jacketed.
Jackets, bosh, destruction of 15–17
 bursting of, accidents from 22
 hot spots on, accident from 22
 repair of 77, 78
 rolled-steel, breakout through 52, 54, 55, 56
 See also Hearth jacket.

K.

Kileen distributor, advantages of 211

L.

Lake Superior ores, smelting of, practice in 209–253
 type of bosh for smelting 6, 7
Laudig, O. O., experiments of 194
Lights, colored, as signals 261, 262
 open, during blowing in, dangers from 147
Lime in slag, excess, objections to 188, 189
Limestone for flux, charging of, sequence of 239–241
 charging of, sequence of 239–241
  decreased charges in blowing out 157, 158
  increased charges for preventing slips 251, 253
   proportion of 133, 135, 143, 145, 146
 silica content of 222
 size of 222
 use of, before blowing out 163
Limestone in coke blank, use of 247
 See also Flux.
Lining at stock line, disintegration of 216
 furnace, defects in 233
  drying of, time required for 221
  failure of 220, 221
  life of, factors 212
  lines of 213
  preservation of 215
  water-cooled, advantages of 220
 inwall, destruction of 217, 218, 219
  proper thickness of 218, 219
 See also Bosh, lining of.
Lyon, D. A., acknowledgments to 1

M. Page.

Manganese, action on fire brick 46
Magnesite brick, use of 58
Metz, N., on carbon deposition in furnaces 193
Mixer valve, automatic, merits of 265
Mud gun, definition of 269
precautions in handling 87, 88

N.

Neelend bucket, advantages of 212

O.

Ore, iron, charging of, sequence of 238–241
coarse, objections to 225
concentrated, smelting of 227
fine, objections to 204
loading of 236, 237
proper size of 225, 226
weighing of, precautions in 236, 237
manganese, in coke blank, proportion of 245, 246
siliceous, in coke blank, proportion of 245
See also Iron ore.
Oxygen, use in opening tapping holes 73
Oxygen burner. *See* Burner.

P.

Packing for hearth wall, materials for 42
value of 41, 42
Potassium cyanide in hanging furnace, formation of 195
Pressure, from slipping, calculation of 199–201
methods of relieving 202, 203
results of 199
See also Blast pressure; Gas pressure.
Purifying of gas, plant for 121
purpose of 122

R.

Receiving hopper, precautions in entering 104
Roberts Farrell bosh, construction of 19, 20
figure showing 21
Rodding devices as stock line indicator 243
Rotary washer, accidents at 118
use of, danger from 117

S.

Safety devices at blast furnaces 123, 124, 125
at cast house 263
automatic, objections to 127

Page

Salamanders, blasting, accidents from 175
explosives used in 175, 177
hazards in 175, 176, 179
holes for, cooling of 178, 179
drilling of 176, 177, 178
firing of 178
precautions in 176, 179
deep, danger from 50, 51
expansion of, difficulties from 67
formation of 45, 46, 47, 50, 51
hot, scaffolding over 146, 147
removal of, discussion of 66, 67
Sand in coke blank, objections to 247
proportion of 245, 246
Sand valve, definition of 269
Scabs, formation of, cause of 189, 190, 210
Scaffold in furnace, falling of, accident from 22
cause of 190, 193, 204
material from, analysis of 195
removal of 245, 248–250
Scaffolds, wood, for starting furnace 133, 134, 135, 136, 140, 142, 143, 145
Schöneweg, Herman, on removing furnace scaffolds 250
Schutte-McHugh valve for hot-blast main 170
Scrap iron in coke blank, proportion of 244–246
sandy, use of 247
Shots, blasting, firing of 180
precautions in 181
Shutdowns, dangers at 122, 123
explosions at 108
prevention of 111–116
precautions during 108, 109, 110, 122, 123
procedure in 150, 151
Signals, blast furnace, code of 123, 124
devices for 261, 262
See also Lights; **Whistle.**
Silica brick, use of 58
Simmersbach, Oscar, on reduction of carbon dioxide 204
Skappel, H., on explosive slips 207
Skull, as protection at casting 87
formation of 18
Slag, action on hearth walls of 38, 39
basic, relation of, to furnace temperature 234
formation of 189, 214
zone of 214
in charge, amount of 133, 135, 143, 145, 146
limy, action of, on tapping hole 79
analyses of 71, 72, 144
objections to use of 235
wrong composition of, as cause of slips 235
wrong volume of, as cause of slips 235, 251
See also Cinder.
Slag breakouts. *See* Cinder breakouts.

Page.

- Slips, accidents from 254–256
  - causes of 190, 209
  - damage from 257
  - dangers in 183
  - definition of 184, 270
  - effect of 188
  - furnace conditions accompanying 187, 188
  - factors governing 213, 266
  - occurrence of 184
  - prevention of 213
    - methods for 183, 250, 251, 257, 258
  - procedure during 251, 252, 258
  - pressure from, relief of 202, 203
  - types of 184, 197, 257
  - *See also* Bottom slips; Top slips.
- Snort valve, definition of 270
  - during blowing in, position of 133, 134, 136, 137, 138, 139, 141, 146
- Splasher plate, as protection at tapping hole 87
  - placing of, precautions in 88
  - shield for 89
    - figure showing 89
- Sprays, use at blowing out 155, 157, 159, 160
- Stead, A., on effect of manganese on fire brick 46
- Steam jets for lining protection 219
  - from cooling water, as cause of tuyère explosions 174
  - from hearth bottom, significance of 76, 77
  - in blast, use of, to check slips 253
  - in gas mains, use of 114
  - use of, at blowing in 129, 135, 136, 137, 138, 139, 143, 150
    - at blowing out 158, 159, 161, 167
- Steam main, repair of, dangers in 105
    - procedure in 105, 106
- Stock, definition of 270
- Stock line, height of 242
    - variation of 211
  - indicator for, value of 243
  - lining at, disintegration of 216
    - preservation of 217
  - relation of, to bell diameter 209, 210
- Stove cleaners, precautions for 102
- Stove tenders, duties of 124
  - precautions for 101, 102
- Stove valves, leaks in, detection of 233
- Stoves. *See* Blast furnace stoves.
- Subjackets, definition of 270
  - for hearth bottoms, types of 65

T.

- Tapping bar, device for holding, figure showing 88
  - precautions in handling 88, 89
- Tapping hole, blowing on, objections to 85
  - breakout above, protection for 77

Page.

- Tapping hole, breakout at, details of 80–82
    - injuries from 81
    - procedure in 77
  - breakouts below, causes of 45–50
    - indication of 75
    - precautions for 76
  - cleaning of 84
  - depth of, safe 90
      - figure showing 91
  - erosion of, precautions against 79, 80
  - explosions at 82
  - opening of 73, 74
    - precautions in 88, 89
  - protection of 68, 69
  - section of, figure showing 90
  - stopping of, difficulties in 83, 86
    - method of 79, 84
      - figure showing 81
    - precautions in 86, 87
  - trough about, advantage of 69
- Telephones at blast-furnaces, value of 123, 262
- Temperature at furnace top in blowing out 154, 156, 157, 158, 159
    - control of 160, 163, 164, 165
    - increased, as sign of hanging 186
  - high, as cause of slips 234
  - lowering of, effect of, on slag 189
    - prevention of slips by 250–252
  - relation of, to carbon deposition 191, 192, 193
  - variations in, as cause of slips 235
- Theisen washer, duties of men at 125, 126
  - signals for 123, 124
- Top hanging, causes of 190, 191, 194, 195, 196
- Top shot, definition of 270
  - *See* Furnace top, explosions at.
- Top slips, description of 197, 198
- Tuyère, auxiliary, description of 248, 249
    - objections to 249
    - use of 248, 249
  - care of, at blowing in 135
  - changing of, dangers in 111–114
    - merits of 232, 233
    - methods for 112, 113, 115
    - precautions in 264
  - cinder at, procedure during 253
  - diameter of 231
  - explosions at, causes of 174
    - prevention of 175
  - gas pressure at, relief of 115
  - long, disadvantage of 115
    - merits of 35, 232
  - temperature at 23, 24
- Tuyère breast, construction of 23, 35, 69
  - cooling of 31
    - reinforcement of 28, 29
    - unreinforced, dangers from 28, 29
- Tuyère-breast breakouts, details of 33–35
  - methods of preventing 35, 36
- Tuyère-breast wall, construction of, figure showing 26

Page.

Tuyère coolers, packing of, methods of 31
need of 30
Tuyère stock, figure showing 30
valves in, value of 171

U.

Uhling, E. A., cited 187

V.

Valve, for dust-catcher outlet 107
value of 107
automatic, inspection of 265
merits of 265
objections to 127
for gas-cleaning plant 121
See also valves named.
Van Flotten, W., on cause of top hanging 191
on explosive slips 203–208

W.

Washer. *See* Theisen washer.
Water about tapping pole, danger from 84
action of, on molten iron 172, 173
Water, addition of, to gun charge, dangers in 83
for cooling bosh, need of 10
for cooling inwall lining 218, 219
for cooling jacket, need of 17, 18
supplying of 17, 18
for cooling plates, amount of 11, 12
use in blowing out 153, 154, 156, 163
amount of 159, 160–161
danger from 152–154
methods of prevention 164
purpose of 151, 156, 157, 162, 164
regulation of 160, 164, 165
feeding of 11
Water cooling, excessive, as cause of hanging 197
Water main, repair of, dangers in 105
procedure in 105, 106
Water seal valves during blowing in, position of 134, 136, 143, 146
Water seals in gas mains, care of 117
during blowing out, position of 156
Whistles as safety signals 123
Workmen, injuries to, details of 254, 256
See also Fatalities.
methods of warning 262, 263

Bulletin 131

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, Secretary
BUREAU OF MINES
VAN. H. MANNING, Director

APPROVED ELECTRIC LAMPS FOR MINERS

BY

H. H. CLARK
AND
L. C. ILSLEY

WASHINGTON
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CONTENTS.

Page.

Introduction ... 7
  Scope of report ... 8
Acknowledgments ... 9
History of the investigation ... 9
  Importance of developing an electric lamp for miners ... 10
  Essential qualifications of electric mine lamps ... 10
  Qualities investigated and test procedure ... 11
    Liability to ignite gas ... 11
    Tendency to unexpected extinction ... 12
    Liability to spill electrolyte ... 13
    Light-producing capacity ... 13
      Candlepower and light flux ... 13
      Time of burning ... 15
    Bulb characteristics ... 15
    Life of batteries ... 16
    Life of lamp cords ... 16
    Mechanical and electrical characteristics ... 17
  Deficiencies remedied ... 18
    Safety device ... 18
    Angle of light stream ... 18
    Distribution of light ... 18
    Bulbs ... 19
    Cord and accessories ... 19
      Cord-test procedure ... 20
      Description of cord-testing device ... 21
      Electrical connections ... 21
      Results of cord test ... 22
Lamps approved and description of tests ... 22
  Edison lamp, approval No. 10 ... 22
    Gas-ignition tests ... 22
    Tendency to unexpected extinction ... 23
    Spilling of electrolyte ... 24
    Light-producing capacity ... 24
      Amount of light ... 24
      Distribution of light ... 24
      Time of burning ... 24
    Bulb characteristics ... 24
    Life of batteries ... 25
    Life of cords ... 25
    Mechanical and electrical characteristics ... 26
  Manlite lamp approval withdrawn ... 27
  Concordia hand lamp, approval No. 12 ... 27
    Gas-ignition tests ... 27
    Tendency to unexpected extinction ... 27

Lamps approved and description of tests—Continued.

Concordia hand lamp, approval No. 12—Continued.

Page.
Spilling of electrolyte 28
Light-producing capacity 28
Amount of light 28
Distribution of light 28
Time of burning 28
Bulb characteristics 28
Life of batteries 29
Mechanical and electrical characteristics 29
General Electric lamp, approvals Nos. 13 and 13A 29
Gas-ignition tests 30
Tendency to unexpected extinction 31
Spilling of electrolyte 31
Light-producing capacity 31
Amount of light 31
Distribution of light 32
Time of burning 32
Bulb characteristics 32
Life of batteries 32
Life of cords 33
Mechanical and electrical characteristics 33
Wico lamp, approval No. 14 33
Gas-ignition tests 34
Tendency to unexpected extinction 34
Spilling of electrolyte 34
Light-producing capacity 35
Amount of light 35
Distribution of light 35
Time of burning 35
Bulb characteristics 35
Life of batteries 36
Life of cords 36
Mechanical and electrical characteristics 36
Concordia cap lamp, approval No. 15 37
Gas-ignition tests 37
Tendency to unexpected extinction 38
Spilling of electrolyte 38
Light-producing capacity 38
Amount of light 38
Distribution of light 38
Time of burning 38
Bulb characteristics 38
Life of batteries 39
Life of cords 39
Mechanical and electrical characteristics 39
Pioneer lamp, approval No. 16 40
Gas-ignition tests 40
Tendency to unexpected extinction 41
Spilling of electrolyte 41
Light-producing capacity 41
Amount of light 41
Distribution of light 42
Time of burning 42

Lamps approved and description of tests—Continued.
Pioneer lamp, approval No. 16—Continued. Page.
Bulb characteristics 42
Life of batteries 42
Life of cords 42
Mechanical and electrical characteristics 42
Summary of results of tests 43
Procedure for Bureau of Mines tests of portable electric mine lamps 45
Definition of permissible 45
Conditions under which lamps will be tested 45
Requirements for approval 46
Character of tests to which lamps will be subjected 48
Design and construction 48
Safety devices 48
Determination of maximum short-circuit current 49
Time of burning, flux of light, intensity of light, and distribution of light 50
Variation in current consumption, candlepower, and life of bulbs 50
Leakage of electrolyte 50
Approval of permissible portable electric mine lamps 50
Approval plates 50
Notification of manufacturer 51
Scope of approval 51
Withdrawal of approval 51
Approval of lamp bulbs alone 51
Fees for testing portable electric mine lamps 51
Remittances 52
Synopsis of procedure to be followed in making application for tests, submitting material, conducting tests, and notifying applicant of results 52
Publications on electrical equipment for mines 54
Index 57

TABLE.

TABLE 1. Detailed information regarding the portable electric miner's lamps approved by the Bureau of Mines 44

ILLUSTRATIONS.

PLATE I. Equipment for testing safety devices of electric miner's lamps in explosive mixture of gas and air 12
II. *A*, Testing safety devices of an electric miner's lamp by striking it with a pick; *B*, Testing safety devices of an electric miner's lamp by striking it with a mallet 12
III. *A*, Machine for testing the nonspilling devices of mine-lamp batteries, assembled for "picking test"; *B*, Machine for testing the nonspilling devices of mine-lamp batteries, assembled for "shoveling test" 14
IV. Device for rotating lamp headpiece when measuring candlepower 14

Page.

PLATE V. *A*, Circle of light cast by lamp "X" as submitted for test; *B*, Circle of light cast by lamp "X" as approved by Bureau of Mines ---- 20

VI. *A*, Circle of light cast by lamp "Y" as submitted for test; *B*, Circle of light cast by lamp "Y" as approved by Bureau of Mines ---- 20

VII. *A*, Machine for testing mine-lamp cords, with guides removed to show amount of slack given cords and methods of mounting; *B*, Cord-testing machine with guides placed in position ---- 22

VIII. Edison lamp, approval No. 10 ---- 22

IX. Principal parts of Edison lamp ---- 24

X. *A*, Concordia hand lamp, approval No. 12; *B*, Wico lamp, approval No. 14 ---- 28

XI. Principal parts of Concordia hand lamp ---- 28

XII. *A*, General Electric lamp with steel casing, approval No. 13; *B*, General Electric lamp with cast aluminum casing, approval No. 13A ---- 30

XIII. Principal parts of General Electric lamp with steel casing ---- 30

XIV. Principal parts of General Electric lamp with cast aluminum casing ---- 32

XV. Principal parts of Wico lamp, approval No. 14 ---- 34

XVI. *A*, Concordia cap lamp, approval No. 15; *B*, Pioneer lamp, approval No. 16 ---- 38

XVII. *A*, Principal parts of Concordia cap lamp; *B*, Principal parts of Pioneer lamp ---- 38

FIGURE 1. Sectional view of headpiece of Edison lamp, showing principle of safety devices ---- 23

2. Sectional view of upper part of Concordia hand lamp, showing principle of safety devices ---- 26

3. Headpiece of General Electric lamp, side view ---- 30

4. Cross section of headpiece of General Electric lamp ---- 31

5. Sectional view of headpiece of Wico lamp, showing principle of safety devices ---- 34

6. Sectional view of headpiece of Concordia cap lamp, showing principle of safety devices ---- 37

7. Sectional view of headpiece of Pioneer lamp, showing principle of safety devices ---- 41

APPROVED ELECTRIC LAMPS FOR MINERS.

By H. H. Clark and L. C. Ilsley.

INTRODUCTION.

In various publications relating to safety in mining the Bureau of Mines has called attention to the hazards attending the use of open-flame lamps. An open-flame lamp is a potential source of danger in any coal mine. It may start a fire in a dry mine or an explosion in a gaseous mine.

The type of open-flame oil lamp now used in mines is the product of long evolution; its principal merit is simplicity. That such a light is unsafe in an atmosphere containing gas (methane) is evident, and Sir Humphrey Davy immeasurably increased safety in coal mines when he found means of protecting a lamp flame so that it would not readily ignite gas. The safety lamp, however, gives less light than the open-flame lamp, is not as simple in construction, is more cumbersome, and must be carried in the hand. Consequently, some miners are reluctant to abandon the open-flame lamp for the safety lamp, and risk their own lives and the lives of their fellow workers with the one rather than to be hampered by the other.

When small electric bulbs, with their clear, white light, and small electric storage batteries became available, the advantage of combining these elements for a miner's lamp seemed manifest. The combination looked simple and safe, but was soon found to have elements of danger and weakness peculiar to itself.

The Bureau of Mines showed, in 1911, that the breaking of a miniature electric bulb in an inflammable mixture of gas and air might ignite the mixture, so that only the fragile glass of the lamp bulb might stand between the miner and injury or death. There was also a possibility that sparks from the small battery would ignite gas, and the accidental spilling of the contents of a battery on the miner's clothing or skin was a possible source of injury.

The first portable electric miner's lamps that were carried in the hand, or with the battery strapped to the back and the light carried on the head, were exceedingly crude and were designed without appreciation of the sources of danger. The possibility, however, of producing a safe and practical miner's lamp by the proper combination and design of the elements available promised to increase safety so greatly that the Bureau of Mines attempted to stimulate better design and construction by careful investigations and the establishment of an approval system that would be an assurance to the miner and an advantage to the maker of the lamp. As a result various manufacturers submitted types of lamps, cords, and bulbs for test and cooperated with the bureau in efforts to so improve the safety features of these devices that the lamps would pass specific tests for approval.

SCOPE OF REPORT.

This bulletin describes the development of a number of types of lamps that meet the requirements, and discusses in detail the features and qualities of these lamps, which in November, 1916, were being put into use by mine operators in this country at the rate of about 2,000 a week.

Those lamps that meet a certain minimum specification are approved by the Bureau of Mines. No manufacturer is required to obtain this approval, but the manufacturers have voluntarily submitted their lamps for investigation and criticism. None of the lamps submitted to the bureau was found acceptable in its original form, and the cooperation with the Bureau of Mines of the makers of lamps, bulbs, and cords has extended over the past three years, and the art of making safe miner's electric lamps has thereby been developed much more rapidly than would have been the case had the development been solely the result of commercial exploitation. Also, it is believed that by this method the mining public has been saved expensive and dangerous experiments, which might have cost many lives and for a time condemned the electric lamp as an impracticable device.

The first specification of the Bureau of Mines as to conditions and requirements for testing electric mine lamps was set forth in Schedule 5,[a] issued in April, 1913, and the requirements were limited to the prevention of gas ignition. Three lamps were tested and approved under this schedule. Accumulated experience by the bureau indicated the desirability of a broader basis of approval, and another schedule, Schedule 6,[b] was issued in February, 1914, which was revised to the present form (Schedule 6A) in February, 1915. Schedule 6A, which is given in full on pages 45 to 53, includes requirements for both the safety and the practicability of the lamp. The original approvals under Schedule 5 were withdrawn and the lamps were retested in accordance with Schedule 6A, and a broader and more comprehensive approval issued to those lamps that passed the later tests.

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[a] Fees for testing permissible portable electric lamps, character of tests, and conditions under which lamps will be tested; Bureau of Mines, 1913, 8 pp.

[b] Procedure for establishing a list of permissible portable electric mine lamps, Bureau of Mines, 1914, 12 pp.

The bureau grants its approval as an indication of its confidence in the excellence of the lamp as a safe and practical device for lighting mines, especially those mines in which there may be inflammable or explosive mixtures of gas and air. The approval is conferred only after thorough examination and tests have shown that the lamp reaches the standard established by the bureau.

Seven portable electric mine lamps had been approved prior to August, 1916, as follows: Edison lamp, approval No. 10, approved February 24, 1915; Manlite lamp, approval No. 11,[a] approved July 16, 1915; Concordia hand lamp, approval No. 12, approved July 26, 1915; Wico lamp, approval No. 14, approved June 10, 1916; Concordia cap lamp, approval No. 15, approved June 17, 1916; General Electric lamp, approvals No. 13 and No. 13A, approved July 11, 1916; Pioneer lamp, approval No. 16, approved July 21, 1916.

Each of these lamps is identified by a plate that bears the seal of the Bureau of Mines, the approval statement, and the number of the approval. No lamp is considered as approved unless it bears this plate.

ACKNOWLEDGMENTS.

Acknowledgment is made of the helpful cooperation that has been given by the manufacturers of the various lamps investigated and approved, and by the manufacturers of electric conductors who made and submitted for test special types of lamp cords. Special acknowledgment is due to Messrs. R. P. Burrows and P. F. Bauder, of the National Lamp Works of the General Electric Co., and Mr. C. M. Bunnell, of the Edison Lamp Works of the General Electric Co., for their successful efforts in developing satisfactory bulbs for mine lamps.

The authors are also indebted to Mr. J. T. Jennings, electrical engineer of the Philadelphia & Reading Coal & Iron Co., for information based on his broad experience with portable electric mine lamps in actual service.

HISTORY OF THE INVESTIGATION.

At the outset of the bureau's work on electric mine lamps little reliable information on such lamps seemed to be available, for the lamps had not been used extensively then, although some pioneer work had been done by several manufacturers. The development of those lamps that have since received the bureau's approval was either in a very early stage or had not started.

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[a] Approval rescinded for cause April 23, 1917.

IMPORTANCE OF DEVELOPING AN ELECTRIC LAMP FOR MINERS.

The bureau's attention was given to portable electric lamps for miners because the use of such lamps would manifestly decrease the hazards of mining operations by the elimination of flames (either open or inclosed) and by providing more light in a more convenient form than the safety lamp. Although the investigation of portable electric lamps was undertaken wholly in the interest of safety, the bureau was not concerned exclusively with the safety of the lamp. An electric lamp was not a relatively dangerous piece of equipment that was indispensable to the operation of mines, for other lamps were available, but on the contrary was a relatively safe piece of equipment, not useful unless practicable, and, if practicable, very desirable.

It was the bureau's hope that the electric lamp could be made so practicable that the miner would avail himself of the advantages of its safety features, as it was manifestly futile for the bureau to urge the adoption of an impracticable device, no matter how safe it might be. Therefore the bureau undertook to conduct examinations and tests and to approve those lamps that were found to be safe and practicable for general service in mines, in order to advance as rapidly as possible the manufacture of portable electric mine lamps and to avert the possibility of having a desirable safety device condemned on account of repeated failure in service of impractical forms.

ESSENTIAL QUALIFICATIONS OF ELECTRIC MINE LAMPS.

Safety is a prime requisite for any mine lamp and was the first consideration of the bureau's engineers, who investigated three possible dangers: The liability of the lamps to cause ignitions of gas, their tendency to go out unexpectedly, and the liability of the electrolyte being spilled.

A lamp must be practicable or it is useless as a safety device. The practicability of a portable electric mine lamp is measured by ability to satisfy completely the requirements of the user, which are as follows:

1. The lamp shall produce for the necessary period a suitable amount of properly distributed light.
2. The lamp shall repeat this production whenever required with a minimum number of interruptions and at a cost commensurate with the benefits received.
3. The lamp shall not spill or leak electrolyte.

4. The lamp shall not be heavy enough to be a burden to the user.

After all of the lamps available at the outset of the investigation were examined and numerous preliminary tests had been made, it was decided that the determination of the safety and practicability of a portable electric mine lamp as set forth above required the investigation of the qualities enumerated below:

Liability to ignite gas.
Tendency to unexpected extinction.
Liability to spill electrolyte.
Light-producing capacity—
  (1) Amount—
    (*a*) Candlepower.
    (*b*) Flux.
  (2) Distribution—
    (*a*) Angle of light stream.
    (*b*) Smoothness.
  (3) Time of burning.
Bulb characteristics.
Life of batteries.
Life of cords.
Mechanical and electrical characteristics—
  (1) Durability.
  (2) Reliability.
  (3) Ease of repair.

Schedule 6A, based on the above analysis, sets forth the requirements that the bureau considers indispensable to the approval of portable electric mine lamps and states the test procedure that is followed in making the investigation. The tests reported in this bulletin are based upon the requirements of that schedule.

QUALITIES INVESTIGATED AND TEST PROCEDURE.

LIABILITY TO IGNITE GAS.

Previous experiments of the bureau, reported in Technical Paper 47,[a] had demonstrated that bulbs such as are used in miners' electric lamps can ignite explosive mixtures of mine gas and air if the bulbs are broken while the filaments are glowing at their normal operating temperature. The bureau had, therefore, prescribed in its testing requirements that mine lamp outfits must be provided with a device that would extinguish the filaments if the bulbs were broken. The standard tests to determine the effectiveness of such devices were conducted, in general, as follows:

The headpiece of the lamp was placed in a box that was filled with the most explosive mixture of natural gas—natural gas supplied to the city of Pittsburgh—and air, and the bulbs were then broken while glowing with normal brilliancy. The breaking was accomplished by means of a tool mounted on the end of a sliding rod and shaped like a miner's pick (Pl. I) that was directed successively at different points in the headpiece, against which it was driven with considerable force by means of a wooden mallet. If, as a result of these tests, a bulb was broken without becoming extinguished, the safety device was considered to have failed, even though the gaseous mixture was not ignited.

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[a] Clark, H. H., Portable electric mine lamps; Tech. Paper 47, Bureau of Mines, 1913, 8 pp.

Tests were also made without the use of gas by striking the headpiece a violent blow with a hammer, a mallet, or a short-handled pick (see Pl. II), or by swinging the headpiece on its cord against stone, iron, or wood. If by this treatment a bulb was broken but not extinguished by the safety devices these were considered to have failed. A single failure of the safety devices made it necessary to redesign them.

Tests reported in Technical Paper 47[a] showed that gas ignition as the result of noninductive sparks produced by the batteries of portable electric mine lamps was a remote contingency, but might occur if the current broken was sufficiently large. A safe maximum value of current for a single cell of lead-acid storage battery was found by test to be 125 amperes, and for two cells of alkaline storage battery (the usual grouping in mine lamps) to be 85 amperes. Tests were therefore made to determine whether or not these currents could be exceeded under any conditions of service, normal or accidental; if excessive currents could be obtained changes in construction were required.

The "standard" short-circuit test was made by connecting the battery to be tested directly across an ammeter in series with a resistance equal to the resistance that the lamp circuit would have if a complete short circuit existed just outside the battery casing.

TENDENCY TO UNEXPECTED EXTINCTION.

This quality bears on safety and is likely to exist in lamps of inferior design and construction on account of loose contacts, low capacity, or poorly designed safety devices.

The standard method of investigating the tendency to unexpected extinction as the result of jars or shocks consisted in dropping the headpiece of cap-lamp equipments, and the hand lamp complete, on a concrete floor from a height of 6 feet.

The likelihood of extinction caused by inferior construction giving rise to uncertain contacts was determined by inspection and confirmed by actual experience in operating the lamps in connection with the bulb-life tests made in the laboratory.

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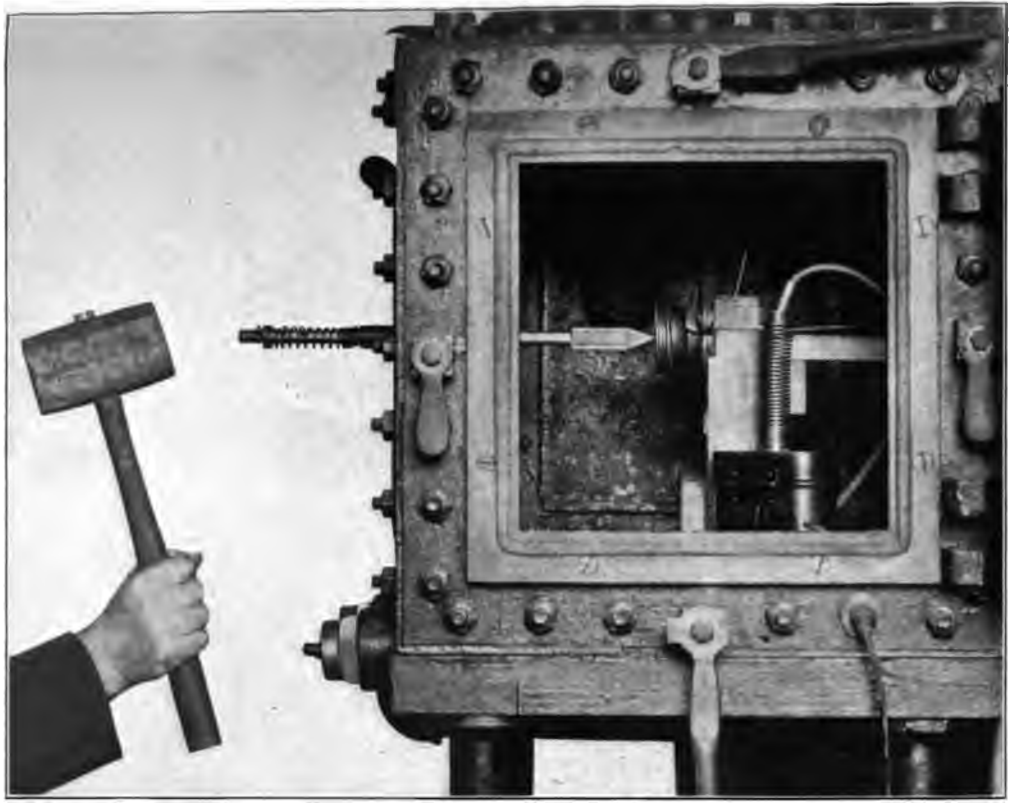
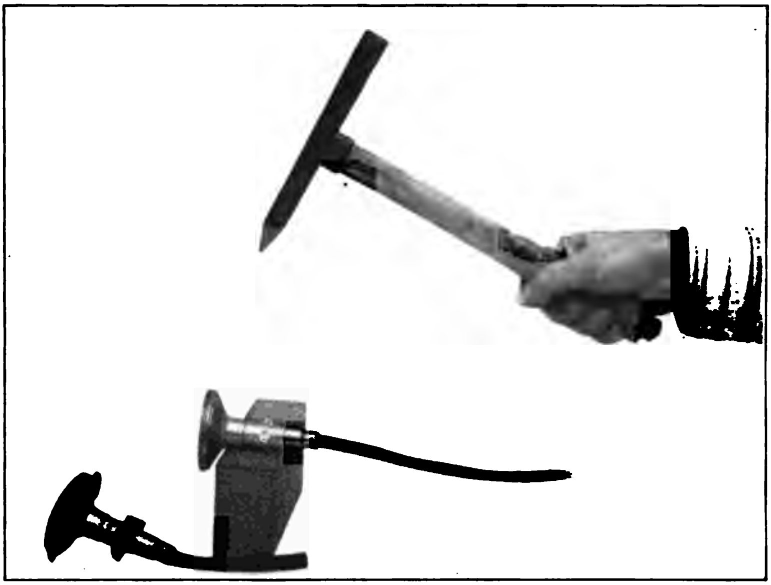
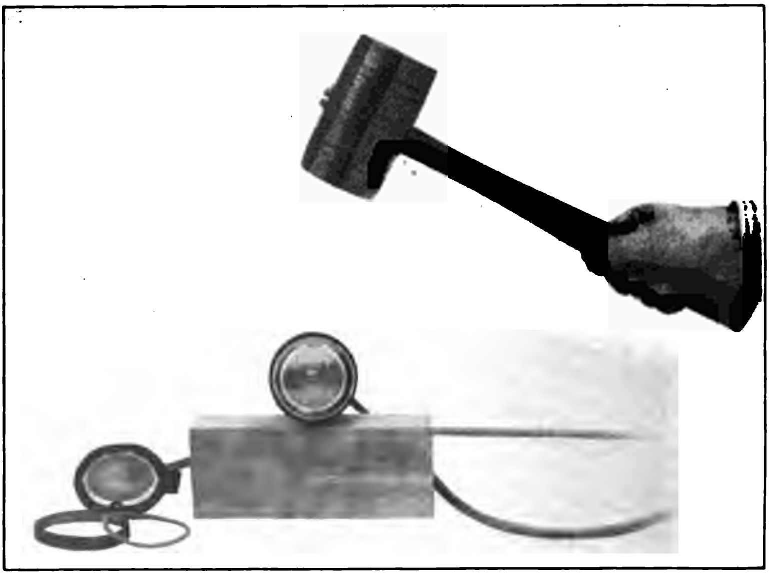
[a] Clark, H. H., Portable electric mine lamps; Tech. Paper 47, 1913, pp. 7-10.



EQUIPMENT FOR TESTING SAFETY DEVICES OF ELECTRIC MINER'S LAMP IN EXPLOSIVE MIXTURE OF GAS AND AIR.



A. TESTING SAFETY DEVICES OF AN ELECTRIC MINER'S LAMP BY STRIKING IT WITH A PICK.



B. TESTING SAFETY DEVICES OF AN ELECTRIC MINER'S LAMP BY STRIKING IT WITH A MALLET.

LIABILITY TO SPILL ELECTROLYTE.

The liability of the electrolyte being spilled bears on safety, and the fault may well be possessed by any batteries not designed and constructed with much care.

The standard "spilling test" was conducted by mounting in a testing machine 4 cells, selected at random from 50 submitted for test, and operating them for seven successive hours by shaking them back and forth in two general ways, designated as the "picking test" and the "shoveling test" from the services that the tests were intended to simulate.

Plate III, *A*, shows the testing machine with batteries assembled for the "picking test." Four cells were grouped in an upright position on a horizontal platform that was hinged on a vertical axis at a point 10 inches from the center of the group, and otherwise arranged so that it could be swung back and forth through a horizontal arc of 23°. Swinging the cell back and forth 72 times a minute for two hours constituted a "picking test," and two tests were given during the seven-hour period.

Plate III, *B*, shows the testing machine with batteries assembled for the "shoveling test." Four cells were grouped on their sides on a horizontal platform that was hinged on a horizontal axis at a point 40 inches below the center of the group and otherwise arranged so that the cells could be swung back and forth through a vertical arc of 21°. This operation repeated 48 times a minute for one hour constituted a "shoveling test." and three such tests were made during the seven-hour period.

Before the seven-hour test was begun the batteries were fully charged and allowed to stand on open circuit for one hour immediately preceding the "spilling test," which was made with the equipments completely assembled and with the bulbs burning. If any battery spilled one drop of electrolyte during the seven hours of test (three hours of "shoveling" and four hours of "picking") the venting system or devices had to be redesigned and retested.

LIGHT-PRODUCING CAPACITY.

The light-producing capacity of a lamp is the first essential of practicability and bears directly on safety.

CANDLEPOWER AND LIGHT FLUX.

The investigation of light-producing capacity requires the consideration not only of the amount of light produced but also of the character of its distribution, which includes the angle of the stream of light and the evenness of illumination of the surface upon which the stream is directed.

The light-producing capacity of a lamp is its capacity to produce a sufficient amount of properly distributed light. The determination of the amount requires the measurement of the candlepower and the area over which it acts, so that the total volume of light, as well as its intensity, may be known.

The bureau fixed as a minimum requirement of candlepower (intensity) and light flux (total light) the values obtained from a standard safety lamp.[a] It was hoped, however, that the electric lamps submitted would considerably exceed the limits set, and that hope has been realized.

The standard tests for light-producing capacity were conducted as follows:

The candlepower and light flux of the lamps were measured by means of a Leeds & Northrup photometer using a Lummer-Brodhun contrast screen. The standard lamps were made and standardized by the National Lamp Works of the General Electric Co. All electrical instruments used for both standard and test lamps were compared at intervals with a Wolff potentiometer, and were compared before and after each test with a secondary standard used for no other purpose. Both the standard and the test lamps were operated from lead storage batteries.

The candlepower and light flux of a portable electric mine lamp depend on the bulb used, the condition of the battery, and the character and condition of the reflector. The candlepower value also depends on the method used to determine it; for instance, whether it is measured in one direction or whether it is measured in several and the average taken.

The bulb variable was eliminated by taking the average performance of 100 bulbs. The battery variable was eliminated by obtaining for 20 cells the average voltage after discharging under normal conditions for 1 hour and for 12 hours. The reflector variable was eliminated by averaging the performance of 6 reflectors.

It was considered that the best way to measure candlepower was to determine the average over the entire stream of light produced by the lamp, and this was done by mounting, as shown in Plate IV, a fully assembled headpiece so that it could be rotated rapidly around an axis that coincided with the axis of symmetry of the cone of light. The first reading on the photometer was then made directly in front of the rotating headpiece, which was swung through an arc of 2½° for each successive reading until the light was no longer visible. By making the proper calculation from the data thus obtained, a very accurate average candlepower value was reached.

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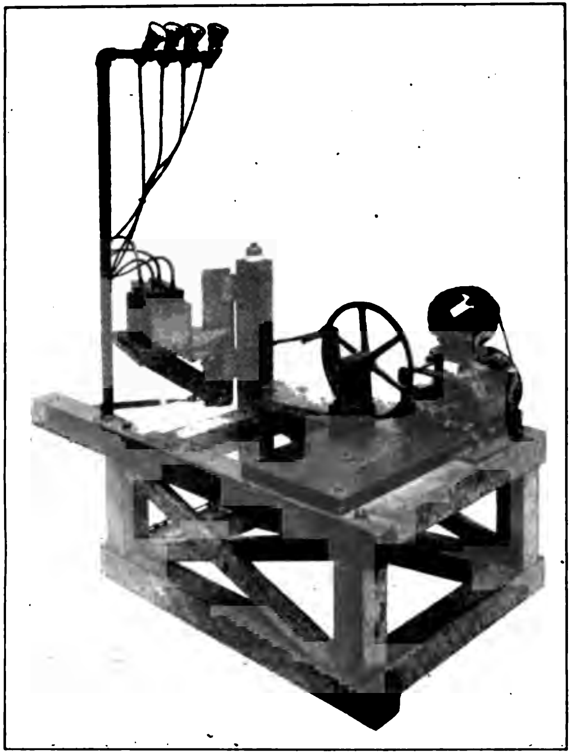
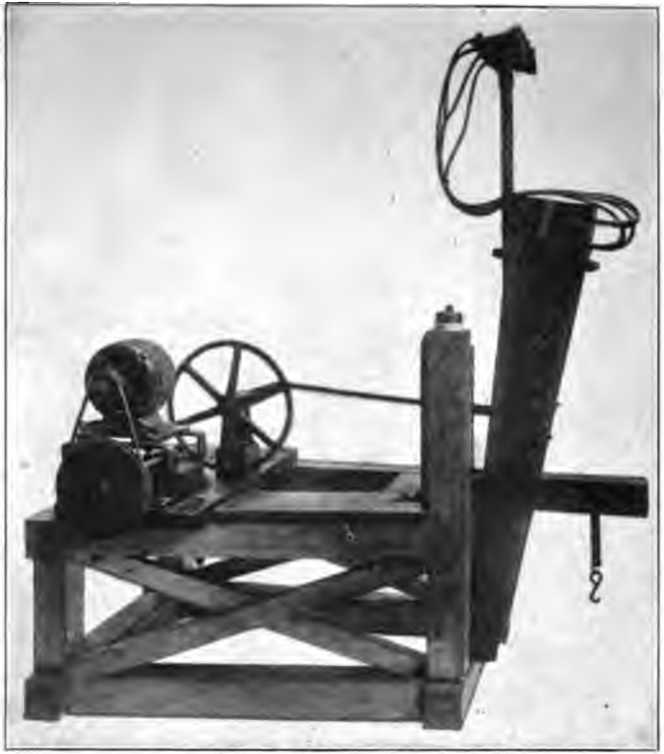
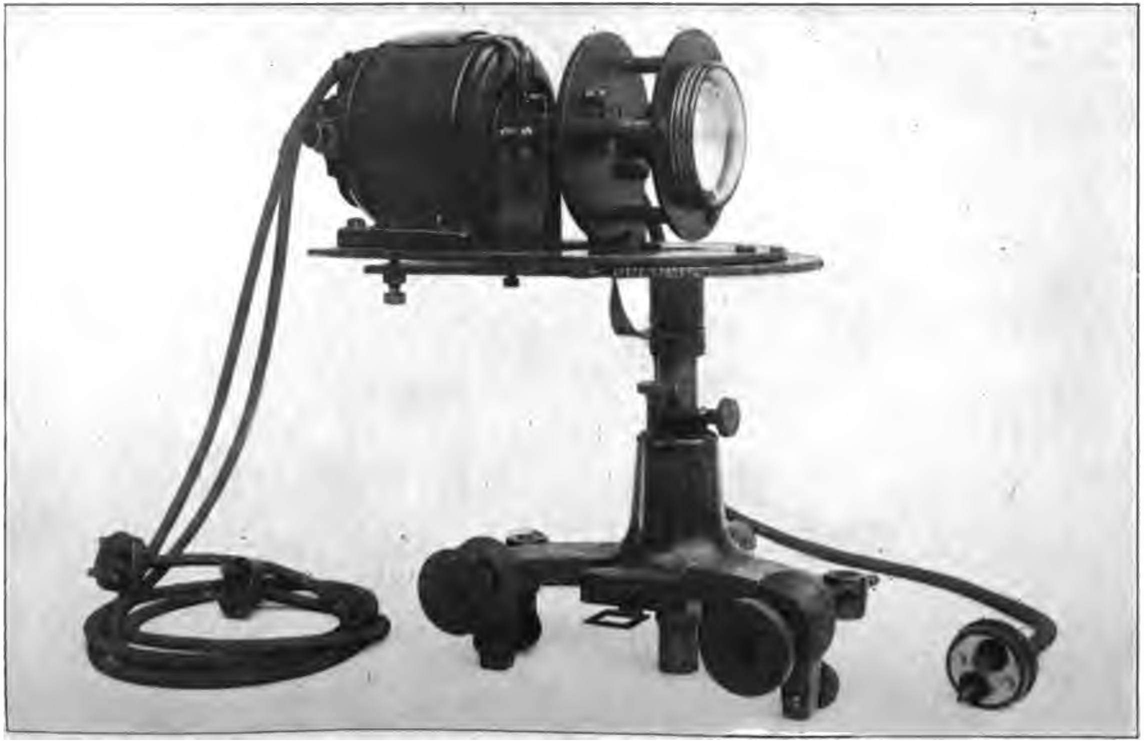
[a] Clark, H. H., Permissible electric lamps for miners: Tech. Paper 75, 1914, p. 14.



A. MACHINE FOR TESTING THE NONSPILLING DEVICES OF MINE-LAMP BATTERIES, ASSEMBLED FOR "PICKING TEST."



B. MACHINE FOR TESTING THE NONSPILLING DEVICES OF MINE-LAMP BATTERIES, ASSEMBLED FOR "SHOVELING TEST."



DEVICE FOR ROTATING LAMP HEADPIECE WHEN MEASURING CANDLEPOWER.

Light from lamps that are carried on the cap is emitted in a cone of rays. The bureau adopted 130° as the minimum angle of this cone that would insure that the eyes of a man wearing the lamp would normally rest on a lighted surface.

The agreement of the lamps with this requirement was determined by means of the readings taken on the photometer while the candlepower was being measured.

The smoothness of distribution of the light was determined in two ways. The readings taken while measuring candlepower with the photometer were plotted as a curve, from which was calculated the corresponding values of illumination; also, the light from six equipments in succession was cast upon a white screen, and the presence or absence of shadows or of areas of contrasting illumination was observed directly.

TIME OF BURNING.

The time of burning on a single charge of the battery is a factor of the capacity of the lamp to produce light. A consideration of this quality is essential to the determination of practicability and has a remote bearing on safety.

The time of burning of an equipment is manifestly the time during which the equipment will produce with one charge of battery at least the minimum amount of light that the bureau prescribes. The standard time-of-burning test was conducted as follows: It was proved by test that a fairly constant relation existed between the candlepower of a bulb and the average candlepower of the stream of light that the bulb would produce when installed in the headpiece of a lamp. As a minimum value had been established for the average lamp candlepower, the determination of the minimum allowable bulb candlepower was a simple mathematical problem. Each of 100 bulbs was measured to determine the voltage that must be impressed upon the bulb to give the minimum bulb candlepower. This value of voltage was then referred to the lamp-battery discharge voltage curve, which was plotted between voltage and hours of discharge, to ascertain for how many hours the battery could be discharged before its voltage fell below the value that was required to give the minimum amount of light. Thus a time of burning was established for each of 100 bulbs and the average of the 100 determinations was taken as the time of burning of the equipment.

BULB CHARACTERISTICS.

The characteristics of the bulbs used with portable electric mine lamps are weighty factors in determining practicability and have a certain though less important bearing on safety. The bulb is the complement of the battery, and if the bulb is not designed to work with the battery the results may be unsatisfactory. Unless the performance expected of the bulb is carefully defined and considered in the design of the bulb the resulting lack of uniformity will affect undesirably the reliability of performance.

The standard tests of bulbs were conducted as follows: The bulbs were measured for uniformity of current consumption, uniformity of candlepower, and length of life. From 1,000 bulbs submitted 100 were selected at random and burned for 12 hours to "season" them. Their current consumption and candlepower were then measured at a constant voltage and the value obtained for each bulb was compared with the average of the 100 bulbs to see if any exceeded the tolerance in uniformity prescribed by Schedule 6A.

The life of the bulbs was measured by operating 20 of them in standard mine-lamp equipments and 80 others in multiple on a large battery that had the same discharge voltage characteristics as the mine-lamp batteries. The bulbs tested on the mine-lamp batteries were burned 12 hours on a single charge of battery. The bulbs tested on the large battery were operated for various periods on a single charge, but the starting and stopping voltage for each discharge was made the same as for the mine-lamp batteries, thus insuring uniformity of test conditions.

LIFE OF BATTERIES.

The life of batteries is manifestly of considerable importance in the determination of practicability because it bears directly on cost of upkeep.

The life of a properly designed and constructed mine-lamp battery should be not less than three hundred 12-hour discharges, and the makers of some mine-lamp equipments guarantee twice this life. It was not practicable for the bureau to make tests of this duration, but the behavior of the batteries was carefully observed while the life test of bulbs was being conducted, and if the batteries exhibited any tendency to early deterioration, or if they required excessive care and attention, they were not approved.

LIFE OF LAMP CORDS.

The life of the lamp cords has such a bearing on practicability that at one time it was almost the limiting feature. Cords of short or uncertain life not only increase the cost of upkeep, but also affect adversely the reliability of the equipment by causing unexpected interruptions. Early in the development of electric lamps for wearing on a miner's cap it became manifest that the design of the cord connecting the battery to the headpiece presented a problem of some difficulty. Long after other weak points of the equipment had been eliminated the cord continued to give trouble, until finally it came to be regarded as the weakest part of a mine-lamp equipment. The bureau therefore undertook to assist in the development of a satisfactory cord, with the gratifying results reported in pages 19 to 22, where is set forth the method of conducting the standard tests of cords.

MECHANICAL AND ELECTRICAL CHARACTERISTICS.

The mechanical and electrical characteristics of an equipment determine the degree of practicability and safety of the equipment, as they bear on continuity of service, the likelihood of sudden interruptions, and the cost of upkeep.

Examination for these characteristics involves consideration of, the strength of the various parts, their ability to function under continuous service, and the extent to which they require unusual attention or expert care in operation and maintenance.

The standard method of investigating the mechanical and electrical characteristics of the lamps was as follows: The mechanical strength of the lamps was determined by test and by inspection. Hand lamps and the headpieces of cap lamps were tested by dropping 10 times on a concrete floor from a height of 6 feet. If a lamp or a headpiece was materially deformed as the result of these tests, changes in construction were required. The test of the complete cap-lamp equipment was made with four batteries, each of which, unless broken previously, was dropped 20 times upon a wooden floor from a height of 3 feet. If any battery developed a leak before it had been dropped 6 times, or if the four batteries could not be dropped an average of 10 times without developing a leak, the jar of the battery had to be made more substantial. Then the tests were repeated.

The consideration of the mechanical and electrical features of a lamp was begun as soon as the lamp was submitted and was completed only when the lamp was finally accepted. The long and varied experience of the bureau's engineers with the operation and testing of lamps has provided them with considerable information which was placed at the service of mine-lamp makers in the form of constructive criticism of the details of construction. Especial attention was given to directing the development so as to insure uninterrupted service with a minimum of repairs, adjustments, or replacements of parts. If a lamp when submitted was manifestly weak in any mechanical feature the bureau required that the defect be remedied before the formal investigation of the lamp was begun. If mechanical or electrical weaknesses were discovered during the course of the investigation the lamp manufacturer was required to strengthen the weak part before the approval was issued. After the completion of all the tests two equipments complete with all modifications were examined in detail and final changes were required if essential, or recommended if merely desirable.

DEFICIENCIES REMEDIED.

The equipments as first submitted to the bureau were deficient in some or all of the following features: Safety device, angle of light stream, distribution of light, life and uniformity of bulbs, cord clamps, strength of cord armor, life of cords, suitable means for locking the battery casings, and workmanship. These deficiencies have all been overcome to a greater or less extent, and most of them completely, in the equipments that have been approved by the bureau.

SAFETY DEVICE.

Each manufacturer developed his own safety device, but the bureau has been largely instrumental in perfecting, as described below, the other features mentioned.

ANGLE OF LIGHT STREAM.

From the outset of the investigation the bureau engineers emphasized the need of a wide angle of light stream so that the eyes of the lamp user may normally encounter no unlighted surfaces. If the angle of the light stream is the same as a man's angle of vision the lamp produces the effect of complete illumination with its attendant freedom of movement. Experiment showed that a light stream having an angle of 130° produced very nearly the desired effect and that value was adopted by the bureau as a minimum requirement. The angle of the light stream of four of the seven types of lamps that have been approved by the bureau was increased to 130° from values as low as 85° in one instance.

DISTRIBUTION OF LIGHT.

The amount of light produced by any of the approved portable electric mine lamps is not large, the most powerful producing about 5.3 lumens (a good flame safety-lamp produces about 3 lumens), but it is adequate if distributed evenly over the surface illuminated. Uneven distribution not only tires the eyes, but is inefficient, because light from the brighter parts causes the pupil of the eye to contract so that the parts more faintly illuminated can not be clearly seen.

Highly polished reflectors are more efficient than dull surfaces but the distribution of light is as a rule decidedly uneven and the surface illuminated is covered with sharply contrasting areas of bright and faint illumination. In order to avoid such sharp contrasts, reflecting surfaces that diffused the light had to be used. Also the design of the shape of the reflectors required considerable skill and care in order to distribute the light evenly over the surface and prevent concentration of light in the center of the light stream. Only one of the lamps as submitted met the requirements for distribution of light (see p. 47) and various changes were necessary in the others. Plates V and VI show two instances where the distribution, and in one instance the angle of light, was improved by the manufacturer at the bureau's request.

BULBS.

One of the most important parts of the lamp equipment is the bulb, which converts into light the energy supplied by the battery. Evidently the bulb should operate at the highest practicable efficiency. The efficiency of a bulb increases with the operating temperature of the filament but unfortunately the life of a bulb decreases as the operating temperature rises so that an increase in efficiency is unavoidably accompanied by a decrease in life.

Therefore, to produce a bulb for mine-lamp service in which the efficiency and the life of the bulb have each received proper consideration required more care and thought than had been given before to the design of miniature bulbs. Uniformity in bulb performance is always a desirable thing, but is especially desirable if the bulb is to be properly designed for the equipment with which it is used. That is to say, if the bulb is designed to give the maximum possible efficiency consistent with the life requirements a departure from this design can not fail to give too short a life or too low an efficiency.

At the outset of its investigation the bureau found that the types of miniature lamp bulbs on the market were unsatisfactory for use in mine lamps and for months the lamp manufacturers could not obtain American bulbs that the bureau could approve. However, the bureau was successful in arousing the interest of bulb manufacturers in making bulbs better suited to mine-lamp service and with their cooperation bulbs suitable for this service have been designed and manufacturing methods have been developed that already insure a gratifying degree of uniformity and promise to accomplish even better results soon.

CORD AND ACCESSORIES.

Long after the remainder of the equipment was fairly well developed the cord that connects the battery and the bulb was still giving trouble of a most trying sort by developing open circuits, partial or complete, often unexpectedly and always after a comparatively short term of service.

Part of this trouble was caused by the lack of means for fastening the cord to the shell of the battery cover and the headpiece in order to relieve the electrical connections of the strains that the cord must receive in service. Such a means of fastening was one of the first requirements of the bureau and all approved lamps have an adequate device for clamping the cord.

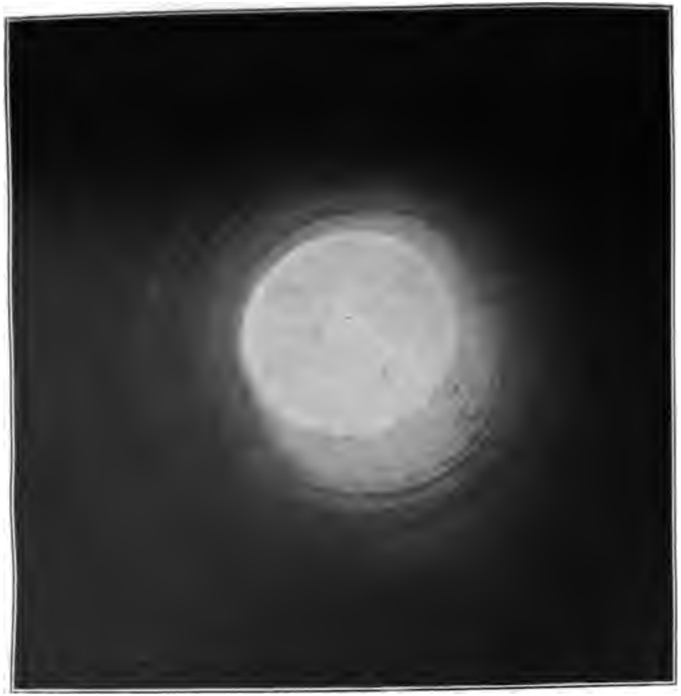
Trouble with the cords was due in even greater degree to inadequate reinforcement where the cords left the battery casing and where they entered the headpiece. Situations of this sort have always been difficult for flexible electric conductors and the electric mine lamp has been no exception. The earlier types of armor were so weak that often they were outlasted by the cord which they were designed to protect. The lamp manufacturers, in cooperation with the bureau, have now developed protective armors that give satisfactory service, although the design of some cords can and no doubt will be improved.

The need for better cords as demonstrated by actual service in mines became so acute that, as stated earlier in this report, the bureau endeavored to assist the development of the lamp by undertaking, with the cooperation of the manufacturers of insulated electric conductors, the development of a suitable mine-lamp cord. Letters were sent to various wire and cable manufacturers stating that the bureau would make, without charge, comparative tests of such cords as were submitted, preference being given to cords made in accordance with the bureau's suggestions, which were outlined in a short memorandum that was sent to each manufacturer addressed. The following companies expressed a willingness to submit for test, without charge to the bureau, samples of flexible cord designed for mine-lamp service: General Electric Co., John A. Roebling's Sons Co., Kerite Insulated Wire & Cable Co., Okonite Co., Simplex Wire & Cable Co., and Standard Underground Cable Co.

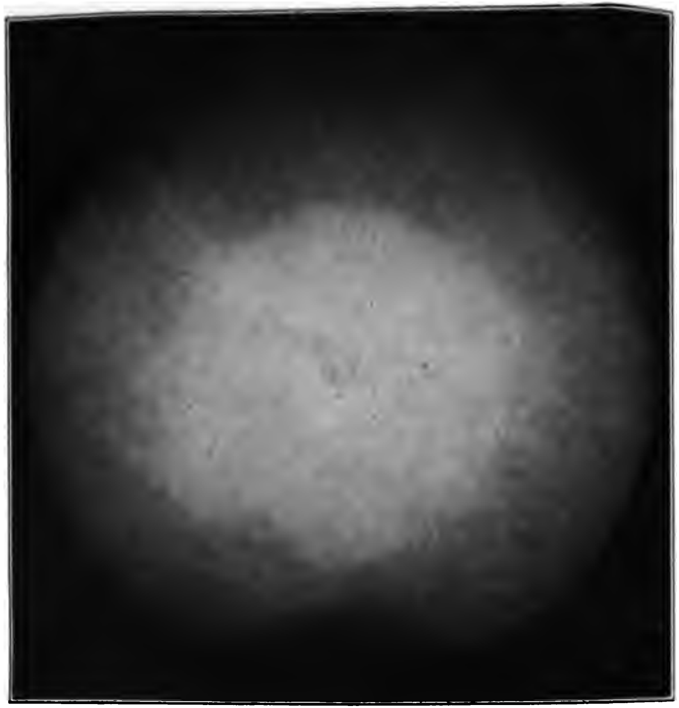
Eighteen types of cord were submitted and tested and a report of the results was sent to each manufacturer cooperating and to each manufacturer of portable electric mine lamps.

CORD-TEST PROCEDURE.

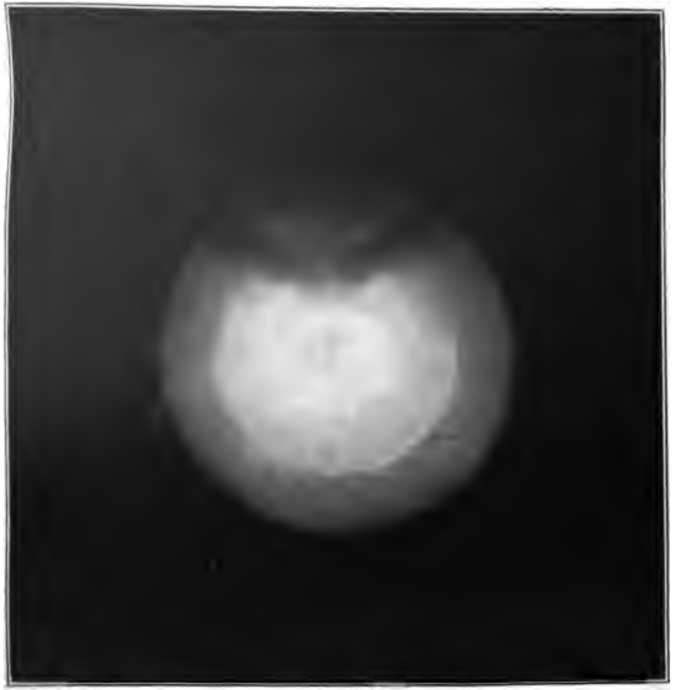
Portable electric mine-lamp cords are destroyed by the switching back and forth while attached to the body of a working miner. In the past this switching has caused the wires to break before the insulation was even damaged. The test to which these cords were submitted by the bureau imitated the switching of the cords in service, but the amplitude, speed, and frequency of the switching was



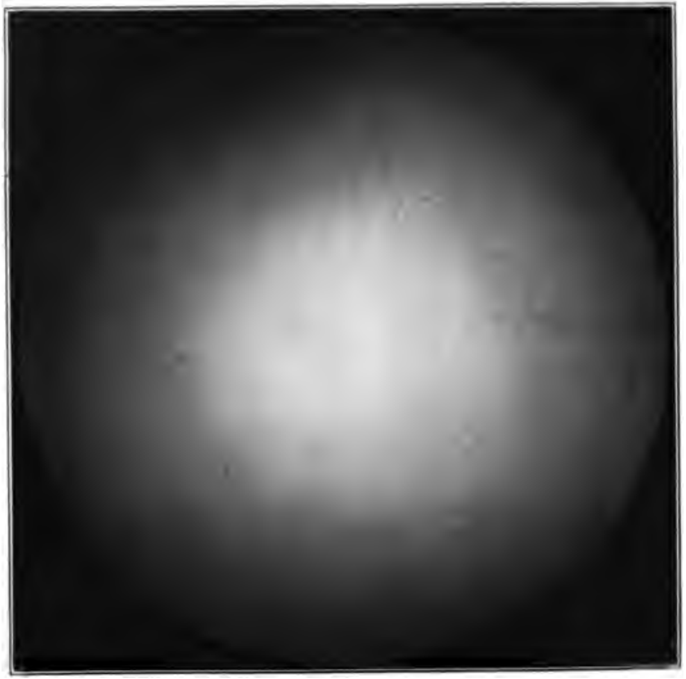
A. CIRCLE OF LIGHT CAST BY LAMP "X" AS SUBMITTED FOR TEST.



B. CIRCLE OF LIGHT CAST BY LAMP "X" AS APPROVED BY BUREAU OF MINES.



A. CIRCLE OF LIGHT CAST BY LAMP "Y" AS SUBMITTED FOR TEST.



B. CIRCLE OF LIGHT CAST BY LAMP "Y" AS APPROVED BY BUREAU OF MINES.

exaggerated in order to hasten the test results and to provide a factor of safety.

DESCRIPTION OF CORD-TESTING DEVICE.

The testing device (Pl. VII) consisted of a rectangular frame that was swung back and forth by a motor, the cords being so mounted on this frame that they slatted as it oscillated. The frame was hung in a vertical position from a horizontal shaft that passed along its upper edge. Ten battery covers were mounted on the lower edge of the frame, their tops being 48 inches below the center of the shaft about which the frame swung. One end of each cord was fastened in the battery cover by the means provided and the other end was fastened to the frame at a point 24 inches above the battery cover. The length of the cord between fastenings was 36 inches, which provided enough slack for the slatting of a cord when the frame oscillated.

The frame was swung back and forth 92 times a minute through a vertical angle of 50° 30′, causing the cords to slat vigorously. Each cord was moved to and fro between horizontal stationary guide bars made of oak, sandpapered and shellacked, and spaced 2½ inches apart. These bars, which were placed about half-way between the points at which the cord was fastened, had an abrading action on the outer covering of the cord, although the abrasion was decidedly slow. An automatic counter was connected to the frame for registering the number of oscillations.

ELECTRICAL CONNECTIONS.

Ten cords were tested simultaneously, the cords being connected in series with each other and with a relay that stopped the motor automatically when the test was completed. The motor was operated until the cord system open-circuited or until it increased in resistance to 10 times the original value. Poor cords were tested until the circuit opened, good cords until their resistance increased 10 times. When the motor was stopped the following method was used to determine which of the 10 cords had developed excessive resistance or an open circuit.

Each cord as connected for test was shunted across a miniature incandescent bulb and all cords were connected in series. Means were provided for switching into this series a miniature bulb shunted across a resistance of 10 times the normal resistance of one cord. This bulb, with the resistance, was called the "comparator." When current was passed through the comparator and the 10 cords in series the shunted lamps glowed in proportion to the resistance across which they were shunted. Therefore the bulb of any cord whose resistance had increased to 10 times the normal glowed as brightly as the comparator bulb, and conversely any cord whose bulb was less brilliant than the comparator bulb had not reached the maximum resistance allowable.

RESULTS OF CORD TEST.

When the investigation began the best lamp cord then tested in the laboratory of the bureau averaged about 30,000 operations. During the cooperative work with the manufacturers of electric conductors there were developed two cords that gave more than 100,000 operations on the average, three that gave more than 200,000 operations, and one that gave over 1,000,000. It is therefore believed that the problem of the mine-lamp cord has been solved.

LAMPS APPROVED AND DESCRIPTION OF TESTS.

The lamps that were approved by the bureau and the tests of these lamps are described in the following pages. All tests reported here were made under the provisions of Schedule 6A, which is given in full on pages 45 to 53, and unless stated to the contrary were the standard tests described previously in this paper.

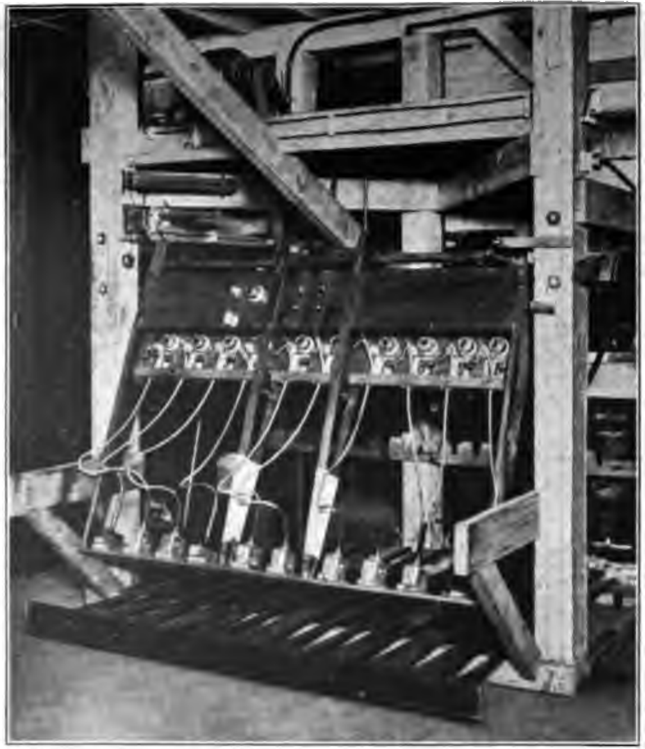
EDISON LAMP, APPROVAL NO. 10.

The Edison lamp, submitted by the Edison Storage Battery Co., of Orange, N. J., is a cap lamp operated by two cells of alkaline battery. The lamp weighs about four pounds and has a capacity of 35 to 44 lumen-hours (depending on the brand of bulb used) for a 12-hour shift. It has a snap lock opened with a key. Detailed specifications for this lamp are given in Table 1 (p. 44). The assembled equipment is shown in Plate VIII, and its component parts in Plate IX.

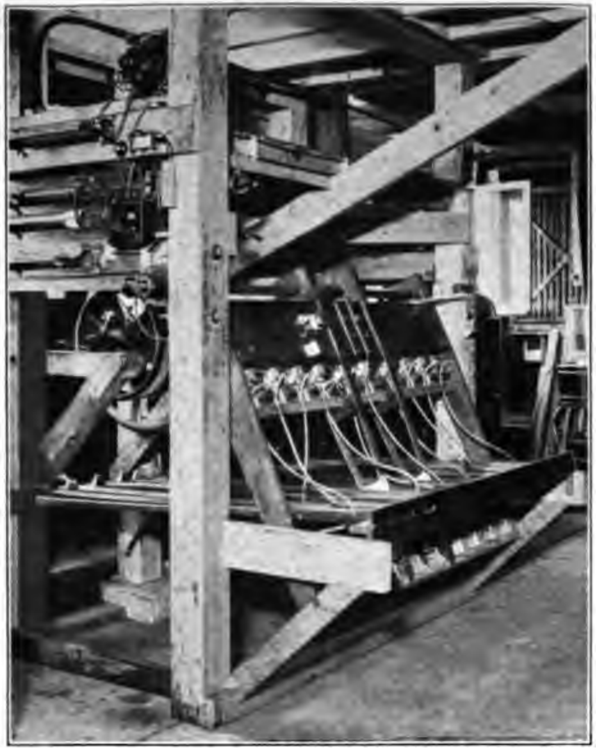
The lamp was tested in accordance with the requirements of Schedule 6A, and after several changes in design and construction had been made was approved February 24, 1915, for safety and for practicability and efficiency in general service (approval No. 10).

GAS-IGNITION TESTS.

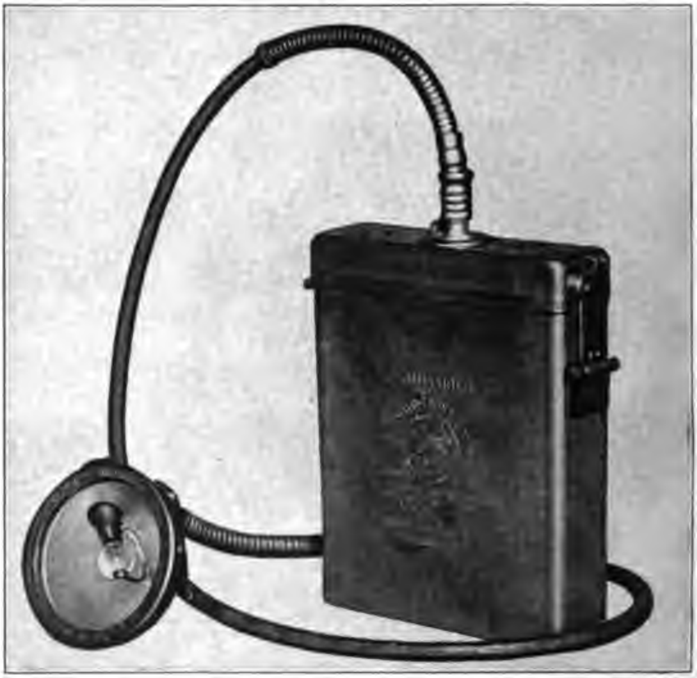
The device that is designed to prevent gas ignition is shown in figure 1. It consists of a spring *a* that holds the bulb in circuit as long as the bulb is intact, but allows it to fall out of circuit if the bulb glass is broken. The release from circuit is assisted by the spring *b* that presses against the center contact *c* in the base of the bulb. The first tests made of the lamp indicated the desirability of certain changes in the design of the safety device, and after these were made the tests were repeated, with satisfactory results.



A. MACHINE FOR TESTING MINE-LAMP CORDS, WITH GUIDES REMOVED TO SHOW AMOUNT OF SLACK GIVEN CORDS AND METHOD OF MOUNTING.



B. CORD-TESTING MACHINE WITH GUIDES PLACED IN POSITION.



EDISON LAMP, APPROVAL NO. 10.

The maximum current that could be obtained by short-circuiting the lamp battery was determined by the standard test to be less than 30 amperes, which is well below the maximum test requirement for batteries of this type.

TENDENCY TO UNEXPECTED EXTINCTION.

As first submitted for test the lamp had several unnecessary contacts and connections between the battery and the bulb, and the methods of making connections were not altogether satisfactory. These contacts were the only probable source of unexpected extinction, for the battery had sufficient capacity and was reliable in action, and it had been proven by the standard tests that the safety devices were not so delicate as to operate prematurely. At the bureau's request the manufacturer eliminated some of the contacts and improved the character of some of the connections, thus eliminating the tendency to unexpected extinction.

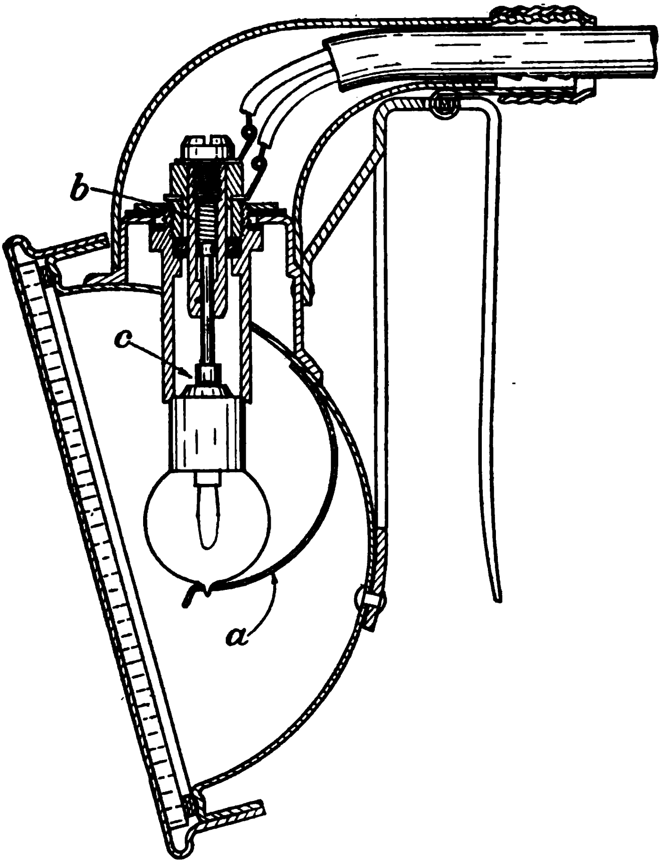


FIGURE 1.—Sectional view of headpiece of Edison lamp, showing principle of safety devices.

SPILLING OF ELECTROLYTE.

The batteries passed the standard tests without spilling a drop of electrolyte.

LIGHT-PRODUCING CAPACITY.

AMOUNT OF LIGHT.

Three different bulbs having different characteristics were approved for use with this lamp. The amount of light that each produced is shown in the table following.

Amount of light produced by the Edison mine lamp, approval No. 10, using various bulbs approved for use with this lamp.

	Bulb symbol.		
	26-V.[a]	Edison BM-10.[b]	National BM-10.[c]
Average candlepower after burning 1 hour	1.19	1.04	1.21
Average candlepower after burning 12 hours	0.83	0.59	0.71
Total lumens after burning 1 hour	4.32	3.76	4.39
Total lumens after burning 12 hours	3.01	2.13	2.58

[a] Made by the Independent Lamp & Wire Co., New York, N. Y.
[b] Made by the Edison Lamp Works of the General Electric Co., Harrison, N. J.
[c] Made by the National Lamp Works of the General Electric Co., Cleveland, Ohio.

DISTRIBUTION OF LIGHT.

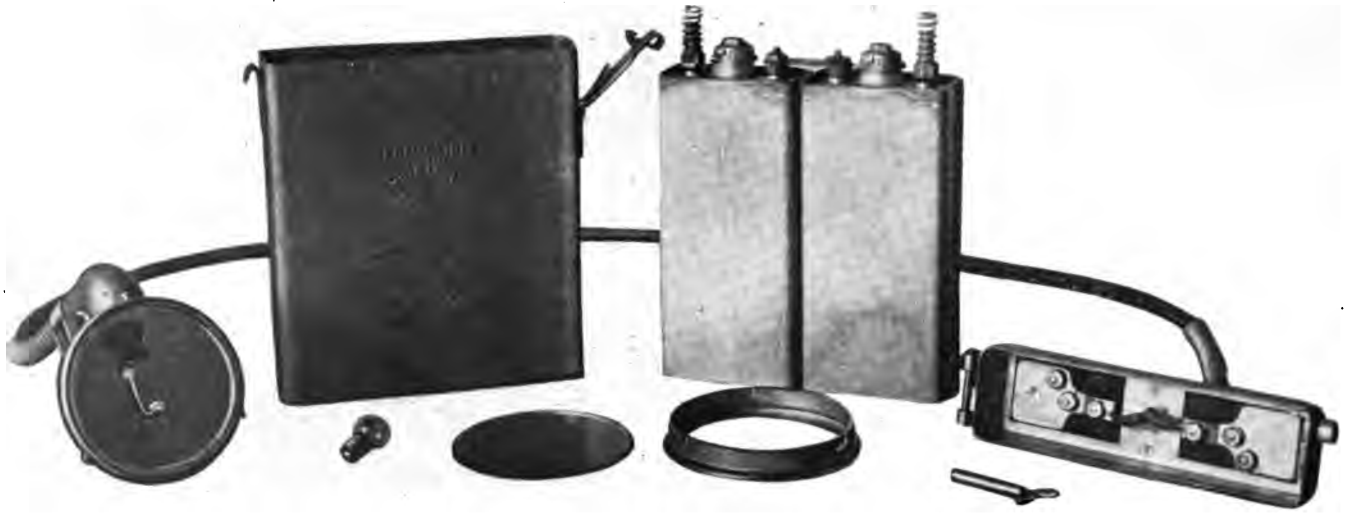
The distribution of light from the headpiece first submitted with the Edison lamp was not smooth and uniform and the angle of the light stream was much too narrow. The manufacturers, therefore, completely remodeled the headpiece. The distribution from the remodeled headpiece was measured in the standard way and was found to meet the test requirements so nearly that it could be approved. The reflector is covered with the so-called "cleanezy" finish and is not detachable.

TIME OF BURNING.

It was determined by the standard test that the lamp would burn on the average about 16 hours on one charge of battery before the average candlepower of the light stream fell below 0.4 (the minimum test requirement).

BULB CHARACTERISTICS.

Eight lots of different bulbs, consisting of 1,000 bulbs each, were submitted for test in connection with this lamp. Three bulbs were approved—the symbol 26–V bulbs, made by the Independent Lamp & Wire Co.; the symbol BM–10 bulbs, made by the Edison Lamp



PRINCIPAL PARTS OF EDISON LAMP.

Works of the General Electric Co.; and the symbol BM-10 bulbs made by the National Lamp Works of the General Electric Co. Each lot of these bulbs burned in the standard test for more than 200 hours, and if the bulbs had been burned to extinction or until their candlepower had fallen below the test requirements the average life would doubtless have been much longer than 200 hours.

The characteristics of the various lots of bulbs approved by the bureau for use with this lamp are given in the table following:

Characteristics of bulbs approved for use with the Edison mine lamp, approval No. 10.

	Bulb symbol.		
	26-V.	Edison BM-10.	National BM-10.
Percentage of bulbs lasting less than 150 hours [a]	2.0	1.0	1.0
Broadside candlepower [b] after burning 1 hour [c]	0.821	0.668	0.805
Current consumption after burning 1 hour (amperes)[c]	0.397	0.384	0.375
Percentage of bulbs that were more than 30 per cent below the average candlepower	0.0	0.0	0.0
Percentage of bulbs that consumed 6 per cent more current than the average	2.0	0.0	0.0

[a] Minimum test requirement for bulbs used with alkaline-battery lamps.

[b] Candlepower measured with the filament loop broadside to the photometer screen.

[c] These measurements were made at the average voltage manifested by the lamp battery after it had been discharged for 1 hour.

The bulbs finally approved were much superior to the bulbs first submitted for test, and marked a distinct advance in the design and methods of manufacture of miniature bulbs for mine-lamp service.

LIFE OF BATTERIES.

The manufacturers of the Edison lamp guarantee their batteries for two years; therefore it is manifest that the bureau could not make very searching tests with respect to the life of the lamp in the comparatively brief time that is usually required to make a complete investigation of a portable electric mine lamp.

The batteries gave the best of service during nearly 600 hours of actual discharge covering a period of more than seven months. Reports from the field covering more than a year of actual service of these batteries indicate that the bureau's approval of battery life was justified.

LIFE OF CORDS.

Considerable trouble was experienced in obtaining a satisfactory cord for the Edison lamp, and six different types were tested before a cord of acceptable durability was found. Part of the cord trouble was caused by weak armor, which the manufacturer has replaced with armor of a more suitable design.

MECHANICAL AND ELECTRICAL CHARACTERISTICS.

The mechanical strength of the lamp was tested by dropping the headpiece 15 times on a concrete floor from a height of 6 feet. The tests indicated that the construction of the headpiece was not sufficiently substantial, and it was reconstructed of heavier material.

The battery and the battery casing were dropped 10 times upon a wooden floor from a height of 3 feet. The equipment passed this test satisfactorily. In subsequent tests of other lamps this test was modified so as to average the performance of several batteries, but the severity of the test was not diminished.

During the tests and since the formal approval of the lamp numerous changes in construction were made by the manufacturers at the suggestion of the bureau engineers and as the result of experience under service conditions in mines. These changes have tended to increase durability, reliability, and the convenience of operation and maintenance.

The construction of the headpiece has been made much more substantial, the strength of the battery casing has been increased, an efficient means has been developed for clamping the cord in the headpiece and battery casing, and the cord armor has been made more substantial. A more convenient lock has been designed and the design of the vent plugs has been changed to make them more reliable and much easier to manipulate.

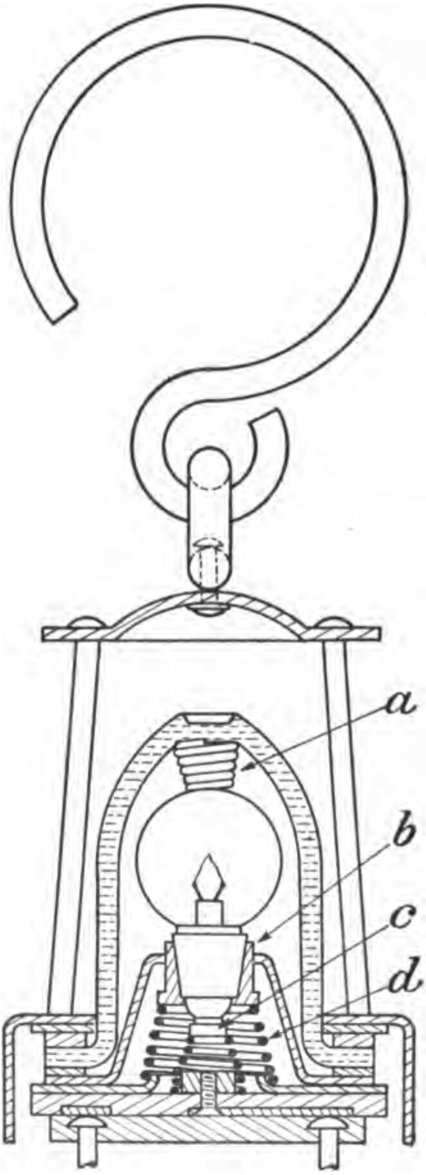


FIGURE 2.—Sectional view of upper part of Concordia hand lamp, showing principle of safety devices.

As the lamp is now constructed it is durable and reliable and has given satisfactory service in many mines where it has been in use for periods ranging from a few months to nearly a year and a half.

The lamp was approved with recommendations by the bureau for (1) improvement in design of the filling plug and (2) improvement in the gasket that cushions the outer glass of the headpiece. The bureau's recommendations in respect to the filling plug have been adopted.

MANLITE LAMP APPROVAL WITHDRAWN.

Approval No. 11, which was granted to the Manlite lamp made by the Mannesmann Light Company of America, was rescinded for cause on April 23, 1917.

CONCORDIA HAND LAMP, APPROVAL NO. 12.

The first lamp submitted by the Concordia Safety Lamp Co., of Pittsburgh, Pa., is a hand lamp operated by a single cell of lead battery. The lamp weighs about 5½ pounds, and has a capacity of about 73 lumen-hours for a 12-hour shift. It is provided with a snap lock operated by an electromagnet. Detailed specifications are given in Table 1 (p. 44). The assembled equipment is shown in Plate X, *A*, and its component parts in Plate XI.

The Concordia hand lamp was tested in accordance with the requirements of Schedule 6A and after several changes in construction had been made was approved on July 26, 1915, for safety and for practicability and efficiency in general service (approval No. 12).

GAS-IGNITION TESTS.

The device that is designed to prevent gas ignition is shown in figure 2. A spiral spring *a* mounted between the bulb and the glass dome that surrounds it presses the bulb firmly against its contacts *b* and *c*, at the same time compressing the spring *d*. If either the bulb or the glass dome is broken the spring *d* pushes the bulb upward and away from the center contact *c*, thus interrupting the circuit. The effectiveness of this device was tested in the standard manner with satisfactory results.

The maximum current that could be obtained by short-circuiting the Concordia hand lamp battery was determined by the standard test to be about 61.6 amperes, which is a safe value for batteries of this type.

TENDENCY TO UNEXPECTED EXTINCTION.

The tendency to unexpected extinction was tested in the standard way. The lamp proved satisfactory.

SPILLING OF ELECTROLYTE.

The batteries were tested in the standard manner except that, because they were used with hand lamps, they were not given the "shoveling test," but instead were placed on their sides during the interval usually devoted to shoveling. The batteries passed these tests without spilling a drop of electrolyte.

LIGHT-PRODUCING CAPACITY.

AMOUNT OF LIGHT.

Only one type of bulb (symbol Osram 08510) was tested in connection with the Concordia hand lamp. The amount of light produced is shown in the table following:

Amount of light produced by the Concordia hand lamp, approval No. 12.

Average candlepower after burning 1 hour	0.74
Average candlepower after burning 12 hours	0.57
Total lumens after burning 1 hour	6.92
Total lumens after burning 12 hours	5.36

DISTRIBUTION OF LIGHT.

The Concordia hand lamp does not use a reflector. The Bureau of Mines does not investigate the distribution of light from lamps not provided with a reflector.

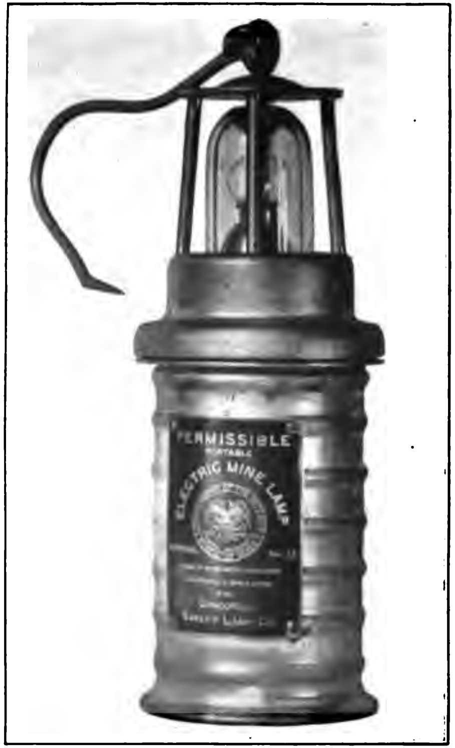
TIME OF BURNING.

The standard tests showed that the lamp would burn, on an average, about 17 hours on one charge of battery before the average candlepower of the light stream fell below 0.4 (the minimum test requirement of the bureau).

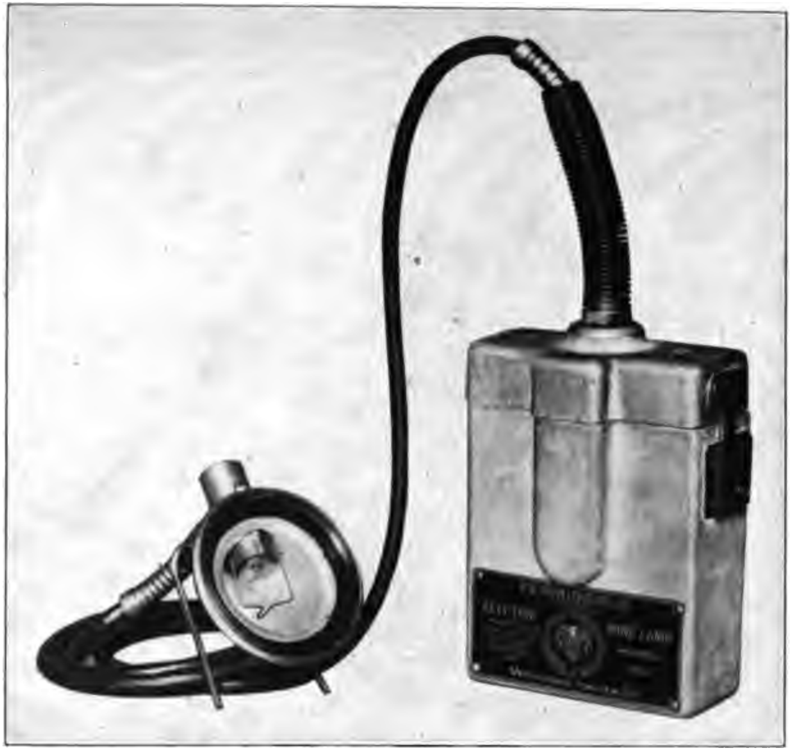
BULB CHARACTERISTICS.

The first lot of 1,000 bulbs submitted in connection with this lamp met the test requirements. These bulbs were identified by the symbol Osram 08510 and were made by the Auergesellschaft, of Berlin, Germany. This lot of bulbs burned in the standard test for more than 300 hours (the minimum test requirement for bulbs used with lead battery), and undoubtedly would have averaged much more than this figure if the bulbs had been burned to extinction or until their candlepower had fallen below the test requirement.

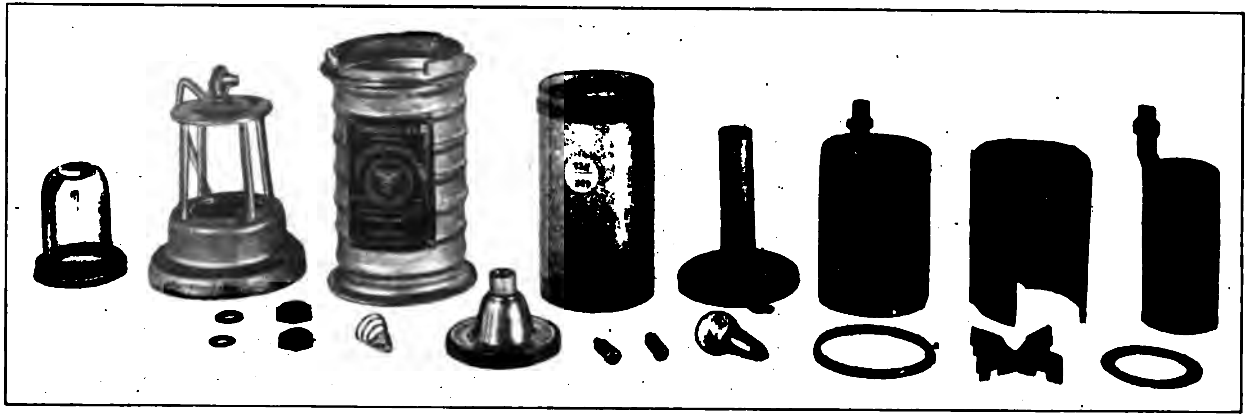
The table following gives the characteristics of the Osram 08510 bulbs:



A. CONCORDIA HAND LAMP, APPROVAL NO. 12.



B. WICO LAMP, APPROVAL NO. 14.



PRINCIPAL PARTS OF CONCORDIA HAND LAMP.

Characteristics of Osram 08510 bulbs for Concordia hand lamp.

Percentage of bulbs lasting less than 250 hours[a]	0.0
Broadside candlepower[b] after burning one hour[c]	0.996
Current consumption after burning one hour,[c] amperes	0.862
Percentage of bulbs that consumed 6 per cent more current than the average	0.0
Percentage of bulbs that were more than 30 per cent below the average candlepower	0.0

LIFE OF BATTERIES.

The Concordia hand-lamp battery as first submitted for test did not meet requirements and the manufacturers remodeled it in accordance with the suggestions of the Bureau of Mines engineers. After the battery had been remodelled it was retested and those tested gave excellent service during the comparatively brief time they were on test and gave no evidence of early deterioration.

MECHANICAL AND ELECTRICAL CHARACTERISTICS.

The mechanical strength of the Concordia hand lamp was tested in the standard way. The batteries failed to pass the first series of tests on account of faulty construction, but a set of properly constructed batteries passed the subsequent tests.

Few changes in construction were made in this lamp, as it came to the bureau thoroughly developed. The principal change was the introduction of a hard-rubber separator between the positive and negative plates of the battery and the substitution of a molded compound base for the mounting of the contacts in the top half of the lamp.

The lamp in the form that was approved is durable and reliable. It was approved with recommendations by the Bureau of Mines for (1) the exercise of greater care in the sealing of battery jars, and (2) the use of a tougher material for the base for mounting the contacts in the upper part. The writer has not been advised that these recommendations have been adopted.

GENERAL ELECTRIC LAMP, APPROVALS NOS. 13 AND 13A.

The lamp submitted by the General Electric Co., of Schenectady, N. Y., is a cap lamp operated by a single cell of lead battery. Two forms of this type of lamp were submitted. The Form C lamp had a sheet steel casing and the Form D lamp has a cast aluminum casing; in all other details they were identical. The Form C lamp weighs 4.41 pounds and the Form D lamp weighs 4.74 pounds. Both have a capacity of 36.6 lumen-hours for a 12-hour shift, and each is provided with a snap lock opened with a key. Detailed specifications of the lamps are given in Table 1. The assembled equipment and the component parts of the Form C lamp are shown in Plates XII, *A*, and XIII, and those of the Form D lamp are shown in Plates XII, *B*, and XIV.

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[a] The minimum test requirement for bulbs used with a lead-battery lamp.

[b] Candlepower measured with the filament loop broadside to the photometer screen.

[c] These measurements were made at the average voltage manifested by the lamp battery after it had been discharging for one hour.

Each of the two forms of this lamp was tested, and after several changes in construction had been made both forms were approved by the Bureau of Mines on July 11, 1916, for safety and for practicability and efficiency in general service (approvals Nos. 13 and 13A).

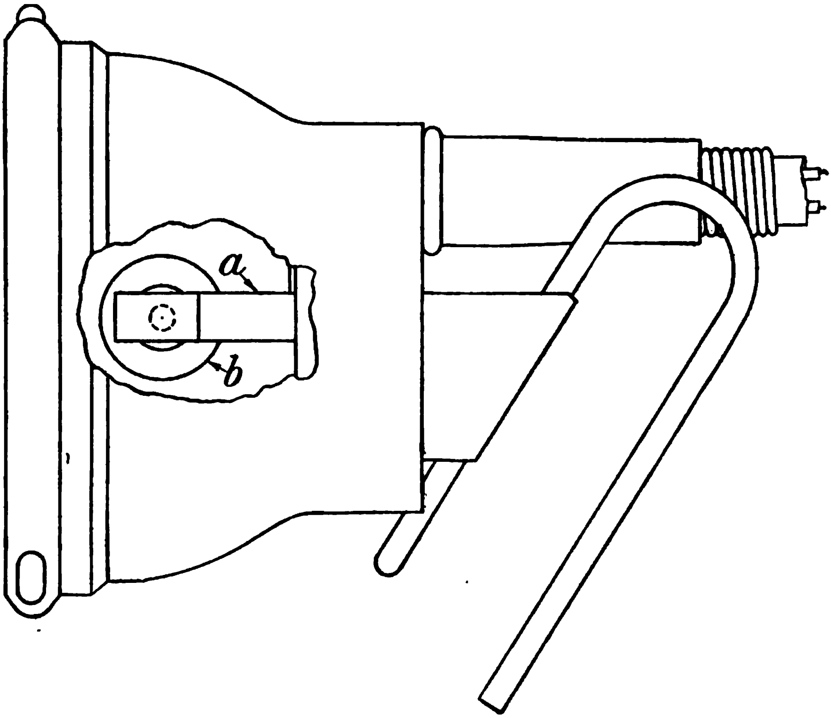
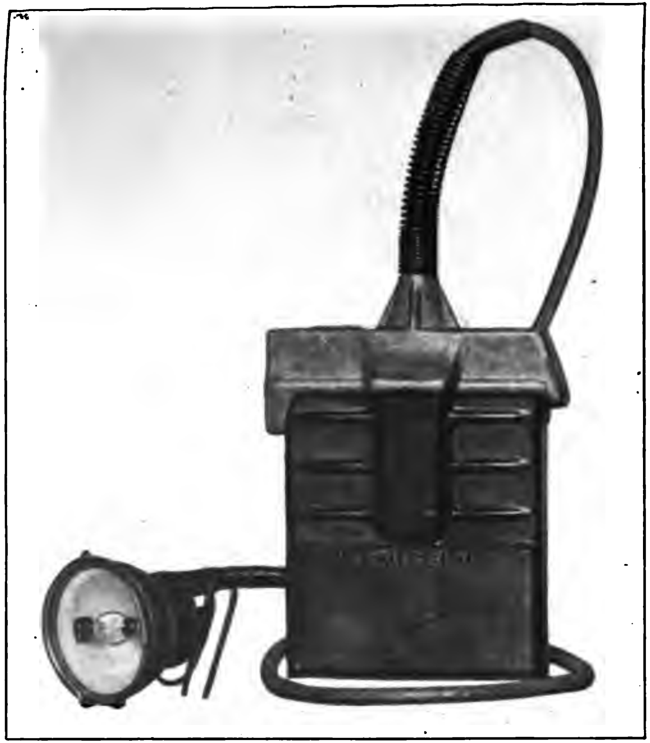


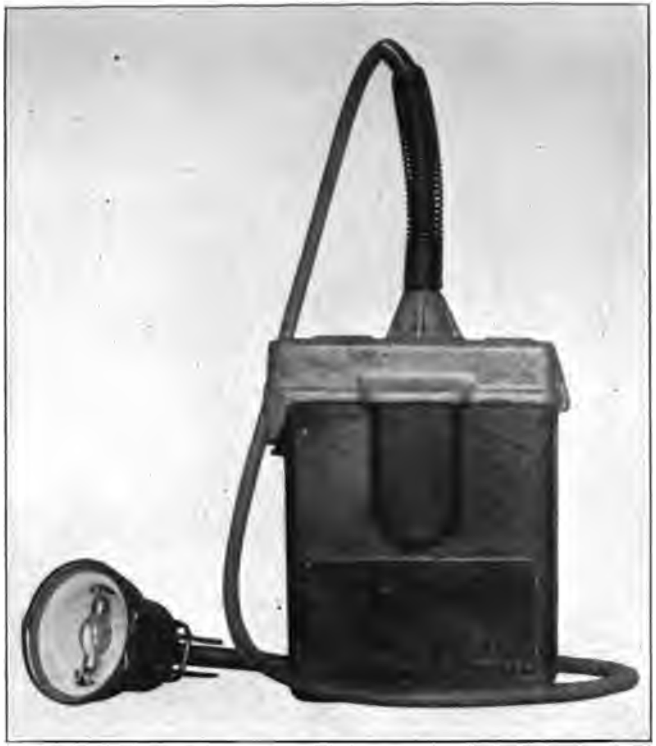
FIGURE 3.—Headpiece of General Electric lamp, side view.

GAS-IGNITION TESTS.

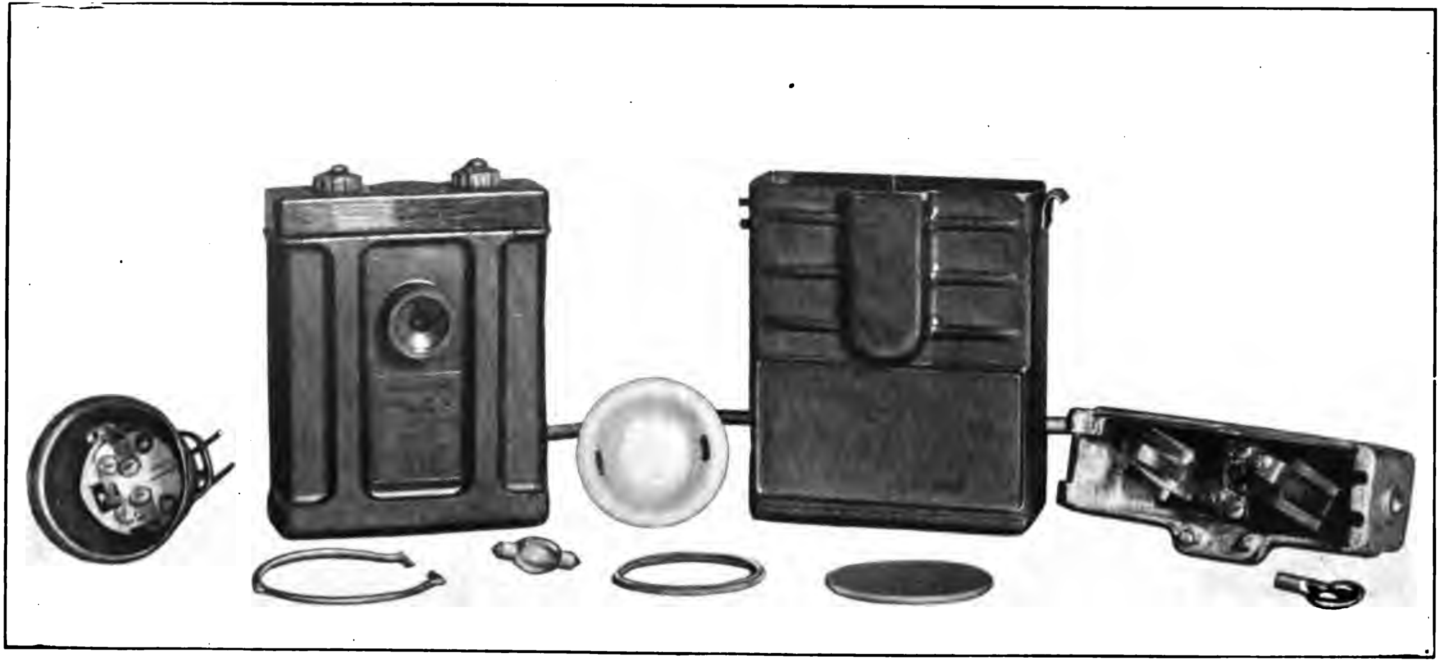
The device that is designed to prevent gas ignition is shown in figures 3 and 4. The bulb *b* has a contact at each end and is held in position between the two springs *aa*. When the bulb is broken it falls out of contact with these springs if the force of the blow has not already thrown the bulb out of contact before breaking it. The circuit is thus interrupted and the lamp extinguished.



A. GENERAL ELECTRIC LAMP WITH STEEL CASING, APPROVAL NO. 13.



B. GENERAL ELECTRIC LAMP WITH CAST ALUMINUM CASING, APPROVAL NO. 13A.



PRINCIPAL PARTS OF GENERAL ELECTRIC LAMP WITH STEEL CASING.

The effectiveness of this device was tested in the standard manner on three occasions. The first two series of tests developed the necessity for changes in the bulb and contact springs, and after these were made the tests were repeated with satisfactory results.

The maximum current that could be obtained by short-circuiting the battery provided with this lamp was determined by the standard tests to be about 73.8 amperes, which is a safe value for batteries of this type.

TENDENCY TO UNEXPECTED EXTINCTION.

Tests of the tendency to unexpected extinction, made in the standard way, showed that the bulb was thrown out of contact with the contact springs rather too easily, and the manufacturers were advised to remedy this by making the springs stiffer.

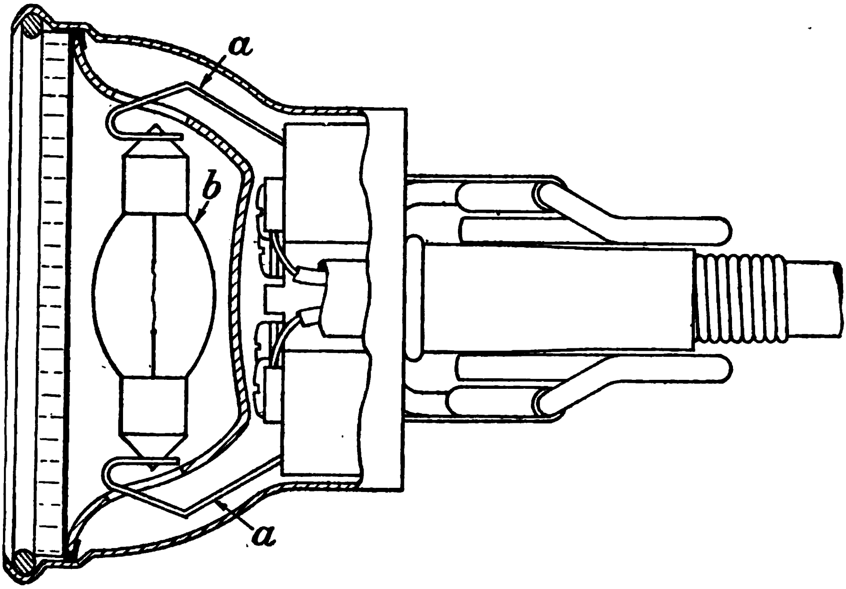


FIGURE 4.—Cross section of headpiece of General Electric lamp. Section at right angles to view shown in figure 3.

SPILLING OF ELECTROLYTE.

The batteries passed the standard tests without spilling a drop of electrolyte. Several different designs of spilling plugs were tested before a satisfactory one was developed.

LIGHT-PRODUCING CAPACITY.

AMOUNT OF LIGHT.

Three lots of 1,000 bulbs each were submitted in connection with the lamp before a satisfactory bulb (symbol BM-13) was obtained. The amount of light produced is shown in the table following.

Amount of light produced by the General Electric lamp, approvals Nos. 13 and 13A.

Average candlepower after burning 1 hour	1.0
Average candlepower after burning 12 hours	0.68
Total lumens after burning 1 hour	3.63
Total lumens after burning 12 hours	2.47

DISTRIBUTION OF LIGHT.

The distribution of light from the headpiece of this lamp, using a detachable porcelain enamel reflector, was considered satisfactory.

TIME OF BURNING.

It was determined by the standard tests that the lamp would burn on an average of about 14½ hours on one charge of battery before the battery voltage fell below the minimum value recommended by the manufacturers.

BULB CHARACTERISTICS.

The bulbs approved by the Bureau of Mines for use with this lamp were made by the Edison Lamp Works of the General Electric Co., at Harrison, N. J., and are identified by the symbol BM-13. These bulbs burned in the standard tests for more than 300 hours (the minimum test requirement for bulbs used with a lead battery) and would undoubtedly have averaged much more than this figure if the bulbs had been burned to extinction, or until their candlepower had fallen below the test requirements.

The table following gives the characteristics of the BM-13 bulbs:

Characteristics of the BM-13 bulbs for the General Electric lamp.

Percentage of bulbs lasting less than 250 hours[a]	5.0
Broadside candlepower[b] after burning one hour[c]	0.767
Current consumption after burning one hour,[c] amperes	0.607
Percentage of bulbs that consumed 6 per cent more current than the average	0.0
Percentage of bulbs that were more than 30 per cent below the average candlepower	0.0

LIFE OF BATTERIES.

The batteries gave excellent service during the comparatively brief time that they were on test at the laboratory and showed no evidence of early deterioration.

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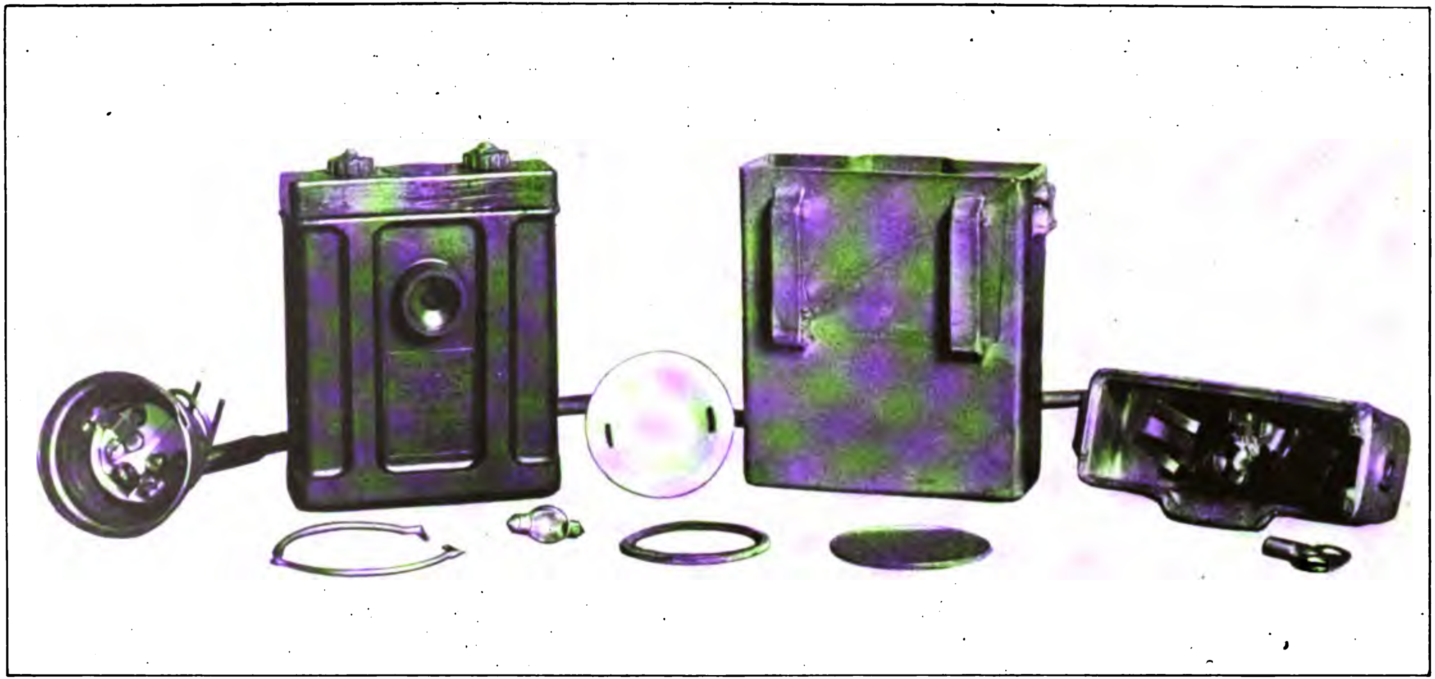
[a] The minimum test requirement for bulbs used with a lead-battery lamp.

[b] Candlepower measured with the filament loop broadside to the photometer screen.

[c] These measurements were made at the average voltage manifested by the lamp battery after it had been discharging for one hour.



PRINCIPAL PARTS OF GENERAL ELECTRIC LAMP WITH CAST ALUMINUM CASING.

LIFE OF CORDS.

Four different types of cord were tested before a cord of acceptable durability was found. This cord gave satisfactory results in the tests, but since then the bureau's researches have developed a cord that is much more durable and the bureau has recommended to the manufacturer that a cord of this type be substituted for the one originally approved with the lamp.

MECHANICAL AND ELECTRICAL CHARACTERISTICS.

The mechanical strength of both forms of the lamp was tested in the standard way and passed the tests. A number of changes in construction were made at the suggestion of the bureau engineers during the time that the lamp was under test.

These changes include the incorporation of a safety device, improvements in the characteristics of the bulbs and in the device to prevent spilling, introduction of cord clamps, and improvements in the distribution of light, in the character of cord and cord armor, and in the strength of the battery jar and the character and durability of the lock.

The lamp in both the forms that were approved is durable and reliable. They were approved with recommendations by the Bureau of Mines for (1) the substitution of a more durable cord, (2) introduction of stiffer contact springs for both battery and bulb, (3) improvement in the design and construction of the cord clamp, and (4) improvement in the character of the rubber gasket used to make the headpiece dust-tight. The aluminum-casing lamp was approved with a recommendation that the yoke over the hinge slot in the battery casing be strengthened. The manufacturer has taken these suggestions under consideration.

WICO LAMP, APPROVAL NO. 14.

The Wico lamp, submitted by the Witherbee Igniter Co., of Springfield, Mass., is a cap lamp operated by a single cell of lead battery. The lamp weighs about 4¼ pounds and has a capacity of about 35.2 lumen-hours for a 12-hour shift. It is provided with a snap lock opened with a key. A detailed description of the lamp is given in Table 1 (p. 44). The assembled equipment is shown in Plate X, B, and its component parts are shown in Plate XV.

The lamp was tested in accordance with the requirements of Schedule 6A and after numerous changes in construction had been made was approved by the Bureau of Mines on June 10, 1916, for safety and for practicability and efficiency in general service (approval No. 14).

GAS-IGNITION TESTS.

The device that is designed to prevent gas ignition is shown in figure 5. The lamp bulb *b* is held in its socket *c* by a wire stirrup *a* against the pressure of spring contacts *d* within the socket that act to eject the bulb if the stirrup be removed or the bulb broken. It was necessary to modify several times the device originally submitted before it was satisfactory.

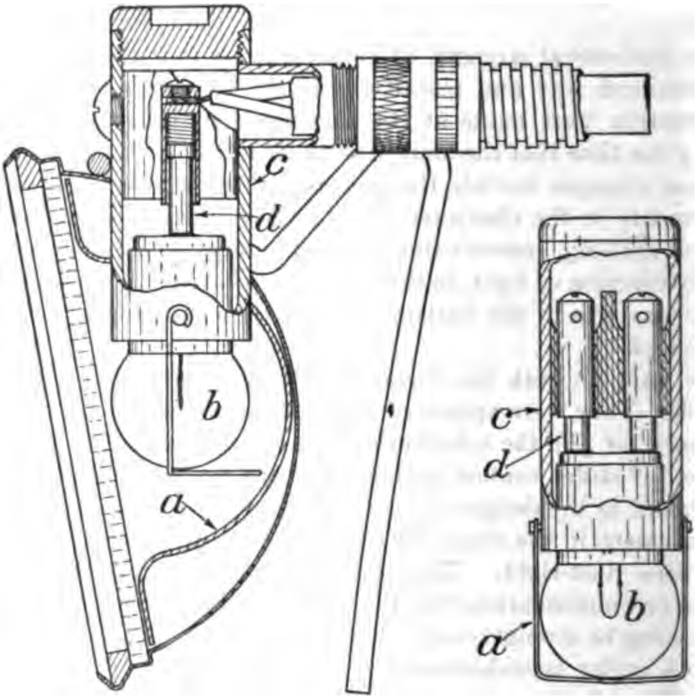


FIGURE 5.—Sectional view of headpiece of Wico lamp, showing principle of safety devices.

The maximum current that could be obtained by short-circuiting the lamp was determined by the standard test to be about 81 amperes, which is a safe value for batteries of this type.

TENDENCY TO UNEXPECTED EXTINCTION.

The tendency to unexpected extinction was tested in the standard way and the lamp was found to be satisfactory in this respect.

SPILLING OF ELECTROLYTE.

The batteries passed the standard test without spilling a drop of electrolyte.

LIGHT-PRODUCING CAPACITY.

AMOUNT OF LIGHT.

Only one type of bulb (symbol BM-14) was approved in connection with the Wico lamp. The amount of light produced is shown in the table following:

Amount of light produced by the Wico lamp, approval No. 14.

Average candlepower after burning 1 hour	0.93
Average candlepower after burning 12 hours	0.69
Total lumens after burning 1 hour	3.37
Total lumens after burning 12 hours	2.50

DISTRIBUTION OF LIGHT.

The distribution of light from the headpiece first submitted with the lamp was not sufficiently uniform to meet the test requirements. The manufacturers, therefore, experimented until a satisfactory reflector was obtained. This reflector is made from sheet aluminum treated with a solution of caustic soda and is detachable. The distribution of light from it was measured in the standard way and was found to meet the test requirements satisfactorily.

TIME OF BURNING.

It was determined by the standard tests that the lamp would burn on an average about 17 hours on one charge of battery before the battery voltage fell below the minimum value recommended by the manufacturers.

BULB CHARACTERISTICS.

Five lots of different bulbs (of 1,000 bulbs each) were submitted for test in connection with the lamp. One lot, made by the Edison Lamp Works of the General Electric Co., at Harrison, N. J., was approved and the symbol BM-14 was assigned to these bulbs. This lot of bulbs burned in the standard tests for more than 300 hours (the minimum test requirement for bulbs used with lead battery) and undoubtedly would have averaged more than that figure if the bulbs had been burned to extinction or until their candlepower had fallen below the test requirements. The table following gives the characteristics of the BM-14 bulbs.

Characteristics of BM-1¼ bulbs made for the Wico lamp.

Percentage of bulbs lasting less than 250 hours,[a]	5.0
Broadside candlepower [b] after burning 1 hour [c]	0.580
Current consumption after burning 1 hour,[c] amperes	0.523
Percentage of bulbs that consumed 6 per cent more current than the average	0.0
Percentage of bulbs that were more than 30 per cent below the average candle power	0.0

LIFE OF BATTERIES.

The Wico lamp batteries gave the best of service during the comparatively brief time that they were on test and gave no evidence whatever of early deterioration.

LIFE OF CORDS.

The cord submitted with the lamp gave good results, but before the tests of the lamp were completed cords had been developed that gave much better results, and the bureau engineers recommended to the manufacturer that one of several cords be substituted for the one originally submitted with the lamp.

MECHANICAL AND ELECTRICAL CHARACTERISTICS.

The mechanical strength of the lamp, as determined by the standard tests, proved satisfactory. During the tests a number of changes in construction were made by the manufacturers at the bureau's suggestion, as follows:

A safety device was added to the headpiece.

The construction of the headpiece was simplified and made dust tight with an improved ring for retaining the outer glass.

The character of the cord armor and the location of the cord clamp was improved.

The design and construction of the battery contact springs and the general arrangement of parts in the cover of the battery casing were improved.

A self-contained snap lock was substituted for the detachable padlock for locking the battery casing.

The efficiency of the reflector and the distribution of light from it were improved.

The lamp in the form approved is durable and reliable, and was approved with a recommendation by the Bureau of Mines that the light-producing capacity of the bulbs be increased if this modification could be made without shortening the life of the bulbs below the bureau's requirements. The bureau also recommended that a more durable cord be adopted.

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[a] The minimum test requirement for bulbs used with a lead-battery lamp.

[b] Candlepower measured with the filament loop broadside to the photometer screen.

[c] These measurements were made at the average voltage manifested by the lamp battery after it had been discharging for one hour.

CONCORDIA CAP LAMP, APPROVAL NO. 15.

The cap lamp submitted by the Concordia Safety Lamp Co., of Pittsburgh, Pa., is operated by a single cell of lead battery. The lamp weighs 4.15 pounds, and has a capacity of 38.4 lumen-hours for a 12-hour shift. It is provided with a snap lock operated by an electromagnet. Detailed specifications of the lamp are given in Table 1. The assembled equipment is shown in Plate XVI, *A*, and its component parts are shown in Plate XVII, *A*.

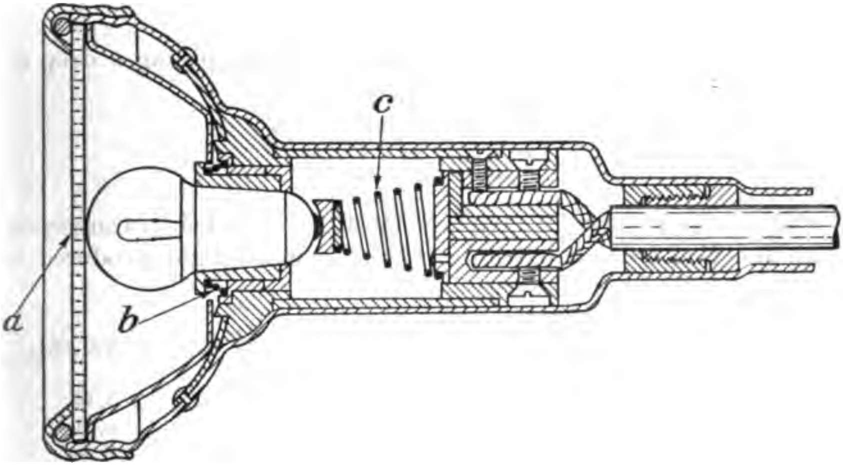


FIGURE 6.—Sectional view of headpiece of Concordia cap lamp, showing principle of safety devices.

The Concordia cap lamp was tested in accordance with the requirements of Schedule 6A and after several changes in construction had been made was approved on June 17, 1916, for safety and for practicability and efficiency in general service (approval No. 15).

GAS-IGNITION TESTS.

The device that is designed to prevent gas ignition is shown in figure 6. The outer glass *a* holds the bulb in position against the force of the spiral spring *b*, which forms one contact of the bulb, the other contact being formed by the spiral spring *c*. If either the bulb or the outer glass is broken the spring *b* pushes the bulb outward and away from the center contact *c*, the spring of which has less travel than the spring *b*. The circuit is thus interrupted and the lamp extinguished.

The effectiveness of this device was tested in the standard manner three times. The first two series of tests developed the fact that the action of the safety devices was not sufficiently protected from outside interference. The manufacturers finally reconstructed the headpiece so that it passed the tests satisfactorily.

The maximum current that could be obtained by short-circuiting the Concordia cap-lamp battery was determined by the standard test to be about 110 amperes, which is a safe value for batteries of this type.

TENDENCY TO UNEXPECTED EXTINCTION.

The tendency to unexpected extinction was tested in the standard way; the lamp proved satisfactory in this respect.

SPILLING OF ELECTROLYTE.

The batteries passed the standard test without spilling a drop of electrolyte.

LIGHT-PRODUCING CAPACITY.

AMOUNT OF LIGHT.

Only one type of bulb (symbol BM–15) was tested in connection with the Concordia cap lamp. The amount of light produced is shown in the table following:

Amount of light produced by the Concordia cap lamp, approval No. 15.

Average candlepower after burning 1 hour	1.05
Average candlepower after burning 12 hours	0.72
Total lumens after burning 1 hour	3.80
Total lumens after burning 12 hours	2.60

DISTRIBUTION OF LIGHT.

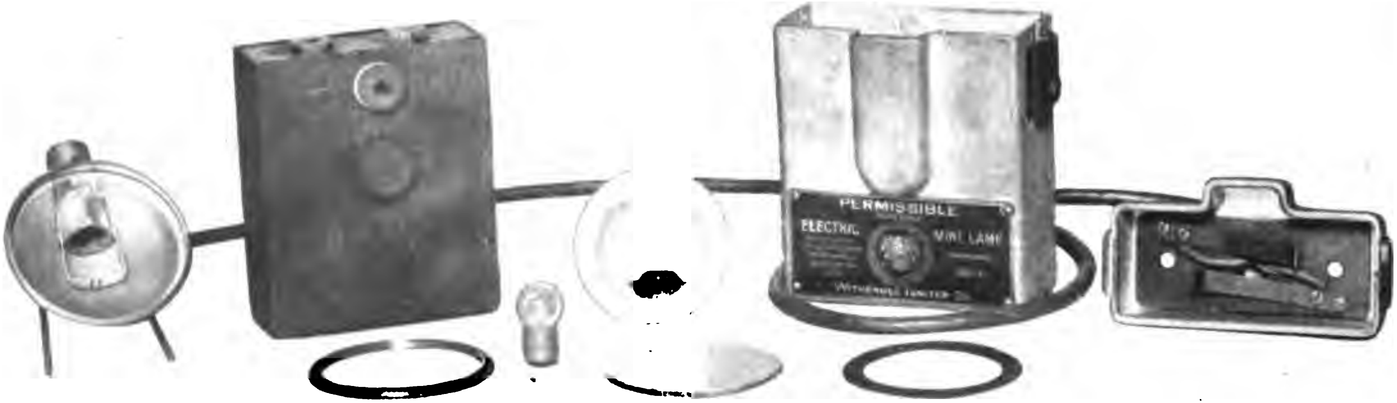
The distribution of light from the Concordia cap lamp more than met the test requirements. The reflector of the headpiece is detachable and is covered with "cleanezy" finish.

TIME OF BURNING.

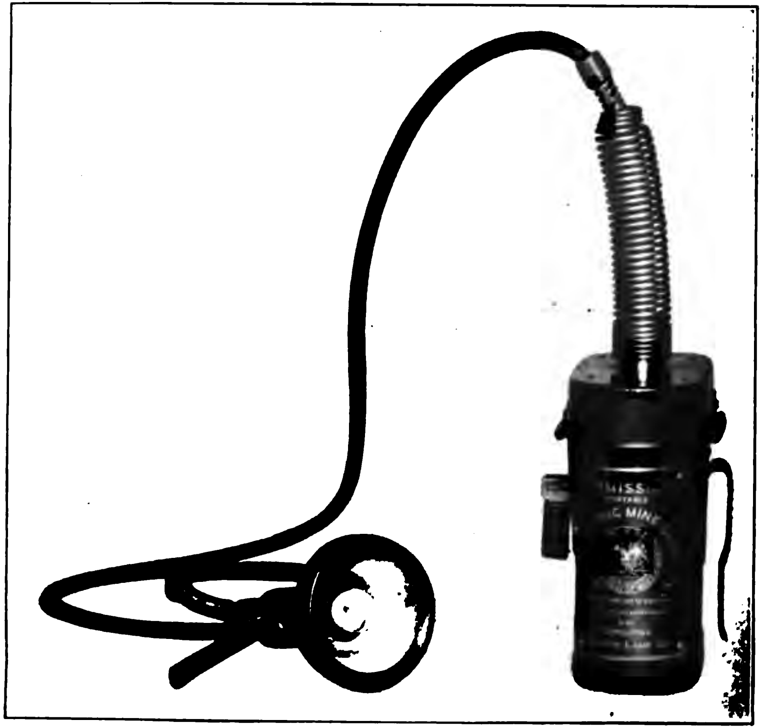
It was determined by the standard tests that the lamp would burn on the average about 15½ hours on one charge of battery before the battery voltage fell below the minimum value recommended by the manufacturers.

BULB CHARACTERISTICS.

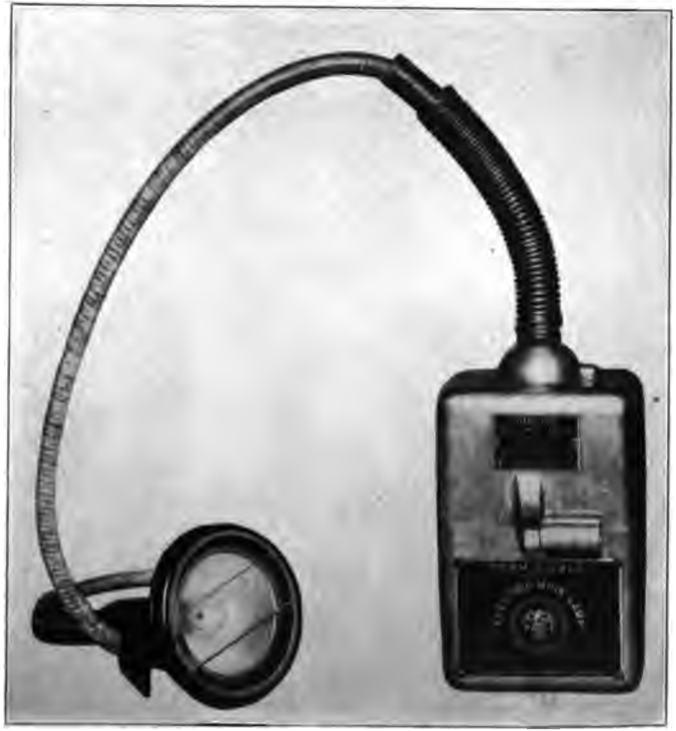
The first lot of 1,000 bulbs submitted in connection with the lamp met the test requirements. These bulbs were identified by the symbol



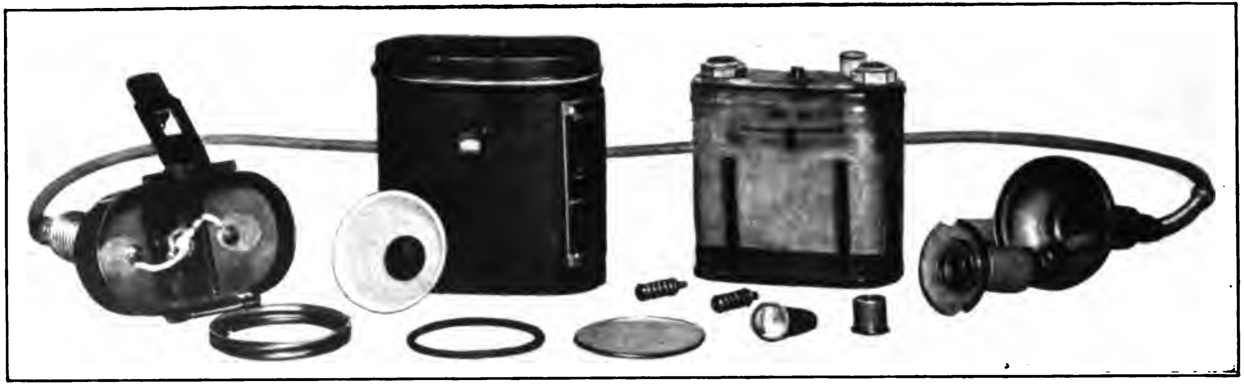
PRINCIPAL PARTS OF WICO LAMP, APPROVAL NO. 14.



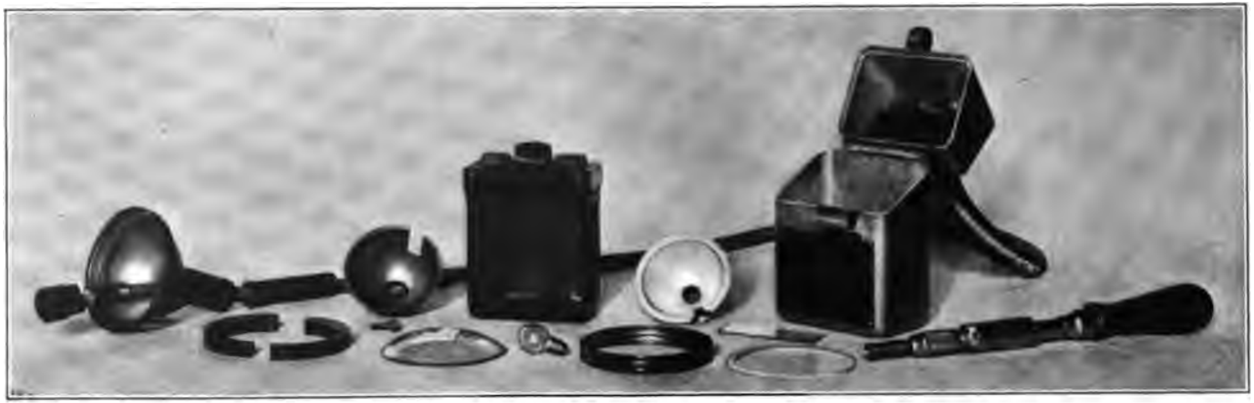
A. CONCORDIA CAP LAMP, APPROVAL NO. 15.



B. PIONEER LAMP, APPROVAL NO. 16.



A. PRINCIPAL PARTS OF CONCORDIA CAP LAMP.



B. PRINCIPAL PARTS OF PIONEER LAMP.

BM-15 and were made by the National Lamp Works of the General Electric Co., Cleveland, Ohio. This lot of bulbs burned in the standard tests for more than 300 hours (the minimum test requirement for bulbs used with lead battery) and undoubtedly would have averaged much more than this figure if the bulbs had been burned to extinction or until their candlepower had fallen below the test requirements. The table following gives the characteristics of the BM-15 bulbs:

Characteristics of BM-15 bulbs for the Concordia cap lamp.

Percentage of bulbs lasting less than 250 hours[a]	2.0
Broadside candlepower[b] after burning one hour[c]	0.716
Current consumption after burning one hour,[c] amperes	0.805
Percentage of bulbs that consumed 6 per cent more current than the average	1.0
Percentage of bulbs that were more than 30 per cent below the average candlepower	2.0

LIFE OF BATTERIES.

The Concordia cap lamp batteries gave excellent service during the comparatively brief time that they were on test and gave no evidence of early deterioration.

LIFE OF CORDS.

The first cord submitted with the lamp did not give satisfactory service in test and the manufacturer adopted as standard a cord that had given good service during the bureau's research to develop a satisfactory mine-lamp cord.

MECHANICAL AND ELECTRICAL CHARACTERISTICS.

The mechanical strength of the lamp passed the standard tests. A number of changes in construction were made in the lamp at the suggestion of the bureau during the time that it was under investigation. These changes included improvements in the design of the cord clamp in the battery casing and the headpiece, and in the design of cord armor; better protection of the moving parts of the safety device; thicker headpiece shell and thicker bulb glass; more reliable locking mechanism; simpler construction of the contacts mounted in the cover of the battery casing.

The lamp in the form that was approved is durable and reliable. It was approved with recommendations by the Bureau of Mines for

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[a] The minimum test requirement for bulbs used with a lead-battery lamp.

[b] Candlepower measured with the filament loop broadside to the photometer screen.

[c] These measurements were made at the average voltage manifested by the lamp battery after it had been discharging for one hour.

(1) improvement in the character of the lamp cord, (2) the use of a cast aluminum casing instead of a sheet-metal casing, and (3) the development of bulbs that will give more light and still meet the bureau's requirements in regard to life of bulbs.

The writer has not been advised that these recommendations have been adopted.

PIONEER LAMP, APPROVAL NO. 16.

The Pioneer lamp, which was submitted by the Hirsch Electric Mine Lamp Co., of Philadelphia, Pa., is a cap lamp operated by a single cell of lead battery. The lamp weighs 4¼ pounds, and has a capacity of about 41.6 lumen-hours for a 12-hour shift. It is provided with a screw-bolt lock operated by a spring wrench. Detailed specifications of the lamp are given in Table 1. The assembled equipment is shown in Plate XVI, *B*, and its component parts are shown in Plate XVII, *B*. The lamp was tested in accordance with the requirements of Schedule 6A, and after numerous changes in construction had been made was approved on July 21, 1916, for safety and for practicability and efficiency in general service (approval No. 16).

GAS-IGNITION TESTS.

The devices that are designed to prevent gas ignition are shown in figure 7. They consist of an open-circuiting device that protects the bulbs against blows from the front, and a short-circuiting device that protects the bulbs against blows from the side. The open-circuiting device is operated by breaking a slip of window glass, *a*, which is mounted directly across the inner surface of the outer glass, *b*. The breaking of this slip of glass releases a spring, *c*, that interrupts the electric circuit of the lamp. The short-circuiting of the bulb is accomplished as follows: The headpiece is made up of three concentric shells, *d*, *e*, and *f*, separated by a narrow space. The outer and inner shell are connected to the positive pole of the battery, and the intermediate shell is connected to the negative pole. These shells completely surround the bulb, and the theory of the safety device is that the bulb can not be broken without so jamming these shells together that they will short-circuit the bulb and thus extinguish the filament before it can ignite gas.

The effectiveness of these devices was tested in the standard way, and they were found to operate satisfactorily.

The maximum current that could be obtained by short-circuiting the Pioneer lamp battery was determined by the standard test to be 95.8 amperes, which is a safe value for batteries of this type.

TENDENCY TO UNEXPECTED EXTINCTION.

The tendency to unexpected extinction was tested in the standard way, and the lamp was found to be fairly satisfactory in this respect, although the bureau has recommended that the reliability of the lamp be increased by eliminating the circuit breaker from the headpiece and substituting an equally satisfactory safety device.

SPILLING OF ELECTROLYTE.

The batteries passed the standard tests without spilling a drop of electrolyte.

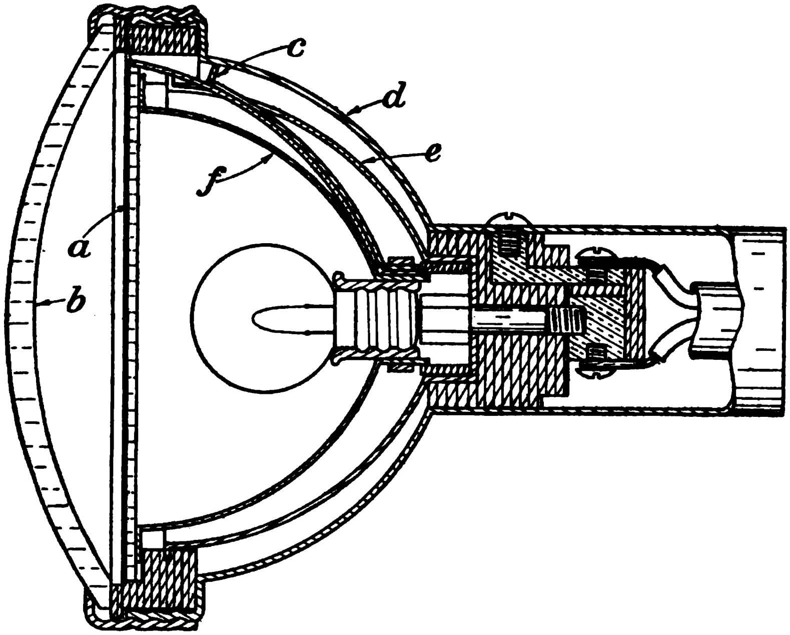

FIGURE 7.—Sectional view of headpiece of Pioneer lamp, showing principle of safety devices.

LIGHT-PRODUCING CAPACITY.

AMOUNT OF LIGHT.

Six lots of 1,000 bulbs each were submitted in connection with the Pioneer lamp before a satisfactory bulb (BM-16) was obtained. The amount of light produced is shown in the table following:

Amount of light produced by the Pioneer lamp, approval No. 16.

Average candlepower after burning 1 hour	1.13
Average candlepower after burning 12 hours	0.78
Total lumens after burning 1 hour	4.10
Total lumens after burning 12 hours	2.83

DISTRIBUTION OF LIGHT.

The distribution of light from the headpiece of the Pioneer lamp using a reflector of sheet aluminum treated with a solution of caustic soda met the test requirements.

TIME OF BURNING.

The standard tests showed that the lamp will burn on an average 15¼ hours on one charge of battery before the battery voltage falls below the minimum value recommended by the manufacturer.

BULB CHARACTERISTICS.

The bulbs approved by the Bureau of Mines for use with this lamp were made by the Edison Lamp Works of the General Electric Co., at Harrison, N. J., and are identified by the symbol BM–16. These bulbs burned in the standard tests for more than 300 hours (the minimum test requirements for bulbs used with lead battery), and would undoubtedly have averaged much more than this figure if the bulbs had been burned to extinction or until their candlepower had fallen below the test requirements.

The table following gives the characteristics of the BM–16 bulbs:

Characteristics of the BM–16 bulbs for the Pioneer lamp.

Percentage of bulbs lasting less than 250 hours [a]	2.00
Broadside candlepower [b] after burning one hour [c]	0.752
Current consumption after burning one hour,[c] amperes	0.60
Percentage of bulbs that consumed 6 per cent more current than the average	0.0
Percentage of bulbs that were more than 30 per cent below the average candlepower	1.0

LIFE OF BATTERIES.

The Pioneer lamp batteries gave excellent service during the comparatively brief time that they were on test and showed no signs of early deterioration.

LIFE OF CORDS.

The Pioneer lamp uses the best cord that was developed during the research to improve the character of mine-lamp cords. This cord is of exceptional durability.

MECHANICAL AND ELECTRICAL CHARACTERISTICS.

The mechanical strength of the lamp passed the standard tests. Numerous changes in construction were made in the lamp at the suggestion of the bureau during the time that it was under investigation. Improvements were made in the character of the safety device, battery, and bulbs, in the mechanical strength of the casing and the battery jar, in the character of the cord and armor, and in the distribution of light.

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[a] The minimum test requirement for bulbs used with a lead-battery lamp.

[b] Candlepower measured with the filament loop broadside to the photometer screen.

[c] These measurements were made at the average voltage manifested by the lamp battery after it had been discharging for one hour.

The lamp that was approved is durable and sufficiently reliable. It was approved with a recommendation that the safety device be so redesigned that it will not require the use of the circuit breaker in its present form.

SUMMARY OF RESULTS OF TESTS.

The characteristics of the various approved electric miner's lamps are summarized in Table 1.

The approved lamps are considered safe, practicable, and efficient for general service in mines. All passed the safety tests satisfactorily and in general they are durable and well constructed. The batteries require intelligent care but not so much, or of such a highly specialized nature, as to be a burden to the mine operator. If the instructions issued by the lamp manufacturers are followed the batteries should render good service with a reasonable amount of attention and at a reasonable cost for labor and supplies.

All the lamps give more light than a safety lamp and distribute the light better. The lamps are reliable in operation and in general are easy to repair. The life of the batteries and the bulbs promises to be reasonably long, and the bulbs that have been approved are exceptionally uniform in their characteristics.

In short all of these lamps have all the qualifications set forth on pages 10 and 11 as essential for a safe and practical electric mine lamp.

Nearly all the lamps can be improved in some minor respects, and these improvements no doubt will be made as soon as the need becomes sufficiently manifest.

The trend of development will probably be toward producing a greater amount of light and refining the various parts of the equipment. Undoubtedly the lamps will be made simpler and still more reliable and efficient, although it is not probable that their weight will be materially reduced until there is some radical improvement in the design of storage-battery cells or of incandescent bulbs.

The rapidity with which mine operators have adopted the approved portable electric lamps has been most gratifying. The first lamp was approved in February, 1915, and in August, 1916, about 70,000 approved lamps were in use in the mines of this country, and more lamps were being installed at the rate of about 2,000 a week.

Safety in mines should be decidedly increased by this widespread substitution of the electric lamp for the flame lamp.

TABLE 1.—*Detailed information regarding the portable electric miner's lamps approved by the Bureau of Mines.*

Name of manufacturer	Edison Storage Battery Co.	Concordia Safety Lamp Co.	General Electric Co.	General Electric Co.	Witherbee Igniter Co.	Concordia Safety Lamp Co.	Hirsch Electric Mine Lamp Co.
Bureau of Mine's approval No.	10.	12.	13.	13 A.	14.	15.	16.
Type of lamp	Cap.	Hand.	Cap.	Cap.	Cap.	Cap.	Cap.
Average candlepower of light stream [a]	0.96 [b]	0.70.	0.84.	0.84.	0.81.	0.88.	0.96.
Capacity, average lumen-hours per 12-hour shift [c]	41.8 [b]	73.	36.6.	36.6.	35.2.	38.4.	41.6.
Weight of lamp, pounds	3.94.	5.54.	4.41.	4.74.	4.23.	4.15.	4.21.
Number of cells	2.	1.	1.	1.	1.	1.	1.
Kind of plates	Nickel-iron.	Lead.	Lead.	Lead.	Lead.	Lead.	Lead.
Kind of electrolyte	Alkaline.	Acid.	Acid.	Acid.	Acid.	Acid.	Acid.
Character of battery jar	Steel.	Celluloid.	Hard rubber.	Hard rubber.	Hard rubber.	Celluloid.	Hard rubber.
Character of case	Sheet steel.	Heavy steel.	Sheet steel.	Cast aluminum.	Cast aluminum.	Sheet steel.	Cast aluminum.
Type of lock	Snap lock operated by key.	Magnetic snap lock.	Snap lock operated by key.	Snap lock operated by key.	Snap lock operated by key.	Magnetic snap lock.	Screw-pin lock.
Character of cord armor	Flexible metallic tube, not reinforced.	None used.	Double spiral wire.	Double spiral wire.	Flexible metallic tube, reinforced with spiral wire.	Flexible metallic tube, reinforced with spiral wire.	Double spiral wire.
Kind of reflector	"Clean-ezy" finish.	do.	Porcelain.	Porcelain.	Treated aluminum.	"Clean-ezy" finish.	Treated aluminum.
Bulb symbol No.	BM-10.	Osram 08510.	BM-13.	BM-13.	BM-14.	BM-15.	BM-16.
Name of bulb manufacturer	National Lamp Works of General Electric Co.	Auergesellschaft, of Berlin, Germany.	Edison Lamp Works of General Electric Co.	Edison Lamp Works of General Electric Co.	Edison Lamp Works of General Electric Co.	National Lamp Works of General Electric Co.	Edison Lamp Works of General Electric Co.
Cord symbol No.	CD17.	No cord used.	CD21.	CD21.	CD1.	CD17.	CD31.
Average cord service, number of operations [d]	87,000.	do.	150,000.	150,000.	26,500.	102,000.	771,700.
Special features							

[a] Average of 100 bulbs that had been burned 150 hours, represents the average between candlepower of lamp after burning 1 hour and candlepower after burning 12 hours.

[b] Performance with best bulb available.

[c] This value has the same basis as "Average candlepower."

[d] Refers to the average number of operations given by the cord when tested in the cord-testing machine described on page 21.

PROCEDURE FOR BUREAU OF MINES TESTS OF PORTABLE ELECTRIC MINE LAMPS.

The procedure of the Bureau of Mines in making tests to establish a list of permissible portable electric lamps for use in mines, the fees charged, the character of the tests, and the conditions under which lamps will be tested by the bureau, as set forth in Schedule 6A, issued under date of February 3, 1915, are given below:

DEFINITION OF PERMISSIBLE.

The Bureau of Mines considers a portable electric lamp to be permissible for use in mines if all the details of the lamp's construction are the same in all respects as those of the lamp that passed the inspection and the tests for safety, practicability, and efficiency made by the bureau and hereinafter described.

CONDITIONS UNDER WHICH LAMPS WILL BE TESTED.

The conditions under which the Bureau of Mines will examine and test portable electric lamps to establish their permissibility are as follows:

1. The tests will be made at the experiment station of the Bureau of Mines at Pittsburgh, Pa.

2. Applications for tests shall be addressed to the Director, Bureau of Mines, Washington, D. C., and shall be accompanied by a complete description of the lamp to be tested and a full set of the drawings mentioned below.

A drawing or drawings clearly showing the size and general appearance of the lamp mounting.

A drawing or drawings clearly showing the character, size, and relative arrangement of the parts of the lamp mounting, and the principle of operation of the safety devices.

Any other drawings that may be necessary to identify the safety devices or to explain how they accomplish their purpose.

A copy of the description, a duplicate set of drawings, and one complete lamp shall be sent to the electrical engineer, Bureau of Mines, Fortieth and Butler Streets, Pittsburgh, Pa.

3. As soon as possible after the receipt of his application for test the lamp manufacturer will be notified of the date on which his lamps will be tested and the amount of material that it will be necessary for him to submit.

4. All material for test shall be delivered by the manufacturer to the electrical engineer, Bureau of Mines, Fortieth and Butler Streets, Pittsburgh, Pa., not less than one week prior to the date set for the test.

5. No lamp equipment will be tested unless it is in the completed form in which it is to be put on the market.

6. Lamps so constructed that they can be used both as cap lamps and as hand lamps must pass the tests for both cap lamps and hand lamps or they will not be approved for either class of service.

7. No one is to be present at these tests except the necessary Government officers, their assistants, and one representative of the manufacturer of the lamp to be tested, who shall be present in the capacity of an observer only.

The conduct of the tests shall be entirely in the hands of the bureau's engineer in charge of the investigation. While the tests are in progress the manufacturer's representative shall not make unsolicited suggestions or criticisms of the method of conducting the test.

8. The tests will be made in the order of the receipt of application for test, provided that the necessary lamp equipment is submitted at the proper time.

9. The details of the results of the tests shall be regarded as confidential by all present at the tests and shall not be made public in any way prior to their official publication by the Bureau of Mines.

REQUIREMENTS FOR APPROVAL.

The requirements that a portable electric-lamp equipment must have, to pass successfully the inspection and tests required by the bureau, are stated below:

1. The lamp equipment must comply with the following requirements for mechanical and electrical construction:

The construction of permissible portable electric-lamp equipment shall be especially durable. All parts shall be constructed of suitable material of the best quality and shall be assembled in a thorough workmanlike manner. Current-carrying parts shall be well insulated from parts of opposite polarity and from parts not intended to carry current.

The battery shall be inclosed in a locked or sealed box so constructed as to preclude the possibility of anyone meddling with the electrical contacts or making an electrical connection with them while the box cover is closed.

The leads connecting the battery with the headpiece shall be made up in a single cable efficiently insulated and provided, where it leaves the battery casing and enters the headpiece, with a reinforcement of flexible metallic tubing. The flexible metallic tubing will not be required if other equally durable means of reinforcement are provided.

It is recommended, but not required, that the headpiece be so designed that it can be sealed or locked. The battery terminals and leads connecting thereto and the gas vent of the battery shall be so designed and constructed as to prevent corrosion of the battery terminals or of the essential metallic parts mounted in the cover of the battery casing.

The following qualities will be considered in determining the excellence of the mechanical and electrical construction of lamps covered by these specifications:

Simplicity of design; mechanical strength of parts and fastenings; suitability of material used; design of moving and removable parts; design and construction of terminals and contacts for permanence and electrical efficiency; and ease of repair.

2. The lamp equipment must be provided with a safety device or devices as follows:

Permissible portable electric lamps shall be so designed and constructed that whenever the bulb of a completely assembled lamp equipment is broken the lamp filament shall at once and under all circumstances cease to glow at a temperature that will ignite explosive mixtures of mine gas and air.

The mounting of the bulb may be designed so that a blow sufficient to break the bulb will short-circuit it, open the electric circuit of the lamp, or otherwise insure that the filament will be wholly or practically extinguished. All safety devices with which the lamps are provided shall be so completely protected from injury or disturbance as to insure that the devices will always be in condition to perform their functions.

The design of the safety features shall be such that their action can not readily be hindered or prevented. The design of the safety devices shall be such that they will not act to extinguish the lamp unnecessarily.

3. The lamp equipment must be provided with a battery having a short-circuit current not in excess of the values here specified.

The bureau's engineers have made tests (reported in Technical Paper 47 of the bureau) which have satisfied them that mine gas can not be ignited by the sparks from portable electric-lamp equipments if the batteries used with such equipments are made so that their maximum short-circuit current can not exceed the following values: For batteries giving 2.5 volts or less, 125 amperes; for batteries giving more than 2.5 volts but not more than 4 volts, 85 amperes; for batteries giving more than 4 volts but not more than 6 volts, 65 amperes; for batteries giving more than 5 volts but not more than 6 volts, 45 amperes. Therefore, lamps whose short-circuit current does not exceed these values will be considered satisfactory in that respect.

4. The lamp equipment must meet the following requirements for time of burning, flux of light, intensity of light, and distribution of light:

All portable electric lamps offered for test under the provisions of this schedule shall produce for 12 consecutive hours on one charge of battery a light stream having an average intensity of light not less than four-tenths of a candlepower. The total flux of light produced by cap lamps shall not fall below 1½ lumens during the 12 hours, and the total flux of light produced by hand lamps shall not fall below 3 lumens during the 12 hours.

The distribution of light by lamps that use reflectors shall be determined both by observation and by photometric measurement. The lamps shall be placed so that the filaments are 20 inches away from a plane surface that is perpendicular to the axis of the light stream of the lamp. When so placed the lamp shall illuminate a circular area not less than 7 feet in diameter.[a] All observations and measurements of distribution shall be referred to this 7-foot circle regardless of how large an area the lamp may illuminate. As observed with the eye there shall be no "black spots" within the 7-foot circle nor any sharply contrasting areas of bright and faint illumination anywhere. As measured with a photometer the distribution of light diametrically across the circle shall fulfill the following requirements:

The curve of light distribution along the diameter of the circle shall be obtained by rotating the lamp, and thus obtaining the average distribution curve.

The average illumination in foot-candles on the best-illuminated one-tenth of the diameter shall be not more than three times the average illumination throughout the diameter, and for at least 40 per cent of the diameter the illumination shall be not less than the average.

5. The lamp equipment must be provided with lamp bulbs that meet the following requirements for variation in current consumption, variation in candlepower, and length of life:

The bulbs submitted for test shall be identified by the name of the manufacturer and by a number or symbol with reference to which approval will be granted.

The current consumption of at least 95 per cent of the bulbs tested shall not exceed by more than 6 per cent the average current consumption of all the bulbs examined.

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[a] This requirement will be met by lamps that have an angle of light stream of 130° or more.

The candlepower of at least 90 per cent of the bulbs tested shall not fall short of the average candlepower by more than 30 per cent.

The life of a lamp bulb will be considered as the number of hours that the bulb can be burned under normal conditions of voltage before it becomes so depreciated that when used with an average, standard, freshly charged equipment it fails to produce for 12 consecutive hours the flux and intensity of light specified in paragraph 4, page 6.

The average life of lamp bulbs shall be not less than 300 hours for acid storage batteries and not less than 200 hours for primary batteries and for alkaline storage batteries. Not more than 5 per cent of the bulbs examined shall give less than 250 hours' life with acid batteries nor less than 150 hours' life with primary batteries and alkaline batteries.

6. The lamp equipment must comply with the following requirements as to leakage of electrolyte:

Lamps shall be so designed and constructed that they will not spill nor leak electrolyte throughout an 8-hour test, during which they will be placed in any position or sequence of positions that in the opinion of the bureau's engineers will be most likely to prove whether or not the electrolyte can be spilled.

CHARACTER OF TESTS TO WHICH LAMPS WILL BE SUBJECTED.

DESIGN AND CONSTRUCTION.

The excellence of the mechanical and electrical features of the design and construction of the lamps will be carefully determined.

The following tests will also be made: Hand lamps and the headpieces of cap lamps will be dropped 10 times upon a concrete floor from a point 6 feet above it. As the result of these dropping tests there must be no breakage of the battery jar nor material distortion of the casing of the battery or of the shell of the headpiece. The engineers in charge of the investigation shall be the sole judges of whether or not material distortion occurs. The dropping tests of the headpiece must demonstrate that the safety devices will not operate unnecessarily.

Cap lamps will be dropped 10 times upon a wooden floor from a point 3 feet above it. There must be no breakage of the battery jar nor material distortion of the casing.

SAFETY DEVICES.

In making tests of the safety devices it will be assumed that if the short-circuit current of the battery does not exceed a certain value, stated previously, the glowing filament of the lamp is the only source of danger.

It will also be assumed (based on tests reported in Technical Paper 23) that the glowing filament presents an element of danger in the presence of mine gas if the bulb of the lamp can be broken without causing the filament to become wholly or practically extinguished as the result of the action of the safety devices with which the lamp is provided.

The tests will therefore be made with a view to determining whether or not the lamp bulb may be broken without causing the safety device of the lamp to extinguish the lamp or cause the filament to glow at a temperature that is not high enough to ignite explosive mixtures of mine gas and air.

If the safety devices are designed to extinguish the lamp before the bulb is broken, it will not be necessary to make the tests in gas unless the safety devices do not completely extinguish the lamp. It will then be necessary to determine whether or not the filament is glowing at a temperature sufficient to ignite gas.

If the safety devices are designed to extinguish the lamp at the same time that the bulb is broken, it will be desirable to make the tests in explosive mixtures of gas and air.

Gas, if used, will be the natural gas supplied to the city of Pittsburgh. The composition of this gas, as determined from recent analyses, is approximately 83.1 per cent methane, 16 per cent ethane, 0.9 per cent nitrogen, and a trace of carbon dioxide.

The details of conducting the tests will manifestly not be the same for all lamps submitted, because different lamps will no doubt have safety devices differing in design, construction, and basic principles. The bureau proposes to determine for each lamp separately a schedule of tests that, after due examination of the lamp and its safety devices, seem best adapted to ascertaining the merits of the equipment submitted. This schedule may be examined and discussed by the manufacturer's representative before the tests are begun.

In general, the tests will consist of striking the mounting or holder of the lamp bulb in an attempt to break the bulb without extinguishing the lamp.

If the safety devices are designed to extinguish the lamp (as by disconnecting the bulb from circuit or by opening the circuit at some other point), the devices will be considered to have acted—

1. If, after the blow has been delivered, the lamp bulb, whether broken or not, is clearly disconnected from circuit.

2. If, after the blow has been delivered—

(*a*) When the lamp filament is not broken by the blow, it does not glow, or

(*b*) When the lamp filament is broken by the blow, a sound filament, replacing the broken filament, does not glow.

If the safety devices are designed to decrease the temperature of the filament (by short-circuiting the filament or by other means), the devices will be considered to have acted if, after the blow has been delivered—

(*a*) When the lamp filament is not broken by the blow, it does not glow at a temperature sufficient to ignite gas, or

(*b*) When the lamp filament is broken by the blow, a sound filament, replacing the broken filament, does not glow at a temperature sufficient to ignite gas.

If there is any question as to whether or not a filament is glowing at a dangerous temperature, the point will be settled by surrounding the filament with an explosive mixture of gas and air.

If, after the blow has been delivered, the bulb has not been broken and the safety devices have not acted, the test will be repeated with the same equipment or with a different equipment, at the discretion of the bureau's engineers.

The bureau believes that approximately 50 tests will be necessary to determine whether or not the safety devices of a lamp are permissible for use in gaseous mines, but more or fewer tests may be made at the discretion of the engineer in charge of the tests.

DETERMINATION OF MAXIMUM SHORT-CIRCUIT CURRENT.

The short-circuit current of the battery will be measured under conditions that will give the same current that would flow through a short circuit between the conductors of the flexible cord at the point in the cord nearest to the battery casing.

TIME OF BURNING, FLUX OF LIGHT, INTENSITY OF LIGHT, AND DISTRIBUTION OF LIGHT.

The tests to determine the time of burning, flux, intensity, and distribution of light will be made for not less than 20 batteries, 6 reflectors or lamp mountings, and 100 lamp bulbs.

The average performance of the various equipments will be taken as the average performance of the lamp. The measurements of flux and intensity of light will be made after the bulbs have been burned for about 10 hours in order to season them somewhat.

VARIATION IN CURRENT CONSUMPTION, CANDLEPOWER, AND LIFE OF BULBS.

Measurements of current consumption and candlepower will be made with bulbs that have been burned about 10 hours.

Measurements of current consumption will be made at approximately the average potential given by the lamp battery after having been used for one hour.

Measurements of bulb candlepower will be made in one direction only. Usually the direction that gives the largest exposure of filament will be selected.

Determination of bulb life will be made with batteries that have the same voltage characteristics as those used with the lamp. Tests will be made with the bulbs in a fixed position.

Although, as stated in Technical Paper 75, Bureau of Mines, the bureau considers that the batteries of portable electric mine lamps should give 3,600 hours of service (300 12-hour shifts) without requiring repairs to or replacements of any part, it is manifestly impracticable for the bureau to carry out the 3,600-hour test upon each battery submitted for approval. Therefore the requirements of the bureau with respect to the durability of batteries will be considered as satisfied if the batteries shall perform their functions without repair while being used by the bureau, in accordance with the written instructions of the lamp manufacturer to conduct the bulb-life tests, and at the completion of these tests the condition of the batteries shall give no evidences of weaknesses that indicate the early failure of any part of the battery.

LEAKAGE OF ELECTROLYTE.

The lamps will be tested for leakage and spilling of electrolyte by placing the batteries for various lengths of time, totaling eight hours, in various positions that seem most likely to cause the cells to leak or spill. If a battery does not leak or spill more than one full drop of electrolyte during the eight-hour test, the battery casing will be regarded as nonspilling.

APPROVAL OF PERMISSIBLE PORTABLE ELECTRIC MINE LAMPS.

APPROVAL PLATES.

The manufacturers will be required to attach to the battery casing of each permissible lamp equipment a plate bearing the seal of the Bureau of Mines, and inscribed as follows:

Permissible portable electric mine lamp. Approval No. —.

Issued for safety and for practicability and efficiency in general service to the ———— Co.

The use of the plate will not be required if the same inscription is stamped or cast into the casing of the battery.

NOTIFICATION OF MANUFACTURER.

As soon as the bureau's engineers are satisfied that a lamp is permissible the manufacturer of the lamp and the mine-inspection departments of the several States shall be notified to that effect. As soon as a manufacturer receives formal notification that his lamp has passed the tests prescribed by the bureau he shall be free to advertise such lamp as permissible.

SCOPE OF APPROVAL.

The bureau's approval of any lamp shall be construed as applying to all lamps made by the same manufacturer that have the same construction in the details considered by the bureau, but to no other lamps.

Manufacturers shall, before claiming the bureau's approval for any modification of any approved lamp, submit to the bureau drawings that shall show the extent and nature of such modifications, in order that the bureau may decide whether or not it should test the remodeled lamp before approving it. Each approval of a permissible lamp will be given a serial number. Approvals of modified forms of a previously approved lamp will bear the same number as the original approval, with the addition of the letters a, b, c, etc.

WITHDRAWAL OF APPROVAL.

The bureau reserves the right to rescind, for cause, at any time any approval granted under the conditions herein set forth.

APPROVAL OF LAMP BULBS ALONE.

The bureau will, upon request, make tests of lamp bulbs to determine whether or not they will comply with the bureau's requirements when used in connection with any lamp that has been approved by the bureau under the provisions of this schedule.

Lamp bulbs that fulfill the requirements will be specifically approved for use with stated lamps.

Applications for tests of bulbs should be made in a manner similar to application for tests of lamps.

FEES FOR TESTING PORTABLE ELECTRIC MINE LAMPS.

The necessary expenses involved in testing portable electric mine lamps have been determined, and the following schedule of fees to be charged on and after the date of issue of this schedule has been established and approved by the Secretary of the Interior:

ITEM 1, COMPLETE TEST.

For a complete official investigation leading to the formal approval of a portable electric mine lamp, the investigation to include tests of the safety devices and the determination of the time of burning, flux of light, intensity of light, distribution of light, bulb characteristics, leakage of electrolyte, and durability	$150.00

ITEM 2, SAFETY TESTS ONLY.

For tests of the safety devices only	30.00
For additional necessary tests under the same investigation (for each five tests or fraction thereof)	2.50

ITEM 3, TESTS OTHER THAN FOR SAFETY.

For tests to determine only the time of burning, flux of light, intensity of light, distribution of light, bulb characteristics, and leakage of electrolyte	$120.00

ITEM 4, BULB TESTS.

For tests to determine only bulb life, variation in bulb candlepower, and variation in bulb current consumption:

If such tests involve making discharge voltage determinations	75.00
If such tests do not involve making discharge voltage determinations	50.00

ITEM 5, INDIVIDUAL TESTS.

The following charges will be made for individual tests included under item 3:

Discharge voltage tests	25.00
Reflector tests	20.00
Time of burning tests	10.00
Light distribution tests	5.00
Electrolyte spilling tests	3.00
Short-circuit tests of battery	1.00
Mechanical tests of cord	6.00
Bulb life tests	35.00
Bulb uniformity tests	15.00

ITEM 6, SPECIAL TESTS.

Special tests that circumstances shall render necessary during the course of the investigation will be made at the request of the lamp manufacturer and will be charged for in accordance with the amount of work involved.

REMITTANCES.

Manufacturers who submit lamps for tests to determine permissibility for use in gaseous mines will be required to furnish certified check or bank draft made payable to the Secretary of the Interior, to cover the total fees required for the desired tests. Such fees must be received at least two weeks prior to the date set for beginning the tests; otherwise the equipment of the next applicant upon the list will be tested.

SYNOPSIS OF PROCEDURE TO BE FOLLOWED IN MAKING APPLICATION FOR TESTS, SUBMITTING MATERIAL, CONDUCTING TESTS, AND NOTIFYING APPLICANT OF RESULTS.

1. Application for tests should be addressed to the Director of the Bureau of Mines, Washington, D. C. This application should be accompanied by check or draft and by a complete description of the lamp to be tested and a set of the drawings described in paragraph 2, page 45. Duplicate copies of the application, description, and drawings should be sent to the electrical engineer, Bureau of Mines, Fortieth and Butler Streets, Pittsburgh, Pa., accompanied by a single complete lamp equipment.

2. As soon as the application has been reviewed by the bureau's engineers the applicant will be notified of the date of test and the number of lamps and lamp parts that it will be necessary for him to submit.

3. After receiving this notification the applicant should send the material required to the electrical engineer, Bureau of Mines, Fortieth and Butler

Streets, Pittsburgh, Pa. This material should be delivered not less than one week in advance of the date set for the beginning of the tests. At this time the applicant should state the name and address of the applicant's representative who will witness the tests.

4. The tests will be begun on the date set and continued until the lamp is approved, rejected, or withdrawn.

5. After the bureau's engineers have considered the results of the tests, a formal report of the approval or disapproval of the lamp will be made to the applicant in writing by the Director of the Bureau of Mines. No verbal report will be made and the details of the tests must be regarded as confidential by all present.

PUBLICATIONS ON ELECTRICAL EQUIPMENT FOR MINES.

A limited supply of the following publications of the Bureau of Mines has been printed and is available for free distribution until the edition is exhausted. Requests for all publications can not be granted, and to insure equitable distribution applicants are requested to limit their selection to publications that may be of especial interest to them. Requests for publications should be addressed to the Director, Bureau of Mines.

The Bureau of Mines issues a list showing all its publications available for free distribution as well as those obtainable only from the Superintendent of Documents, Government Printing Office, on payment of the price of printing. Interested persons should apply to the Director, Bureau of Mines, for a copy of the latest list.

PUBLICATIONS AVAILABLE FOR FREE DISTRIBUTION.

BULLETIN 46. An investigation of explosion-proof mine motors, by H. H. Clark. 1912. 44 pp., 6 pls., 14 figs.

BULLETIN 52. Ignition of mine gases by the filaments of incandescent electric lamps, by H. H. Clark and L. C. Ilsley. 1913. 31 pp., 6 pls., 2 figs.

BULLETIN 68. Electric switches for use in gaseous mines, by H. H. Clark and R. W. Crocker. 1913. 40 pp., 6 pls.

TECHNICAL PAPER 4. The electrical section of the Bureau of Mines, its purpose and equipment, by H. H. Clark. 1911. 12 pp.

TECHNICAL PAPER 19. The factor of safety in mine electric installations, by H. H. Clark. 1912. 14 pp.

TECHNICAL PAPER 22. Electrical symbols for mine maps, by H. H. Clark. 1912. 11 pp., 8 figs.

TECHNICAL PAPER 23. Ignition of mine gas by miniature electric lamps with tungsten filaments, by H. H. Clark. 1912. 5 pp.

TECHNICAL PAPER 28. Ignition of mine gas by standard incandescent lamps, by H. H. Clark. 1912. 6 pp.

TECHNICAL PAPER 44. Safety electric switches for mines, by H. H. Clark. 1913. 8 pp.

TECHNICAL PAPER 47. Portable electric mine lamps, by H. H. Clark. 1913. 8 pp.

TECHNICAL PAPER 58. Action of acid mine water on the insulation of electrical conductors, a preliminary report, by H. H. Clark and L. C. Ilsley. 1913. 26 pp., 1 fig.

TECHNICAL PAPER 75. Permissible electric lamps for miners, by H. H. Clark. 1914. 21 pp., 3 figs.

TECHNICAL PAPER 79. Electric lights for oil and gas wells, by H. H. Clark. 1914. 8 pp.

TECHNICAL PAPER 108. Shot firing in coal mines by electricity controlled from the outside, by H. H. Clark, N. V. Breth, and C. M. Means. 1915. 36 pp.

TECHNICAL PAPER 159. Production of explosives in the United States during the calendar year 1915, with notes on coal-mine accidents due to explosives and a list of permissible explosives, lamps, and motors tested prior to June 1, 1916, compiled by A. H. Fay. 1916. 24 pp.

MINERS' CIRCULAR 5. Electrical accidents in mines, their causes and prevention, by H. H. Clark, W. D. Roberts, L. C. Ilsley, and H. F. Randolph. 1911. 10 pp., 3 pls.

PUBLICATION THAT MAY BE OBTAINED ONLY THROUGH THE SUPERINTENDENT OF DOCUMENTS.

TECHNICAL PAPER 101. Permissible explosion-proof electric motors for mines; conditions and requirements for test and approval, by H. H. Clark. 1915. 14 pp., 2 pls., 1 fig. 5 cents.

INDEX.

B.

Page.
Batteries, life of, determination of 50
  need of testing 11, 16
  short-circuit current of, determination of 49
    fees for testing 52
    permissible values for 47
  protection of, requirements for 46
  See also lamps named.
Bauder, P. F., acknowledgment to 9
Bulbs, efficiency of, factors governing 19
  life of, measurement of 16, 50
  requirements for 47, 48
  tests of 11, 15, 16, 51
    fees for 52
  See also lamps named.
Bunnell, C. M., acknowledgment to 9
Bureau of Mines, approval of lamps by 8
  investigations of 8, 10
  specifications for testing schedules for 8
  tests of lamps, conditions governing 9, 45
Burning, time of, requirements for 47
  tests for determining 15, 50
    fees for 52
  See also lamps tested.
Burrows, R. P., acknowledgment to 9

C.

Candlepower, of lamps, factors governing 14
  measurement of 14, 50
    device for, view of 14
Cap lamps, strength of, tests determining 48
  See also Concordia cap lamp; Edison lamp; General Electric lamp; Pioneer lamp; Wico lamp.
Concordia cap lamp, approval of 9, 37, 39
  batteries of, life of 39
    maximum current from short-circuiting 38
  bulb of, characteristics of 39
  construction of, changes in 39, 40
  cords of, life of 39
  description of 37
  headpiece of, figure showing 37
  light produced by 38
  parts of, view of 38
  safety device of, description of 37, 38

Concordia cap lamp—Continued. Page.
  specifications for 44
  time of burning of 38
  view of 38
Concordia hand lamp, approval of 9, 27, 29
  batteries for, life of 29
    current in short-circuiting 27
    tests of 28
  bulbs for, characteristics of 28, 29
  construction of, changes in 29
  description of 27
  light produced by 28
  parts of, view of 28
  safety device of, description of 27
  specifications for 44
  time of burning of 28
  view of 28
Cord, improved, development of 20
  suitable, difficulties in obtaining 16, 17, 19, 20
  tests of 20
    description of 21
    device for 21
      views of 22
    fees for 52
    results of 22
  See also lamps tested.

E.

Edison lamp, approval of 9, 22
  batteries of, life of 25
    current from short-circuiting 23
  bulbs of, characteristics of 24, 25
  connections of, improvement in 23
  construction of, changes in 26
  cord of, life of 25
  description of 22
  distribution of light by 24
  headpiece of, figure showing 23
  parts of, view of 24
  safety device of 22
    figure showing 23
  specifications for 44
  suggestions for improvement of 27
  time of burning of 24
  view of 22
Electric lamps, approval of, notification of 51
    plates for 50
    requirements for 46
    scope of 51
    withdrawal of 51
  approved, adoption of 48
    qualifications of 48

Electric lamps—Continued. Page.
- construction of, factors in 46
- dangers from 7
- early, defects of 7
- essential qualifications of 10, 11
- investigations of 9, 10
- mechanical strength of, determination of 17, 48
- practicability of, factors determining 10
- probable development of 43
- specifications for, schedules for 8, 9
- tests of, application for 52, 53
  - conditions governing 45, 46
  - fees for 51, 52
  - remittances for 52
- unexpected extinction of, tests determining 12
- *See also* lamps named.

Electrolyte, leakage of, equipment for testing, views of 14
- requirements regarding 48
- tests determining 13, 50
  - fees for 52

F.

Fees for testing electric lamp equipment 51, 52

G.

Gas, liability to ignite, tests determining 11, 12
- equipment for, view of 12
- need for 10, 11
- *See also* lamps tested.

General Electric lamp, approval of 9, 30, 33
- batteries of, life of 32
- bulbs used for, characteristics of 32
- construction of, changes in 33
- cord used with, life of 33
- description of 29, 30
- headpiece of, figure showing 30, 31
- light produced by 32
- parts of, view of 32
- safety device of, description of 30, 31
- specifications for 44
- time of burning of 32
- views of 30

H.

Hand lamps, mechanical strength of, tests determining 48
- *See also* Concordia hand lamp.

J.

Jennings, J. T., acknowledgment to 9

L.

Light, distribution of, determination of 15
- even, need of 18, 19

Light, distribution of—Continued. Page.
- improvements in 18, 19
- requirements for 47
- tests to determine 50
  - fees for 52
- variations in, views of 20
- flux of, factors determining 13, 15
  - measurement of 14
  - requirements for 47
  - tests to determine 50
    - need for 11
- *See also* Candlepower, and lamps named.

Light stream, angle of, requirement for 18

M.

Manlite lamp, approval of 9
- withdrawal of 27

O.

Open-flame lamp, in coal mines, dangers from 7

P.

"Permissible" portable electric lamp, definition of 45

"Picking test," description of 13
- machine assembled for, view of 14

Pioneer lamp, approval of 9, 43
- batteries of, life of 42
  - current from short-circuiting 40
- bulb of, characteristics of 42
- construction of, changes in 42, 43
- cord used in, life of 42
- description of 40
- headpiece of, figure showing 41
- light produced by 41, 42
- parts of, view of 38
- safety devices of, description of 40
- specifications for 44
- tendency to unexpected extinction in 41
- time of burning 42
- view of 38

R.

Reflectors, efficiency of 18, 19
- tests of, fees for 52

S.

Safety, as requisite of lamps 10

Safety devices, of lamps, development of 18
- requirements for 46, 47
- tests of 11, 12
  - equipment for, views of 12
  - factors determining 48, 49
  - fees for 51
  - purpose of 49
- *See also* lamps named.

Safety lamp, flame, disadvantages of — Page. 7
"Shoveling test," description of — 13
 machine assembled for, view of — 14
"Spilling test," method of conducting — 13

W.

Wico lamp, approval of — 9, 33, 36, 37
 batteries of, life of — 36
  maximum current by short-circuiting — 34
Wico lamp—Continued. Page.
 bulbs used with, characteristics of — 35, 36
 construction of, changes in — 36
 cord used with, life of — 36
 description of — 33
 headpiece of, figure showing — 34
 light produced by — 35
 parts of, view of — 34
 safety device of, description of — 34
 specifications for — 44
 time of burning of — 35
 view of — 28

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Bulletin 132

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY
BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

SILICEOUS DUST IN RELATION TO PULMONARY DISEASE AMONG MINERS IN THE JOPLIN DISTRICT, MISSOURI

BY

EDWIN HIGGINS, A. J. LANZA, F. B. LANEY
AND GEORGE S. RICE

WASHINGTON
GOVERNMENT PRINTING OFFICE
1917

The Bureau of Mines, in carrying out one of the provisions of its organic act—to disseminate information concerning investigations made—prints a limited free edition of each of its publications.

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SUPERINTENDENT OF DOCUMENTS,
Government Printing Office,
Washington D. C.

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CONTENTS.

Page.

Preface, by Van. H. Manning.......... 1

Mining operations as related to production of siliceous dust, by Edwin Higgins. 3

- Introduction.......... 3
- Acknowledgments.......... 3
- General information.......... 4
- Geology and ore deposits.......... 7
- Prospecting and development work.......... 8
- Mining methods.......... 9
  - Breaking ground in the heading.......... 10
  - Carrying stope and heading.......... 11
  - Drilling.......... 12
  - Squibbing, blowing holes, blasting.......... 13
  - Pillars.......... 14
  - Shoveling, tramming, and hoisting.......... 14
- Mine water.......... 16
- Ventilation.......... 16
- Temperatures.......... 18
- Wages and costs.......... 18
- Comments on the methods employed.......... 19
- Origin of siliceous dust in mine air.......... 20
  - Drilling.......... 20
  - Squibbing.......... 20
  - Blowing of dry holes.......... 21
  - Blasting.......... 21
  - Shoveling.......... 21
  - Bowlder popping and other causes.......... 21
- Methods of sampling siliceous dust.......... 22
  - Accuracy of sampling device.......... 23
- Results of the dust sampling.......... 24
  - Sizing and chemical tests.......... 24
  - Microscopic examination of dust particles.......... 26
    - General description.......... 26
    - Physical character of dust particles.......... 26
    - Description and discussion of photomicrographs.......... 26
  - Suspension of dust particles in the air.......... 28
    - Microscopic examination of the particles caught on slides.......... 28
- Size of dust particles taken into the lungs.......... 31
- Weight of dust in mine air.......... 32
  - Blowing drill holes.......... 44
  - Drilling.......... 45
  - Squibbing.......... 47
  - Blasting.......... 48
  - Shoveling.......... 49
  - General conditions.......... 49
- What constitutes a dusty mine.......... 50

Mining operations, etc.—Continued.

Page.
- Attempts to abate siliceous dust in the mines........ 51
  - Cooperative work of Federal and State officials........ 52
  - Southwestern Missouri Mine Safety and Sanitation Association........ 52
  - Preliminary report of the Federal representatives........ 53
  - Results of educational work and publicity........ 54
- Legislation looking to improved sanitary conditions........ 54
  - Mining laws of Missouri, passed by the Forty-eighth General Assembly, 1914........ 55
- Summary and conclusions........ 57
- Notes regarding health conditions in metal mines in foreign countries........ 59
  - Report on health of Cornish miners........ 59
    - Summary........ 59
    - Recommendations........ 60
  - Mining regulations in South Africa........ 61
  - Mining regulations of New Zealand........ 61

Physiological effects of siliceous dust on the miners of the Joplin district, by A. J. Lanza........ 62
- Introduction........ 62
- Educational work........ 62
- Prevalence and nature of pulmonary diseases........ 64
  - Nature of miners' consumption........ 65
- Results of physical examination of 720 miners........ 67
  - Cases of tuberculosis without rock-dust injury........ 68
  - Cases of miners' consumption only........ 69
  - Cases of miners' consumption with a tuberculous infection........ 69
  - Doubtful cases........ 70
  - Miners suffering from miscellaneous diseases........ 70
  - Miners seemingly well........ 70
  - General remarks on physical examinations........ 70
- Sanitary conditions underground........ 71
  - Prevalence of siliceous dust........ 71
  - Supply of drinking water........ 72
  - Human waste........ 72
  - Metallic poisoning........ 73
  - Effect of use of carbide lamps........ 73
  - Compressor air........ 74
  - Overwork........ 74
  - Change houses........ 74
  - Housing conditions........ 75
- Care of the sick........ 77
- Sickness insurance........ 78
- Conclusions........ 79
- Recommendations........ 79

Historical review of silicosis, by George S. Rice........ 81
- Silicosis in South African gold mines........ 84
- Silicosis in the quartz mines of Australia........ 94
- Phthisis in Germany........ 95
- Prevalence of silicosis among miners of the United States........ 95

The chert or flint of the Joplin district, by F. B. Laney........ 99
- Introduction........ 99
- Gray chert........ 100
  - General characteristics........ 100
  - Characteristics as shown by the microscope........ 101

The chert or flint of the Joplin district, by F. B. Laney—Continued.
Gray chert—Continued. Page.
Analyses of samples of gray chert 102
Origin of gray chert 103
Jasperoid or black chert 105
General characteristics 105
Characteristics as seen by the microscope 105
Analyses of jasperoid 106
Origin of jasperoid 107
Publications on mine accidents and methods of metal mining 109
Index 113

TABLES.

TABLE 1. Number of employees, wages paid, tonnage, and value of ore mined in the Joplin district during 1914 4
2. Location and number of employees of active mines in southwestern Missouri in May, 1915 5
3. Description and analyses of air samples taken in mines of Joplin district, November, 1914 17
4. Results of chemical examination of drill cuttings 25
5. Results of sizing tests on drill cuttings 25
6. Results of tests to show size of dust particles in mine air and duration of suspension 30
7. Analyses of 39 samples of siliceous rock dust taken in sheet-ground mines of the Joplin district, Missouri, in November, 1914 33
8. Description of siliceous dust samples taken in sheet-ground mines of the Joplin district 34
9. Data regarding samples of dust produced by drilling without water .. 45
10. Data regarding samples of dust produced by drilling wet or damp holes 46

ILLUSTRATIONS.

PLATE I. *A*, Prospector near Carterville, Mo., operating horse whim by hand; *B*, Nearly completed dry and change house near Duenweg, Mo... 10
II. View of plant near Carterville, showing mill buildings and tailing heaps 12
III. Another view of plant shown in Plate II, showing offices and tailing heaps 12
IV. Apparatus for determining rock dust in mine air 22
V. Details of dust sampling device 22
VI. *A*, Rock dust from sample 215, Particles magnified 250 diameters; *B*, rock dust from sample 215, Particles magnified 500 diameters; *C*, Rock dust from sample 424 26
VII. *A*, Rounded and polyhedral particles from sample 424; *B*, Particles from sample 566; *C*, Rock dust from sample 591 26
VIII. *A*, Larger particles in rock dust from sample 567; *B*, Rock dust from sample 576; *C*, Rock dust from sample 576, showing relatively large proportion of finer particles 28

Page.
PLATE IX. *A*, Rock dust particles from sample 590; *B*, Dust from mine air, caught on balsam-covered plate 28
X. *A*, Drilling dry hole with solid steel piston drill; *B*, Drilling breast hole with water-injection drill; *C*, Drilling stope hole with water-injection drill 46
XI. *A*, Type of change house formerly common and termed "dog house"; *B*, Change house of new type in course of construction. 76
XII. *A*, Scene near Webb City, Mo; *B*, The water wagon on its rounds.. 76
XIII. *A*, One-room shack, occupied by two people; *B*, Two-room shack, occupied by two adults and two children 78
XIV. *A*, Three-room boarding house; *B*, Miners' homes of the best class.. 78
XV. *A*, Photomicrograph, crossed nicols, of thin section of dense gray chert; *B*, Photomicrograph, crossed nicols, of thin section of the coarser gray chert 102
XVI. *A*, Photomicrograph, crossed nicols, of thin section of fine-grained older gray chert, showing remains of trident-shaped sponge spicule; *B*, Photomicrograph, crossed nicols, of thin section of jasperoid 102
FIGURE 1. Sketch map of southwestern Missouri and parts of Kansas and Oklahoma, showing area of active mining in sheet-ground district.... 5
2. Map of Webb City-Carterville mining district, in which is situated a large proportion of the sheet-ground mines 6
3. Plan of workings in a typical sheet-ground mine 10
4. Methods of breaking ground in thick deposits 11
5. Methods of breaking ground in deposits of medium thickness...... 11
6. Three methods of placing drill holes 12

PREFACE.

Soon after the Federal mining investigations into safety and health conditions were begun, the late Dr. J. A. Holmes, who was in charge, had brought to his attention the serious conditions that prevailed among the metal miners of certain districts, seemingly from inhaling siliceous or flinty dust in their underground work and thereby contracting pulmonary diseases. With the assistance of the United States Public Health Service, Dr. S. C. Hotchkiss made a preliminary survey in Western mines. The result of the preliminary investigations indicated a widespread prevalence of pulmonary troubles among metal miners in certain districts.

Subsequently a special investigation was conducted in the Joplin district of Missouri in the so-called "hard-rock" or "sheet-ground" mines. Dr. A. J. Lanza, passed assistant surgeon of the United States Public Health Service, was detailed to the Bureau of Mines by the Surgeon General to cooperate with Edwin Higgins, mining engineer of the bureau, in conducting under the general supervision of George S. Rice, chief mining engineer, a careful investigation of the health of the miners and the effect of the mining conditions with especial reference to the prevalence of silicosis. The investigators enlisted the heartiest cooperation from the mine inspectors of Missouri and from the operators of the Joplin district. The miners themselves also proved anxious to assist in the inquiry. Accordingly, a large amount of valuable information was obtained, which is given in this publication. A preliminary report, published before the inquiry was finished, appeared in Bureau of Mines Technical Paper 105.

The effect of the investigation and of the cooperation with the inspectors, operators, and miners was remarkably good. Regulations were adopted by the State of Missouri, and the operators, many of them willingly, put into effect many improvements and introduced appliances to allay the dust and to lessen the exposure of the men to it. In consequence, it is believed that the serious conditions that still prevail in the district will be rapidly improved and that the health not only of the miners, but of the community, will be improved thereby.

The success of the Joplin inquiry has led me, in conjunction with the Surgeon General of the Public Health Service, to organize a similar inquiry into the health conditions of the Butte copper mines of Montana, with special reference to the prevalence of silicosis, as preliminary reports indicate that this disease is prevalent in that district. It is hoped that, despite the more serious conditions due to the high temperatures in the deep workings, methods may be indicated by which the operators and miners may improve the health conditions, as has been done so successfully in the Joplin district.

VAN. H. MANNING.

SILICEOUS DUST IN RELATION TO PULMONARY DISEASE AMONG MINERS IN THE JOPLIN DISTRICT, MISSOURI.

By EDWIN HIGGINS, A. J. LANZA, F. B. LANEY, and GEORGE S. RICE.

MINING OPERATIONS AS RELATED TO PRODUCTION OF SILICEOUS DUST.

By EDWIN HIGGINS.

INTRODUCTION

Under its organic act the Federal Bureau of Mines is directed to conduct investigations relating to the improvement of health conditions in the mineral industries. This report describes the lead and zinc deposits and the mining methods employed in the sheet-ground area of the Joplin district, Missouri, and discusses the causes and the methods of abating rock dust in the mines, the chemical and physical characteristics of the dust, and the quantities present in mine air. Although the preliminary investigation included a number of the "soft-ground" mines, the writer spent the larger part of his time in the sheet-ground mines, for in them alone was siliceous dust found in appreciable quantities. Some results of the preliminary investigation have been published by the Bureau of Mines in Technical Paper 105.[a]

The territory covered in the preliminary investigation, during the period between November 7 and December 6, 1914, embraced parts of Jasper, Lawrence, Newton, and Greene Counties, Mo., and outlying districts in Kansas and Oklahoma. The remainder of the field work, to which 84 days during the period from February 9 to July 19, 1915, were devoted, was confined to the sheet-ground mines, the most important of which are situated in the vicinity of Joplin, Webb City, Carterville, and other smaller near-by towns in Jasper County, Mo.

ACKNOWLEDGMENTS.

The writer is deeply indebted to the mine operators of the district for their uniform courtesy and cooperation, and to Messrs. C. M. Harlan, W. W. Holmes, and I. L. Burch, State mine inspectors, who did much to facilitate the field work. Thanks are extended also to various officials of the Bureau of Mines—to George S. Rice, chief mining engineer, for valuable suggestions regarding the conduct of the investigation; to G. A. Burrell, chemist, for assistance in perfecting

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[a] Lanza, A. J., and Higgins, Edwin, Pulmonary disease among miners in the Joplin district, Missouri, and its relation to rock dust in the mines. 1915. 48 pp.

the rock-dust sampling device and for supervision of the analysis of air samples; to A. C. Fieldner, chemist, for directing the chemical and sizing tests and the weighing of the rock-dust samples; and to Reinhardt Thiessen, assistant chemist, for the microscopic work.

GENERAL INFORMATION.

In 1914, as shown in Table 1, the total number of employees in Greene, Jasper, Lawrence, and Newton Counties, Mo., was 5,369. This was a decrease of 982 as compared with 1913.

TABLE 1.—*Number of employees, wages paid, tonnage, and value of ores (concentrates) mined in the Joplin district during 1914.*[a]

County.	Number of miners employed.	Average daily wage.	Number of other employees.	Average daily wage.	Total number of employees.	Zinc (blende).	
						Amount mined, tons.	Value.
Greene	14	$2.00	7	$2.50	....	800	$30,500
Jasper	3,460	2.69	353	2.66	4,956	136,646	5,370,880
Lawrence	29	2.00	20	2.25	49	146	2,780
Newton	234	2.38	127	2.50	364	7,731	216.367
Total	3,737	....	507	....	5,369	145,323	5,620,527

County.	Zinc silicate (calamine).		Lead (galena).		Dry bone (smithsonite).		Total value of all minerals mined.
	Amount mined, tons.	Value.	Amount mined, tons.	Value.	Amount mined, tons.	Value.	
Greene	....	....	160	$5,646	....	....	$36,146
Jasper	1,503	$25,255	18,718	927,229	330	$14,024	6,337,388
Lawrence	7,868	27,291	6	240	261	8,101	38,412
Newton	11,595	268,902	1,474	64,923	81	2,236	522,858
Total	20,966	321,448	20,358	998,038	672	24,361	6,934,804

[a] Taken from the Twenty-eighth Annual Report of the Bureau of Mines, Mining, and Mine Inspection, of the State of Missouri.

Because of an unprecedented demandfor zinc ore in the first half of 1915, occasioned by the war in Europe, the number of men employed in and about the mines probably increased 30 per cent, many abandoned mines were reopened, and new development work was started. In short, the mining throughout the district became more active than ever before.

Table 2, which gives the number and location of the active mines in the State of Missouri on May 1, 1915, shows that the number of sheet-ground mines in operation had increased to 55 and the number of employees to 926 on the surface and 2,744 underground. The total number of employees in all types of mines increased to 6,914; this total does not take into account an unknown number of prospectors and "gougers."

TABLE 2.—*Location and number of employees of active mines in southwestern Missouri in May, 1915.*

Location.		Number of active operations, sheet ground.	Number of employees, sheet ground.		Number of active operations, disseminated and others.	Number of employees, disseminated and other mines.	
Town.	County.		Surface.	Under-ground.		Surface.	Under-ground.
Joplin	Jasper	11	132	390	40	425	740
Webb City	do	16	237	705	7	50	110
Carterville	do	6	170	606	15	85	150
Duenweg	do	4	71	217	20	115	180
Neck City	do				9	112	230
Oronogo	do	2	48	140			
Prosperity	do	10	152	334	8	25	40
Carthage	do				5	40	75
Carl Junction	do	1	20	35	2	10	20
Alba	do				4	38	66
Porto Rico	do	5	96	317	2	15	20
Sarcoxie	do				1	9	15
Spring City	Newton				6	86	177
Granby	do				8	61	127
Melva	Taney				1	10	20
Springfield	Greene				3	23	45
Aurora	Lawrence				6	40	85
Total		55	926	2,744	137	1,144	2,100
Total for November, 1914[a]		51	774	2,475	118	1,063	1,986

[a] Lanza, A. J., and Higgins, Edwin, Pulmonary disease among miners in the Joplin district, Missouri, and its relation to rock dust in the mines; a preliminary report: Tech. Paper 105, Bureau of Mines, 1915, p. 14.

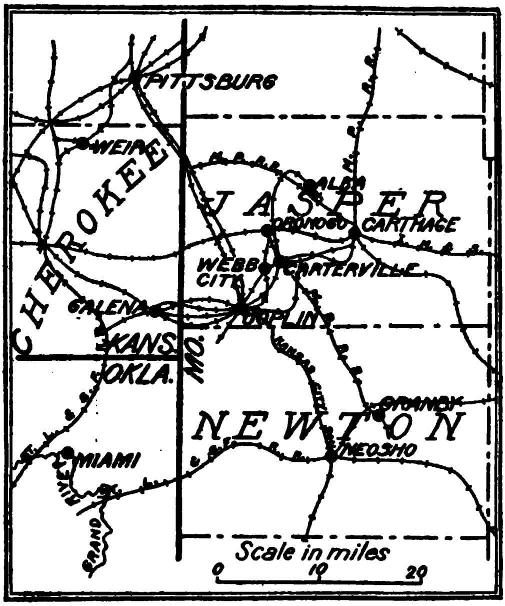

FIGURE 1.—Sketch map of southwestern Missouri and parts of Kansas and Oklahoma, showing *area of active mining* in sheet-ground district.

Several counties in southwestern Missouri and closely adjoining parts of Kansas and Oklahoma are shown in figure 1. Figure 2 shows the territory from Oronogo, Jasper County, Mo., southeasterly through Webb City, Carterville, Prosperity, Porto Rico, and Duenweg, wherein are situated a large proportion of the sheet-ground mines.

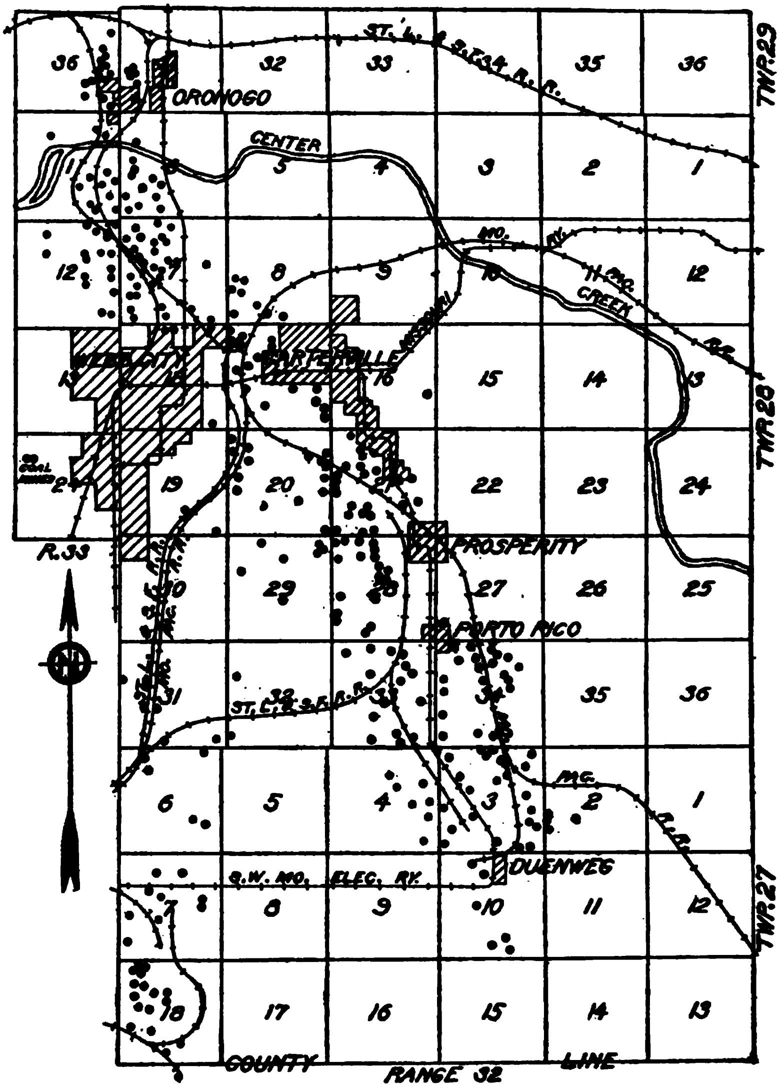

FIGURE 2.—Map of Webb City-Carterville mining district, in which is situated a large proportion of the sheet-ground mines.

The area and extent of the sheet-ground district has been described by Siebenthal[a] in his report on the Joplin district.

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[a] Siebenthal, C. E., Joplin district, Missouri, U. S. Geol. Survey Folio 148, 1907.

GEOLOGY AND ORE DEPOSITS.

Details of the geology of the Joplin district may be found in various publications,[a] and have been briefly discussed in Technical Papers 41 [b] and 105.[c] The following statements are general, but bring out some of the more important features.

The ore deposits may be divided broadly into two classes: (*a*) Those termed "sheet-ground deposits" (in chert), and (*b*) those in which the ore lies in "runs" or irregular pockets (in limestone). As regards prevalence of siliceous dust, the latter deposits need not be seriously considered. In Jasper County, where are the greater number of sheet-ground mines, the ore deposits occur chiefly in the Grand Falls member of the Boone chert, a formation in the Mississippian series of the Carboniferous system. The sheet-ground deposits consist of bedded chert in layers a few inches to 30 inches thick, which lie almost horizontally. The chert beds range from 6 to 30 and sometimes 45 feet in thickness.

The "runs" or irregular bodies of ore occur in both hard and soft ground. As a rule they lie at shallower depths than the sheet-ground deposits, but in places they lie directly below the chert, and although the ore is closely associated with chert the ground may, or may not, consist largely of limestone in the form of bowlders or masses.

The following description[d] of the rock composing the sheet ground is of interest as bearing on the nature of the dust produced:

> The typical Grand Falls chert of the Boone formation is of the splintery, fresh, unaltered appearing type known as "live" or "butcher-knife" flint. Much of it has been thoroughly crushed in place and recemented with a darker bluish siliceous cement, the original bed remaining practically undisturbed. Where the chert has been subjected to rather sharp deformation instead of suffering simple brecciation, it loses its bedded character and becomes gnarled and knotted in structure, weathering with a very rough surface.

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[a] Schmidt, A., and Leonard, A., Lead and zinc regions of southwestern Missouri: Missouri Geol. Survey, vol. 1, 1874, pp. 381–502; Winslow, A., Lead and zinc deposits: Missouri Geol. Survey, vols. 6, 7, 1894; Jenney, W. P., Lead and zinc deposits of the Mississippi Valley: Trans. Am. Inst. Min. Eng., vol. 22, 1894, pp. 171–225, 642–646; Bain, H. F., Van Hise, C. R., and Adams, G. I., Preliminary report on the lead and zinc deposits of the Ozark region: Twenty-second Ann. Rept. U. S. Geol. Survey, pt. 2, 1901, pp. 23–228; Smith, W. S. T., Lead and zinc deposits of the Joplin district, Missouri-Kansas: U. S. Geol. Survey Bull. 213, 1903, pp. 197–204; Buckley, E. R., and Buehler, H. A., Geology of the Granby area: Missouri Bureau Geology and Mines, 2d ser., vol. 4, 1906, p. 57; Haworth, Erasmus; Crane, W. R.; Rogers, A. F.; and others, Special report on lead and zinc: Univ. Geol. Survey Kansas, vol. 7, 1904, pp. 20, 45, 48, 484.

[b] Wright, C. A., Mining and treatment of lead and zinc ore in the Joplin district, Missouri, a preliminary report: Tech. Paper 41, Bureau of Mines, 1913, pp. 9–11.

[c] Lanza, A.J., and Higgins, Edwin, Pulmonary disease among miners in the Joplin district, Missouri, and its relation to rock dust in the mines: Tech. Paper 105, Bureau of Mines, 1915, pp. 15–16.

[d] Siebenthal, C. E., Joplin district, Missouri, U. S. Geol. Survey Folio 148, 1907.

The following record of a drill hole near Carterville, Mo., in the heart of the sheet-ground area, shows 45 feet of sheet ground:

Record of drill hole near Carterville, Mo.

	Feet.
Soil	2
Yellow clay	18
Gravel	10
Yellow clay	10
Soapstone	64
Chert and dolomitic limestone, in alternate layers	116
True sheet ground	45
Dolomitic limestone	25
	290

The sheet ground varies in depth below surface, in thickness, and in the number of beds that may be encountered. In the principal area, in the vicinity of Joplin and the towns between Oronogo and Duenweg, it may lie anywhere from 140 to 240 feet below the surface, and may be divided into two or more beds, with limestone between. In other parts of the district, however, because of differences in the dip of the strata and erosion, the sheet ground is encountered at depths of less than 100 feet.

Sphalerite, or zinc sulphide, locally termed "jack," and galena, or lead sulphide, termed "lead," are the valuable minerals of the ore bodies. These minerals are usually closely associated, but in widely varying proportions, in well-defined bands a fraction of an inch to 6 inches thick, which lie between the bedding planes of the flint at varying intervals. An ore body may consist only of one band of mineral, or it may be made up of two or more bands at varying distances apart.

PROSPECTING AND DEVELOPMENT WORK.

As the surface is comparatively level, and soil, clay, sand, and bowlders extend to varying depths, rock outcrops are seldom seen. Churn drilling has been found to be the most satisfactory method of prospecting. Although such drilling is well adapted to the flat-lying ore deposits, the results obtained may be misleading unless enough holes are drilled. As a rule the ore deposits are not pockety, but a single drill hole may pass through an ore body at a point where it is richer, or much leaner, than the average.

Keystone and Star churn drills are chiefly in use. Usually a hole is started with 6-inch and finished with 4-inch casing. Rarely is drilling continued below the sheet ground, so that there are few drill holes in the sheet-ground district deeper than 250 feet. Drilling costs from 90 cents to $1 per foot. The average time required to drill a 200-foot hole is about two weeks.

After the location of the ore body by drilling, a shaft, measuring usually 4 by 5 to 5 by 7 feet inside the timbers, is sunk. In sinking shafts, corner, side, and sump holes are drilled; only one set of sump holes is required in soft ground. Shafts are timbered as far down as the limestone with 2 by 4 inch rough lumber placed crib fashion, without special framing, on the four sides of the shaft, the space between the 2 by 4 inch timbers being filled with short pieces of the same material. Practically no blocking is used, but the cribbing is strengthened by 2 by 4 inch lumber, one or two strips of which are nailed against each side of or in the corners of the shaft, and extend from the collar to the bottom of the timbering. No ladders are used in the shafts. Men are raised and lowered in buckets and supplies and machinery either in the bucket or with the cable. No guides or crossheads are used, buckets being allowed to swing freely in their passage through the shaft. Water columns, air lines, and electric-power lines, when used, are usually placed in a corner of the shaft. Plate I, *A*, shows a prospector operating a horse whim by hand.

At the bottom, a station is cut, pillars are left to protect the shaft, and mining is begun. As a rule, development follows the ore, and a mine may be developed in one or more directions from the shaft; however, it is usually possible to work in all directions. This permits almost any arrangement, as far as trackage and layout of station are concerned, that may be desired. The "ground" (rock) is drilled and blasted down and the resulting "dirt" (broken rock) is shoveled into "cans" (buckets). The cans are trammed singly by hand, or in trips by power or mules, to the shaft station, where they are hooked onto the cable, hoisted to the top of the "derrick" (headframe), and dumped onto a 5 or 6 inch grizzly. The dirt then starts on its way through the mill. With a few slight modifications, this may be said to be the method followed in all the sheet-ground mines.

The surface works of a mine near Carterville are shown in Plates II and III.

MINING METHODS.

The winning of ore in the mines presents no great problem, only the excavation and removal of varying thicknesses of cherty flint from comparatively shallow depths. Probably the greatest difficulties encountered are in drilling and blasting the tough flint, much of which contains fissures. Experience has shown that it is much cheaper to mine from many shafts simply equipped, rather than to utilize fewer shafts of more elaborate equipment. Probably 300 feet is a fair average for the distance between shafts. The workings of a typical sheet-ground mine present the appearance of a large horizontal chamber, the roof being supported by pillars that differ in size and spacing according to the nature of the rock and the height of the roof. The height of the workings, of course, depends on the thickness of the pay dirt; the lateral dimensions on the extent of the ore body. A plan of the workings of a typical mine is shown in figure 3.

The methods of mining used can be divided into what may be termed heading work, in which the rock is broken from floor to roof with a single round of holes; and mining by stope (bench) and heading, the heading being carried above and in advance of the stope. Without reference to the method of breaking the ground, the term "longwall advancing" is applicable to the manner of carrying the face.

BREAKING GROUND IN THE HEADING.

Deposits of from 7 to 18 feet in height are usually broken with a single round of shots, the face being carried vertically. Although the stope and heading method is used in some of the mines, where the deposit is 15 to 18 feet thick (see A, fig. 4), the tendency in recent years has been to carry only a heading. The method of carrying the face and placing drill holes in heading work, the system most commonly employed, is shown in figure 5. The plan of the working face (A, fig. 5) is irregular in outline, as obviously the ground breaks more easily when an irregular face is carried.

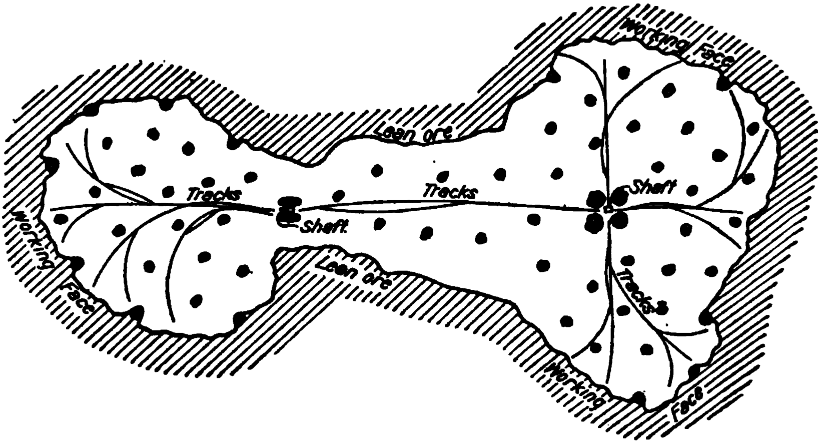
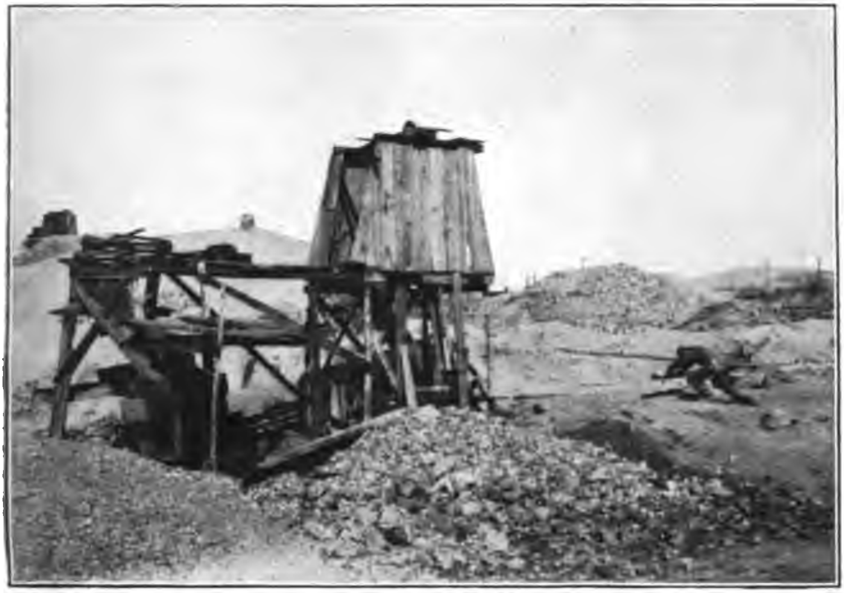


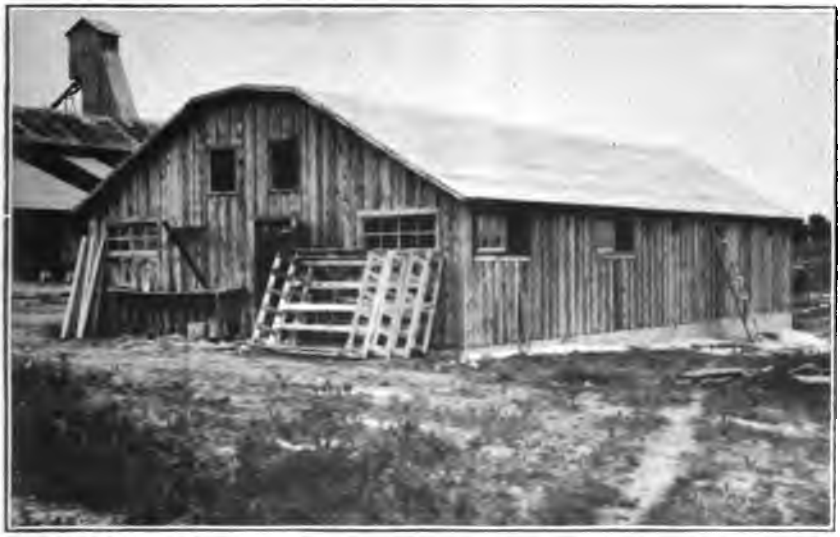
FIGURE 3.—Plan of workings in a typical sheet-ground mine.

The condition and shape of the face are the main factors in determining the number and location of drill holes; three to six holes constitute a round. Three common methods of placing holes are shown in figure 6. The numbers indicate the order in which the holes are fired. The upper holes are called roof holes; those midway between the roof and floor, breast holes; and those near the floor, stope holes.

In the 6-hole round, No. *1* is termed the relief hole; *2*, the front breast; *3*, the front stope; *4*, the back roof; *5*, the back breast; and *6*,



A. PROSPECTOR NEAR CARTERVILLE, MO., OPERATING HORSE WHIM BY HAND.



B. NEARLY COMPLETED DRY-AND-CHANGE HOUSE NEAR DUENWEG, MO.

the back stope hole. The holes are drilled approximately parallel to what might be termed a subface, and are 6 to 15 feet long, corresponding in length roughly to the height of the roof. The dotted lines at the points *a* and *b* (A, fig. 5) indicate the ground that will be broken by the rounds of shots at those places.

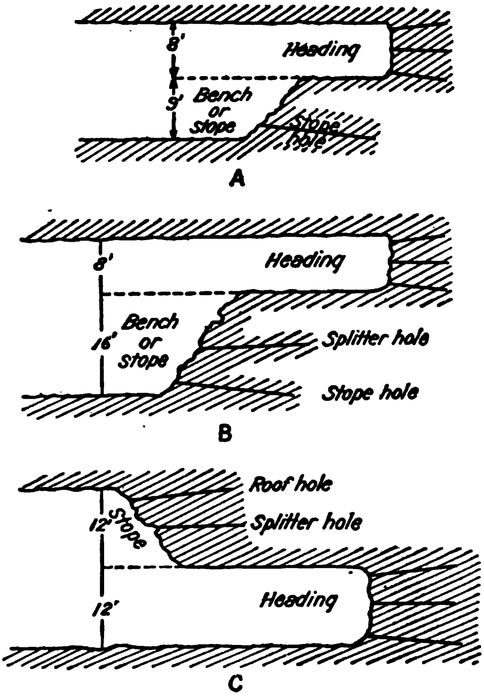


FIGURE 4.—Methods of breaking ground in thick deposits.

CARRYING STOPE AND HEADING.

Deposits 15 to 30 feet or more in thickness are usually mined with a stope (bench) and heading. The methods used in one of the larger mines are typical of the practice in the entire district. At this property the mineralized ground is 24 feet thick. The usual method of breaking the ground is shown in figure 4 (B). The chert is uniformly hard, with the exception of a small band of what is called "cotton rock" in the floor of the heading. The roof, which consists of 3 to 4 feet of chert, is considered good; limestone lies above the chert.

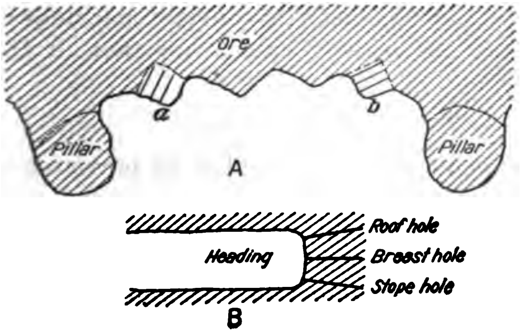


FIGURE 5.—Methods of breaking ground in deposits of medium thickness.

The heading is carried 8 feet high and 15 to 20 feet in advance of the stope, the drill holes being 7 to 8 feet long. The bench or stope is 16 feet thick, and is broken with stope and "splitter" holes 18 feet in length. The splitter hole is pointed slightly upward; the stope hole slightly downward. These holes are chambered three or four times with charges ranging from 4 or 5 sticks of dynamite for the first charge to about 30 sticks for the last one. When ready for blasting each hole will contain from 3 to 4 boxes (150 to 200 pounds) of dynamite. With such charges, two stope and two splitter holes will break the 16-foot bench across a width of 10 to 12 feet. In this mine an 18-foot hole has been drilled in 1½ hours, but holes of this depth where the chert contained many fissures have required as much as 8 hours.

In another part of this mine a practice was noted that is not usual in the district. The chert at that place is also 24 feet thick, but the upper 12 feet of the deposit is extremely tough. The heading is carried in the soft ground below, and the stope or bench above, each 12 feet high, as shown in figure 4 (C). The heading is carried in the usual way, the drill holes being 12 feet in length. The upper part, in this case the stope, is broken with roof and splitter holes 12 to 14 feet long. The splitter holes are chambered, but the roof holes are left as drilled for fear of loosening the roof. This stope is carried from 50 to 100 feet back of the heading. This method of mining is much less costly than the first one described. Five machines here produce an average of 650 (1,000-pound) buckets of dirt in 8 hours. This amount of dirt is loaded by 11 shovelers.

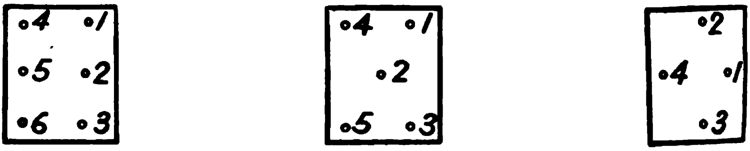


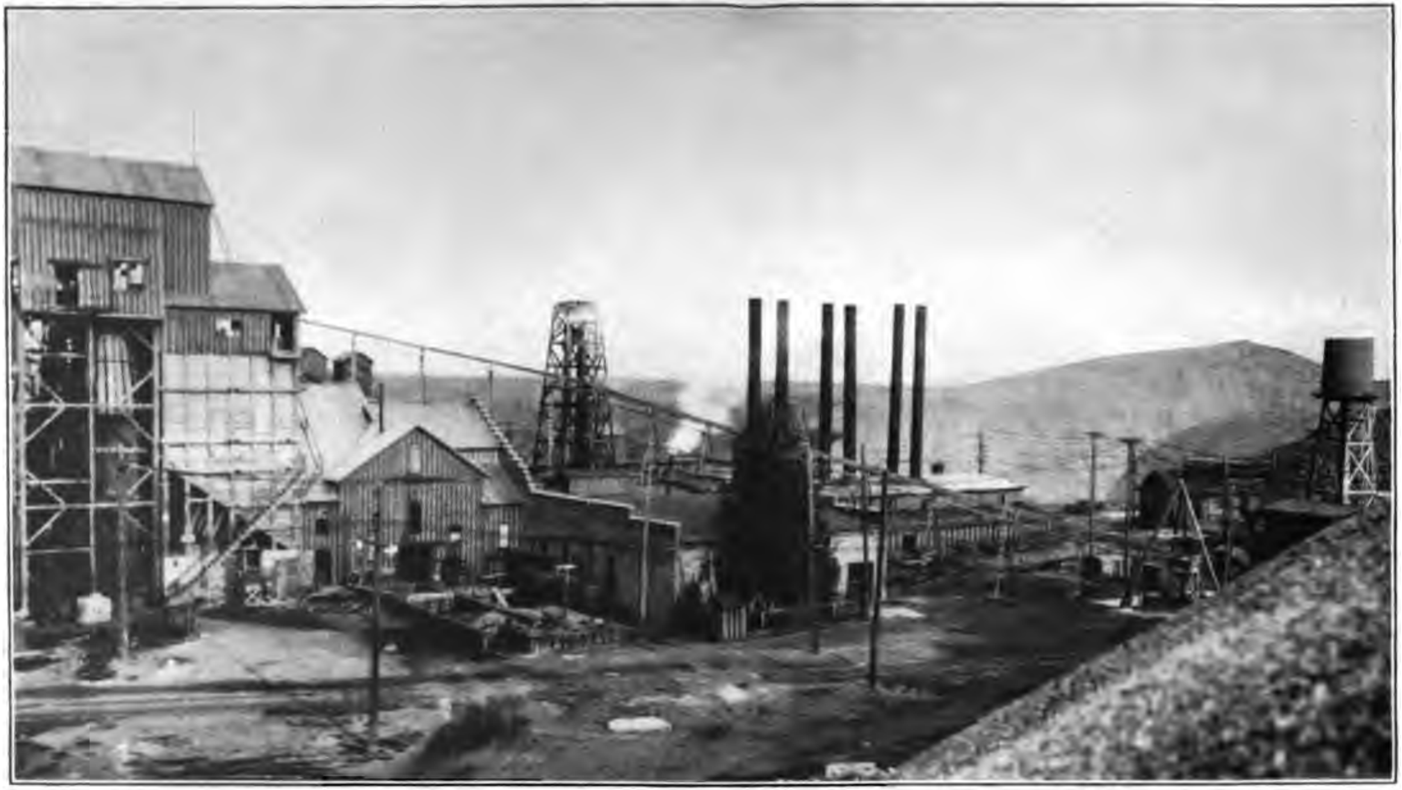
FIGURE 6.—Three methods of placing drill holes.

In some places, after a certain thickness of ground has been worked, further prospecting shows pay dirt in the floor. To mine this lower stratum all that is necessary, after cutting out the floor at the shaft station, is to drill and blast long stope holes or lifters. This procedure is called "taking up stope," and is a very cheap method of mining. One machine will produce as much dirt as eight machines in the heading; also, the cost of explosives is much less than in the heading.

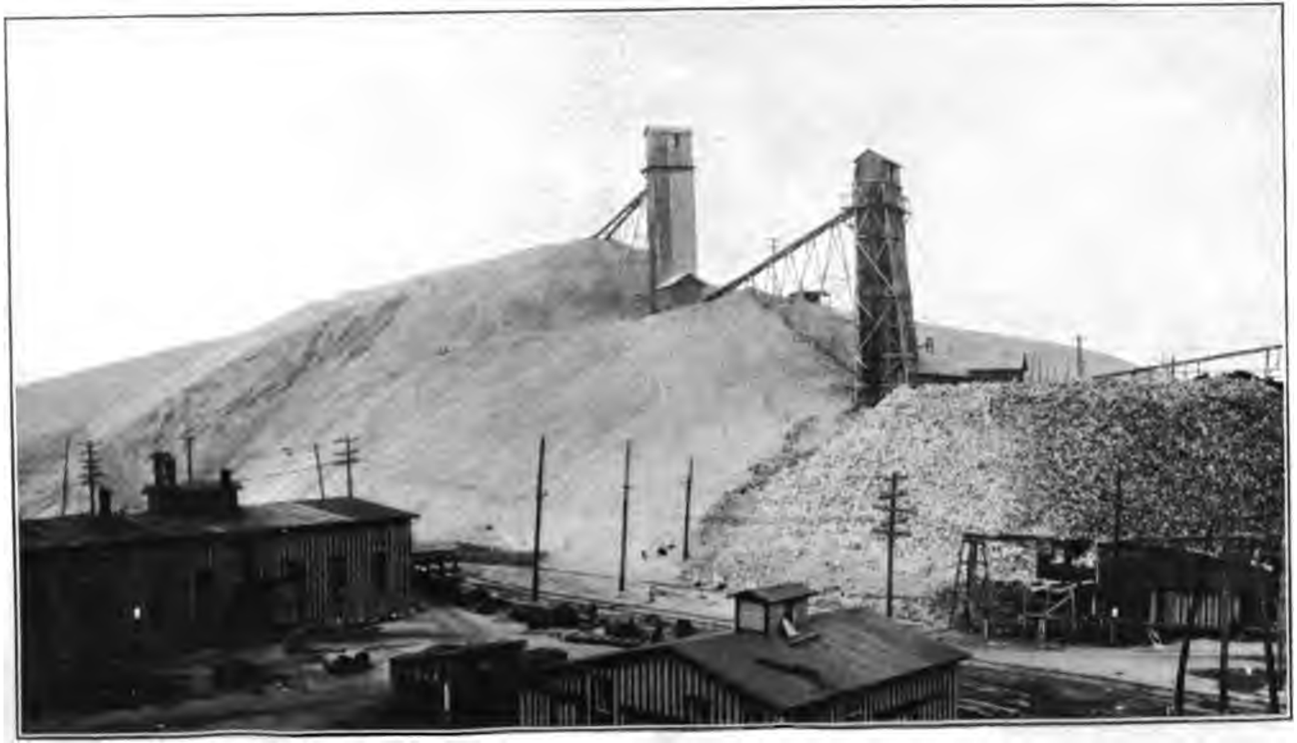
DRILLING.

All drilling is done with machine drills operated at air pressures of 60 to 90 pounds at the machine. A large proportion of the drills are of the solid-steel piston type; also successful experiments have recently been made with water-injection hammer drills.

For heading work the machine is mounted on a column; for stope work a tripod is used. A compilation of drilling records from practically all of the sheet-ground mines shows that one machine will drill 20 to 35 feet per shift and produce 35 to 45 tons of dirt, depending on the nature of the ground.



VIEW OF PLANT NEAR CARTERVILLE, SHOWING MILL BUILDINGS AND TAILING HEAPS.



ANOTHER VIEW OF PLANT SHOWN IN PLATE II, SHOWING OFFICES AND TAILING HEAPS.

SQUIBBING, BLOWING HOLES, BLASTING.

"Shelly" ground and fissured ground cause the drill steel to stick. The drill helper usually tries to prevent this sticking by repeatedly striking the drill steel with some heavy tool. If this fails to make it work freely, the general practice is to remove the drill and fire a stick or two of dynamite in the hole. This practice, called "squibbing," is one of the chief causes of siliceous dust in the mines. Although squibbing is an effective method of cleaning holes, it is usually unnecessary, because in most cases the sticking of the drill steel may be remedied by properly aligning the machine. Also, squibbing is expensive because it uses up explosive and causes the drill crew, together with the shovelers and other drill crews near by, to lose 3 to 10 minutes' time, for these men must all leave their working places and seek safety behind a pillar when the squib is fired.

Owing to the toughness of the chert, the drill holes must be chambered in order that sufficient dynamite may be introduced into the hole to break the ground. The chambering of holes is also referred to as squibbing.

Practice varies in the different mines, but it has been found most satisfactory to drill a round of holes, squib or chamber them when the men leave the mine, and shoot them at the end of the following shift, so that a round of holes is drilled one shift before blasting. If squibbing or blasting is done while men are underground, they are exposed to an atmosphere containing noxious gases from the blasts and large quantities of siliceous dust.

At various intervals, but chiefly before squibbing or blasting, it is necessary to remove the cuttings from the drill hole. A pipe attached to the compressed-air line is inserted in the drill hole, the air turned on, and the pipe worked backward and forward until all the cuttings have been blown out. This practice effectively cleans the hole, but unless it has been thoroughly wet the workings in the vicinity will be filled with a dense cloud of dust.

Ammonia and gelatin dynamites of 33 to 40 per cent strength are used in most of the mines, the ammonia dynamite for dry and the gelatin for wet holes. In exceptional cases stronger dynamites are used in long stope holes. In some of the mines "80 per cent" gelatin dynamite is used for squibbing.

In mining sheet ground the explosives constitute one of the chief items of expense, varying from 20 to as much as 30 per cent of the total cost. A box of 50 pounds of dynamite will break from 30 to 45 tons of dirt, according to the nature of the ground and the kind of face carried.

PILLARS.

The chief factors to be considered in determining the size and frequency of pillars are the nature of the ground, the thickness of the deposit, and the character of the roof. No set rule governs the practice, pillars being left as conditions may require. In some of the mines the roof is very good, but in others the cherty flint tends to fall readily; usually when the roof falls it comes down in slabs 3 to 30 inches thick. In some of the old workings of the sheet-ground mines pillars 100 feet or more apart may be seen. In some of these places the roof has fallen over a large area in sheets 1 to 2 feet thick.

The tendency in recent years has been to leave pillars 40 to 60 feet apart, arranged in "five-spot" form. This arrangement supports a slabby roof better and is superior to the method used in former years of placing pillars in well-defined rows. Some idea of the arrangement of pillars may be obtained from figure 3, which also shows pillars in the making. Ten to thirty per cent of the ground is left for pillars. In some of the mines leasers are allowed to clean up floors and pillars after company work has ceased. The leasers usually cut the pillars down to a width that allows little or no factor of safety. Some serious caves have resulted from this practice, and were it not for the vigilance of the mine inspectors caves would be more frequent.

In leaving shaft pillars sufficient attention is not always given to convenience of future operation. In some of the mines where the ground around the shaft station was very rich, rock has been removed that should have been left. In exceptional cases shafts have been left almost entirely unprotected. Figure 3 shows two different arrangements of shaft pillars. The shaft on the left is protected by two oblong pillars. In deposits of medium thickness, the shaft pillars would be about 20 feet wide and 30 feet long; in thicker deposits they would be larger. The shaft shown on the right is protected by four circular pillars, a desirable arrangement, as it permits haulage in any direction. Pillars so arranged would range from about 20 feet in diameter for deposits of medium thickness to 30 feet for thicker deposits.

SHOVELING, TRAMMING, AND HOISTING.

The broken ore, or dirt, is shoveled into buckets holding 1,000 to 1,650 pounds and trammed on small trucks to a switch, called the lay-by, near the face. From the lay-by the trucks are usually hauled to the shaft by mules or power, but frequently the shoveler trams the bucket to the shaft.

Most of the ore breaks in pieces small enough for shoveling, but there are many large bowlders. Bowlders that can not be broken with a sledge are blasted. This practice is known as "bowlder popping."

The shovelers, or "cokeys," as they are called, are paid by the bucket, and load 30 to 70 or 80 buckets in an 8-hour shift. There are some records of one man loading 90 and even 100 buckets and more per shift. The average day's work for a shoveler in all the sheet-ground mines is 22 tons per 8 hours.

The track gage ranges from 12 to 24 inches, 8 to 12 pound rails being used. There are two methods of spotting the loaded truck at the shaft station. The truck carrying the bucket may be stopped under the exact center of the shaft, so that it is not necessary to balance or steady the bucket after it leaves the truck. The descending empty bucket, however, must be drawn to one side before it is landed. The other method is to place the truck for the empty bucket in the center and stop the loaded bucket to one side of it. In this method it is necessary to center the loaded bucket after it leaves the truck.

The "derricks" or headframes range from 40 to 70 feet in height, depending on the elevation at which the dirt must be delivered to the mill.

Geared hoists, operated either by steam or electricity, are commonly used. There are some first-motion steam hoists. The hoist is installed near the top of the "derrick" in such a position that the hoist engineer from his seat at the throttle or controller has an unobstructed view to the bottom of the shaft. The hoisting rope is usually one-half to five-eighths of an inch in diameter and of steel. When a truck with a loaded bucket reaches the shaft station, the "tub hooker" or station tender attaches the cable hook to the bail of the bucket, which is then hoisted to the top of the derrick, drawn to one side by means of a hook attached by the engineer to a ring in the bottom, and quickly dumped, the ore falling on an inclined grizzly with bars spaced 5 to 6 inches. The bucket, without having been removed from the cable, is immediately returned to the shaft and lowered to the bottom, where it is immediately detached from the cable and a loaded bucket made fast.

In hoisting dirt an efficient hoist man and tub hooker work together with practically no loss of time. Dirt is hoisted without signals, the engineer apparently knowing just how much time to allow for the changing of the cable hook from the empty to the loaded bucket. It is common practice to raise and dump 120 buckets or more in one hour.

The fact that the engineer, from his place at the throttle or lever, has an unobstructed view of the bucket as it passes through the shaft makes for safety. Should the bucket, loaded either with dirt or men, approach the sides of the shaft too closely, the engine may be stopped or the speed lessened. In some of the mines there is in the shaft station an alarm gong, which may be sounded by a push button convenient to the engineer. In case a bucket of dirt is overturned in the shaft he is thus enabled to warn the tub hooker and any others who may be in danger at the shaft station.

MINE WATER.

Records of the various mines examined show that quantities of water are pumped ranging from a few gallons up to as high as 1,500 gallons per minute. In some places one pumping plant handles the water from several mines. The amount of water made by a mine, however, is no indication of the amount of siliceous dust that may be produced, for the working faces may be higher than the principal source of water, and therefore may be dry. Mines making as much as 1,000 gallons of water per minute were found to have dusty working faces, whereas mines producing much smaller amounts showed relatively less dust at the faces.

VENTILATION.

Most of the sheet-ground mines are well ventilated, after development, by natural air currents, as most mines have two or more shafts. In the vicinity of Webb City and Carterville the sheet-ground mines under a large area are connected, and it is possible to walk several miles underground through various properties without returning to the surface.

In such mines or in mines having more than one opening, although the air circulation between shafts is usually excellent, at the working faces, especially as they get farther away from a shaft, the air circulation may be poor. However, as indicated by the analyses of the air samples given in Table 3, the quality of the air at the faces seems to be affected but little. This is probably due to the usually great size and extent of the mine workings. With the relatively large volume of air, vitiation by the breathing of men, burning of lights, and combustion of explosives is evidently almost negligible. Also these mines use practically no timber in the workings, so that the absorption of oxygen and the production of carbon dioxide by decaying timbers is a factor that need not be considered. Pure atmospheric air contains approximately 20.93 per cent oxygen, 79.04 per cent nitrogen, and 0.03 per cent carbon dioxide. Table 3 shows that all the mine air sampled was nearly normal.

TABLE 3.—*Description and analyses of air samples taken in mines of Joplin district, November, 1914.*

Laboratory No.	Name of mine.	Place in mine where sample was taken.	Date taken (1914).	Analysis.				Remarks.
				CO$_2$.	O$_2$.	CO.	N.	
				P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	
5637	Schoolhouse	Heading 13	Nov. 16	0.26	20.79	0	78.95	Taken 2 minutes after squibbing 9-foot hole with 2 sticks of 35 per cent gelatin dynamite.
5638	....do	....do	...do	.11	20.85	0	79.04	Duplicate of No. 5637.
5639	Little Princess	West slope, No. 2 mill shaft.	Nov. 17	.21	20.74	0	79.05	Taken after squibbing 14-foot hole with 15 sticks of 35 per cent ammonia dynamite.
5640	....do	....do	...do	.16	20.79	0	79.05	Duplicate of No. 5639.
5641	Ice Plant	Heading 7	...do	.11	20.83	0	79.06	Taken after squibbing hole with 2 sticks of 27 per cent ammonia dynamite.
5642	....do	....do	...do	.08	20.88	0	79.04	Duplicate of No. 5641.
5635	Longacre & Chapman.	Bottom of main shaft	Nov. 16	.07	20.92	0	79.01	Main return air.
5636	....do	....do	...do	.07	20.92	0	79.01	Duplicate of No. 5635.
5630	Davey No. 3	Drift south of No. 7 shaft.	Nov. 9	.07	20.91	0	79.02	Main return air.
5629	....do	....do	...do	.07	20.92	0	79.01	Duplicate of No. 5630.
5627	....do	Face of No. 10 drift	...do	.10	20.81	0	79.09	
5628	....do	....do	...do	.07	20.90	0	79.03	Duplicate of No. 5627.
5631	Coahuilla	Station at main shaft	Nov. 10	.07	20.84	0	79.09	Main return air.
5632	....do	....do	...do	.10	20.84	0	79.06	Duplicate of No. 5631.
5626	Electrical	Southeast heading	Nov. 19	.11	20.86	0	79.03	Taken one-half hour after squibbing 2 holes with 36 sticks of 80 per cent gelatin dynamite.
5625	....do	....do	...do	.07	20.86	0	79.07	Duplicate of No. 5626.
5623	Oronogo Circle	South heading	Nov. 18	.11	20.88	0	79.01	Taken after squibbing 11-foot hole with 3 sticks of 35 per cent gelatin dynamite.
5624	....do	....do	...do	.10	20.85	0	79.05	Duplicate of No. 5623.
5622	Florine	....do	...do	.10	20.81	0	79.09	Taken after squibbing hole with 2 sticks of 35 per cent ammonia dynamite.
5621	....do	....do	...do	.21	20.69	0	79.10	Duplicate of No. 5622.
5633	Wilson	Heading 300 feet east of main shaft.	Nov. 11	.14	20.75	0	79.11	Taken after squibbing hole with 3 sticks of 33 per cent gelatin dynamite.
5634	....do	....do	...do	.11	20.93	0	78.96	Duplicate of No. 5633.
5680	Hornbrook	Second south face	Nov. 25	.07	20.78	0	79.15	Taken 5 minutes after shooting two 7-foot holes with twenty ⅞-inch sticks of 40 per cent gelatin dynamite.
5679	....do	....do	...do	.11	20.74	0	79.15	Duplicate of No. 5680.

TEMPERATURES.

The temperature in the working places of the mines ranges from 52° to 65° F., the corresponding relative humidity being 82 to 100 per cent. The surface temperature ranges from 30° to 60° F., the relative humidity averaging 50 per cent, during October, March, and April, and up to 80° and 95° F. in July. The surface temperature apparently has little or no effect on the underground temperature, despite the shallowness of the mines. Although the humidity was high in most of the working places visited by the writer, this condition caused no great inconvenience, owing to the relatively low temperature.

WAGES AND COSTS.

During the three years previous to 1915, drillmen received $2.25 to $3 for an 8-hour day, drill helpers $2 to $2.50, and shovelers 6 to 9 cents per bucket. During this period the zinc concentrate produced was sold at an average of about $43 per ton, and the lead (galena) concentrate at $52 per ton. The cost of mining and milling ranged from 80 cents to $1.25 per ton of dirt handled, a fair average for the larger operations being $1 per ton. The cost, then, of mining and milling 100 tons of dirt under such conditions would be $100. If this were "2 per cent" dirt,[a] carrying only zinc, it would yield 2 tons of concentrate, which, at $43 per ton, would return $86. Thus it may be seen that unless costs are kept below the average, it is not possible to make a profit on 2 per cent dirt with low prices for ore.

During the first quarter of 1915, owing to the great demand for spelter, occasioned by the European war, the price of zinc concentrate began to rise. Sales were recorded during the first half of 1915 as high as $135 per ton. At the same time the cost of labor increased. Drillmen received from $3.50 to $4.50, drill helpers $2.75 to $3.75, and shovelers as high as 15 cents per bucket.

During 1912, 1913, and 1914 the costs of mining were distributed as follows by one large company:

	Cents per ton.
Ground boss	1.0
Drilling	18.5
Blasting	16.0
Roof protection	.7
Shoveling	15.0
Mule haulage	6.0
Track work	2.3
Hoisting	4.0
Lighting	.6
Sundry	1.0
Total mining cost	65.1

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[a] In the Joplin district, "2 per cent" dirt means dirt that yields 2 tons of concentrate for each 100 tons mined, without allowing for the usual 25 to 30 per cent loss in concentrating.

The cost of milling averaged 27 cents per ton, making the total cost of mining and milling 92.1 cents.

The summarized mining costs of another large company for the six-year period 1909 to 1914 were as follows:

	Cents per ton.
Labor	33
Explosives	10
Air	4
Other expenses	8
Total	55

At this plant the cost of milling averaged 30 cents, and the cost of mining and milling, including everything except depreciation, was 98 cents per ton.

The costs of mining and milling by another large company for the year 1914 were distributed as follows:

	Cents per ton.
Labor	49.15
Explosives	20.49
Fuse	.57
Gas and electricity	14.28
Superintendence and repairs	12.12
Other expenses	3.39
Total mining and milling	100.00

The above figures, of course, should not be used in connection with the high prices of ore existing in 1915. With the increase in wages, the cost of mining and milling increased to $1.50 up to $2 per ton, and even more at some properties.

COMMENTS ON THE METHODS EMPLOYED.

To one not familiar with conditions in the Joplin district, the mining and milling methods may seem crude and inefficient, but the methods used are actually about as efficient as those in use in any metal-mining district in the United States. In the sheet-ground mines the working faces are, in two or three years' mining, extended to points far distant from the shaft, making necessary the sinking of new shafts from time to time. Obviously, elaborate shaft equipment, with ore pockets and skips, would be an extravagance. As a matter of fact, expensive mine equipment does not pay in this district, and it would be difficult to improve greatly on the methods in use, as far as costs are concerned. There are few metal-mining districts in the United States where ore can be mined and milled for $1 and less per ton, or where the output averages 10 tons per man employed underground, as is the case in the sheet-ground mines of the Joplin district.

ORIGIN OF SILICEOUS DUST IN MINE AIR.

Siliceous dust is present in varying quantities in the air of all operating sheet-ground mines, the quantity made and kept in suspension depending largely on the methods employed. Mines that are well ventilated and those that have wet working places are usually not dusty; however, they may be very dusty unless care be observed in certain details.

The causes of the production of siliceous dust, and its suspension in the mine air, are drilling, squibbing, blowing cuttings out of drill holes with an air blast, blasting, shoveling, bowlder popping, tramming, and roof and pillar trimming. The blowing of dry drill holes, squibbing, and bowlder popping are the chief practices that raise the dust into the mine air. On the surface dust is made by the dumping of the bucket and the crushing of rock in the mill.

The method of mining employed, involving the excavation of large open areas underground, is an important factor in the prevalence of dust in the mines. At the working faces, which may be continuous for several hundred feet, the operations of drilling, squibbing, blowing holes, shoveling, and tramming, and at times roof trimming, bowlder popping, and even blasting may be in progress at the same time. The dust made by these operations is densest at the working faces, where most of the men underground are working, but often it is carried by air currents to all parts of the mine, so that practically every man underground is in a dusty atmosphere during the entire shift. In other metal-mining districts, especially where mining is conducted on different levels, it is usual to find only part of the mine dusty, and that for only part of the day.

DRILLING.

In general, it may be said that drilling does not set in motion much of the dust made by the drill, except when the hole is being started. The bulk of the cuttings fall to the floor of the mine, but only a slight disturbance of the mine air is required to set the lighter particles in motion. Part of the cuttings remain in the drill hole, and are expelled as a cloud of dust either by squibbing or blowing the hole. Attention is directed to the fact that practically all holes drilled in these mines are approximately horizontal.

SQUIBBING.

Squibbing is one of the chief factors in setting the siliceous dust in motion. This is effected in two ways: (*a*) Dust is forcibly expelled from the drill hole, and (*b*) the vibrations set up in the mine air by the shot raise dust that has settled about the working face.

BLOWING OF DRY HOLES.

The operation of cleaning drill holes with a blast of air has been described on a previous page. The practice of blowing out holes without first wetting them is termed "blowing dry holes." More dust may be produced by it than by any other mining practice; in extreme cases the dust is so dense that the flame of a carbide lamp 15 or 20 feet distant appears as a blur of light.

BLASTING.

The blasting of one or more rounds of holes, especially where the face is dry, produces large quantities of dust. As blasting is usually done when the shift goes off, the dust so produced is chiefly harmful in that it may be set in motion when the next shift starts mining. If, as in some mines, the time between shifts is only a half or even a full hour, the dust produced by blasting will not have settled, and the second shift will suffer from it. If ventilation is good, the dust is soon removed by the air current; otherwise it hangs in suspension for several hours. The mine air after blasting is not only contaminated with dust, but contains also the gases and fumes produced by the explosives.

SHOVELING.

The quantity of dust made by shoveling depends on whether the dirt is wet, damp, or dry. If the shoveler works, as he often does, in a small puddle of water, less dust is produced than if he shovels from a dry floor. In either case the quantity of dust produced in shoveling is not excessive, but as the shoveler works with his head close to the source of the dust, and breathes deeply while working, he may take into his lungs an injurious quantity of dust during a shift.

BOWLDER POPPING AND OTHER CAUSES.

The concussion from the popping of bowlders, as in squibbing, causes violent vibrations in the mine air, which in turn stir up dust produced in other ways. Roof and pillar trimming is a minor cause of dust. If a piece of rock trimmed from the roof falls on a pile of dry dirt, some dust will be set in motion. Tramming raises appreciable quantities of dust only when the roadway is dry, the feet of the men, or mules, causing the dust to rise from the floor.

METHODS OF SAMPLING SILICEOUS DUST.

As regards the siliceous dust in the mine air, information was desired on the following points: Specific gravity of the dust particles; their size and physical and chemical characteristics; the weight of dust contained in a given volume of air; and the length of time that the dust would remain suspended.

It was a simple matter to obtain pieces of rock for the determination of specific gravity, and samples of drill cuttings for testing the physical and chemical characteristics of the rock particles. In order to determine the length of time the dust particles would remain in suspension, and the sizes of the dust particles, samples were collected at various periods of time by permitting dust to settle from the mine air on thin glass slides on which a drop of cedar oil had been placed. Cover glasses were then placed over the samples and they were set aside for examination under the microscope.

The apparatus and methods used in obtaining the samples of mine air and determining the weight of dust contained in a given volume have been described in Technical Paper 105.[a] Such work requires a suitable medium for filtering the dust-laden air, and a device that can be depended on to measure accurately the air passing through the filtering medium, simple in design, and not of great bulk.

In devising this apparatus devices that have been used for similar work in other countries were carefully studied. After conclusive experiments it was found that ordinary granulated sugar was entirely satisfactory as a filtering medium. Suction pumps, of the type used for pumping air into tires, no matter how well made, were found unsatisfactory for drawing a measured amount of air through the filtering bulb. With continued use these pumps became inaccurate on account of worn parts or from particles of dust getting into the valves and preventing them from seating properly. After various methods for measuring the air passing through the bulb containing the sugar had been considered, a device actuated by the breathing of the wearer was adopted, the volume of air breathed to be determined by receiving the exhaled air in a graduated receptacle.

The dust sampler, as finally worked out, is shown in Plates IV and V. Plate IV shows the apparatus assembled and Plate V shows the nose clip, mouthpiece, valves, and connections. The device, which has been described in Technical Paper 105 [b], consists essentially of the following parts:

A glass bulb or container for the filtering medium (4½ inches long, 1¼ inches inside diameter at the large end, and ½ inch inside diameter at the stem). This bulb is

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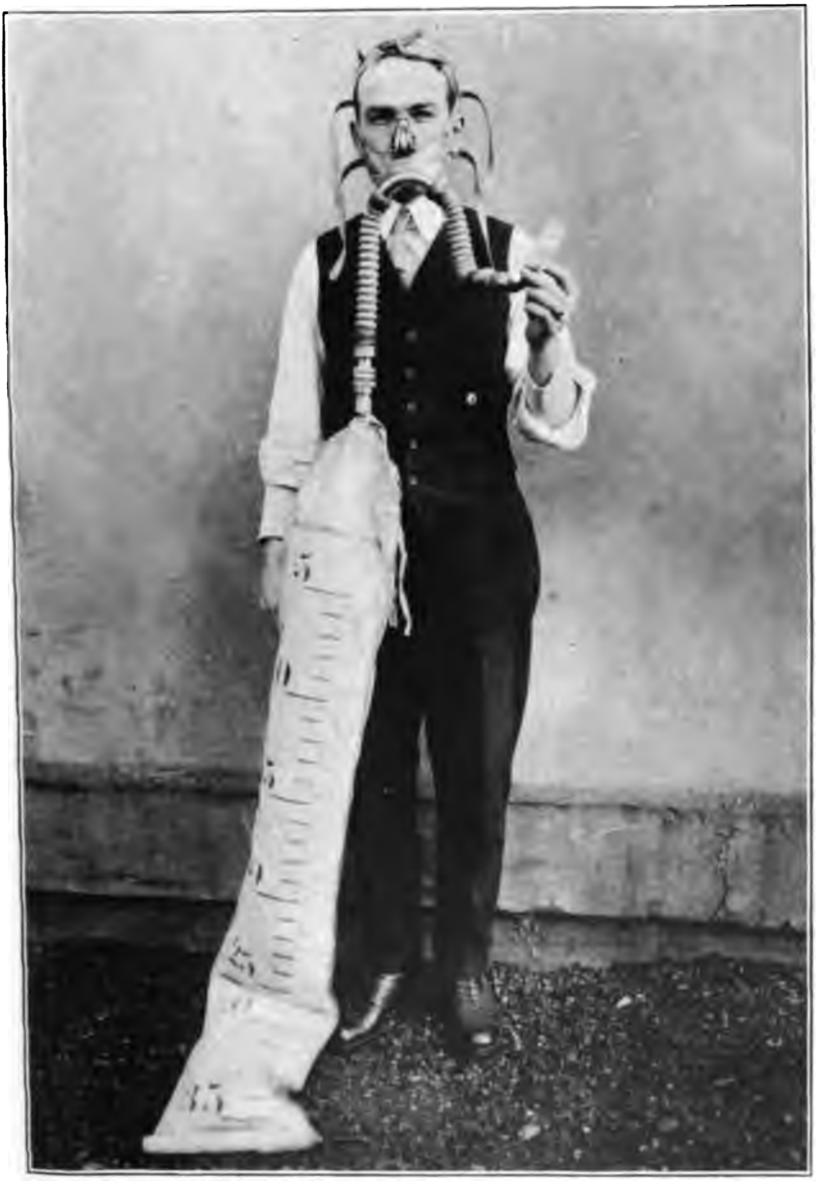
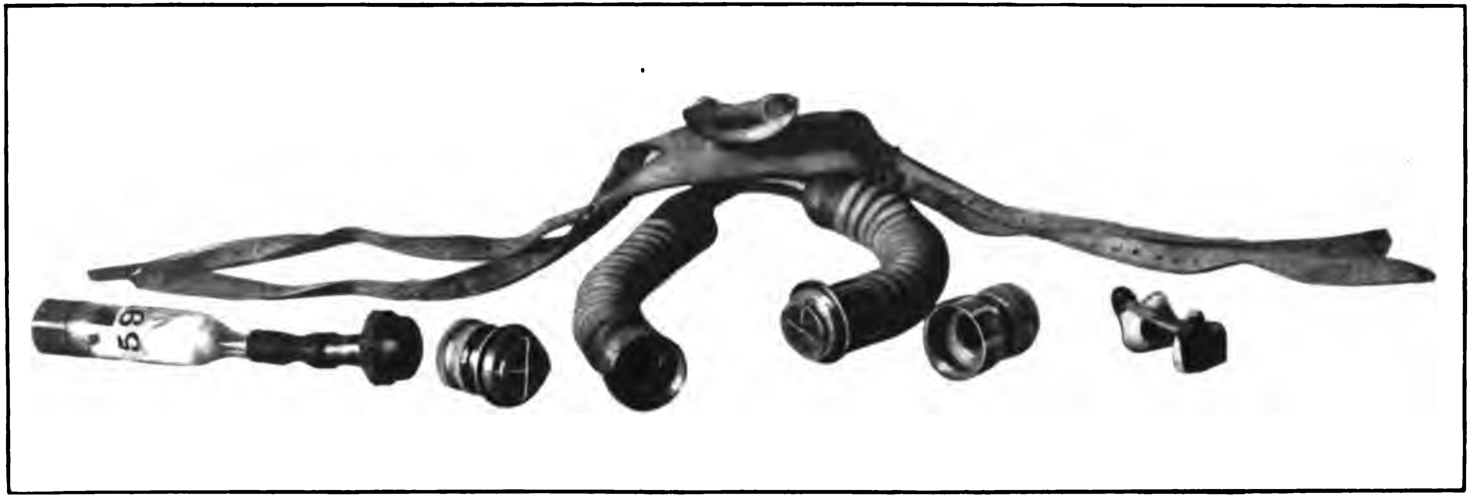
[a] Lanza, A. J., and Higgins, Edwin, Pulmonary disease among the miners in the Joplin district, Missouri, a preliminary report: Tech. Paper 105, Bureau of Mines, 1915, pp. 22 to 32.

[b] Lanza, A. J., and Higgins, Edwin, Work quoted, pp. 24-25.



APPARATUS FOR DETERMINING ROCK DUST IN MINE AIR.



DETAILS OF DUST-SAMPLING DEVICE.

partly filled with granulated sugar, which is prevented from passing into the sampling device by means of a perforated glass partition inserted just above the stem of the bulb, but at a point where the diameter is not less than 1 inch.

That part of the oxygen-breathing apparatus known as the mouthpiece, with inhalation and exhalation tubes attached. The mouthpiece may be held securely in position, when the device is in use, by means of a cap to which straps are attached. It is not necessary, however, to use a cap when one becomes familiar with the device.

A nose clip to prevent air from passing in or out of the nostrils.

What is known as a Draeger liter bag, having a capacity of 35 to 45 liters.

In order to collect a sample the apparatus is adjusted in the mouth with a dust bulb and liter bag attached. The nose clip is then placed on the nose and the person taking the sample breathes naturally. When air is inhaled the valve in the inhalation tube opens, allowing the air to pass to the lungs; at the same time the valve in the exhalation tube closes, making it impossible to draw air from the liter bag. When air is exhaled from the lungs the valve in the inhalation tube closes and that in the exhalation tube opens. In taking a sample, then, the air passes through the bulb (where the dust is intercepted by the sugar) into the lungs, and thence through the exhalation tube to the liter bag, where it is measured. Ordinarily, sufficient accuracy may be obtained by inhaling and exhaling a total of 35 liters of air. However, if the place in which the sample is taken is only slightly dusty the bag may be filled a second time, thus making a total of 70 liters of air for the sample.

In order to determine the amount of dust collected, the contents of the bulb are washed out into a weighed Gooch crucible and dried at 105° F. After successive washings with water, to dissolve and remove all of the sugar, the crucible is again dried at 105° C. and weighed. The increase in weight represents the weight of rock dust in the sample.

ACCURACY OF SAMPLING DEVICE.

In breathing a man takes a certain amount of oxygen from the air. It has been shown, by many analyses of exhaled air, that not all of the oxygen consumed is contained in the carbon dioxide given off. Under normal conditions, the ratio of carbon dioxide given off to oxygen consumed is approximately as 8 to 10, or 0.8. However, if a man exerts himself this ratio may rise to 0.9 or even 1 or slightly more. In these exceptional cases doubtless some residual carbon dioxide from the lower part of the lungs is exhaled. As exhaled air contains only 4 per cent carbon dioxide, if it be assumed that the ratio of oxygen converted into carbon dioxide to oxygen consumed is 0.8 to 1.0, the volume of the exhaled air would not be more than 1 per cent less than that of the inhaled air.[a] This difference is compensated for to some extent by the increase in temperature of the exhaled air. Owing to the variations in the factors concerned, as indicated above, no correction was made for the small error involved.

As to the velocity of the air entering the bulb, and the frequency of the inhalations, it is possible for the operator to imitate exactly the breathing of the miner at work. Thus, with the sampling bulb held near the head of the miner, the operation of this device in taking a sample is almost identical with that of the miner's respiratory organs. The only difference is that in the sampling device the dust is caught in the granulated sugar, whereas in breathing the dust lodges in the lungs and air passages leading to them.

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[a] 0.8 : 1 :: 4 : 5.

The fact that a large volume of air is taken in sampling (the liter bag may be filled as many times as deemed necessary) makes for accuracy. Although the liter bag is not elastic, there may be a small source of error in determining at just what point it should be considered to be full of air. This source of error, however, may be reduced to a minimum by practice in using the device. A sample may be taken in about two minutes, or the time may be extended by breathing alternately through the nose and through the device.

The liter bag may be tested for leaks by filling it with air and allowing it to stand for a few minutes; a leak will be indicated by loss of air. If the bag leaks, the exact location of the puncture or rent may be determined by the use of soap lather. A very small opening will cause bubbles to appear. The tubes leading to the mouthpiece may be tested for leaks by coating them with lather, holding the palm of the hand over the end of the exhalation tube and blowing through the mouthpiece. Bubbles will appear at leaky points. Thin sheet rubber, cemented on with rubber solution, makes a satisfactory patch; it is safer, however, to carry extra parts to replace those that develop leaks.

RESULTS OF THE DUST SAMPLING.

SIZING AND CHEMICAL TESTS.

Fifteen samples of drill cuttings made by drilling in the cherty flint were procured from various sheet-ground mines. The content of siliceous residue was determined in each sample; then sizing tests were made, after which the siliceous residue in that part of the sample passing through a 240-mesh screen was determined. Tests were also made to detect the presence or absence of carbonates and sulphides.

Table 4, which gives the results of the chemical analyses, shows that the content of siliceous residue in the average (unscreened) sample ranged from 70.5 to 95.3 per cent, 10 of the samples showing a content of over 90 per cent. As compared with the average samples, the determinations made on the portions passing a 240-mesh screen show more siliceous residue in some cases and less in others. All of the samples gave tests for carbonates, as indicated by effervescence with hydrochloric acid; seven showed the presence of sulphides, as indicated by the action of nitric acid.

TABLE 4.—*Results of chemical examination of drill cuttings.*

Laboratory No.	Sample No.	Siliceous residue.		Relative amount of carbonate as indicated by effervescense with hydrochloric acid.	Relative amount of sulphide present as indicated by action of nitric acid.
		Unscreened sample.	Through 240-mesh.		
		Per cent.	*Per cent.*		
20788	211	95.3	96.8	Medium	Nil.
20789	202	89.4	80.1	do	Do.
20790	215	70.5	61.3	Much	Much sulphide.
20791	422, 423	89.8	85.6	do	Little sulphide.
20792	424	91.7	93.2	Little	Nil.
20793	426	90.2	93.6	do	Much sulphide.
20794	566, 575	90.8	88.8	Medium	Do.
20795	567	94.9	95.3	do	Nil.
20796	569	95.2	97.0	Little	Do.
20797	570	94.9	96.2	Medium	Do.
20798	571, 572	86.8	83.6	Little	Little sulphide.
20799	576, 574	78.8	75.9	do	Do.
20800	580	91.2	90.6	do	Nil.
20801	590	94.8	94.6	do	Little sulphide.
20802	591	90.8	93.7	do	Nil.

Table 5 shows the results of the sizing tests. The variation in the percentage of drill cuttings passing through a 240-mesh screen indicates a variation in the dust-producing tendency of the chert in different mines. The sizes of the openings in the screens used in the sizing tests were as follows:

Standard screens used for sizing tests.

Mesh of screen.	Size of opening.		Mesh of screen.	Size of opening.	
	Inches.	*Mm.*		*Inches.*	*Mm.*
10	0.065	1.651	100	0.0058	0.147
20	.0328	.833	150	.0041	.104
35	.0164	.417	200	.0029	.074
65	.0082	.208	240	.0026	.066

TABLE 5.—*Results of sizing tests on drill cuttings.*

Laboratory No.	Sample No.	On 10-mesh.	On 20-mesh.	On 35-mesh.	On 65-mesh.	On 100-mesh.	On 150-mesh.	On 200-mesh.	On 240-mesh.	Through 240-mesh.
		P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
20788	211	11.2	9.3	14.9	40.5	5.2	6.7	4.9	3.4	3.9
20789	202	12.0	12.5	17.5	15.3	19.1	4.5	7.2	2.2	9.7
20790	215	12.3	5.5	12.4	17.9	11.0	9.0	9.2	7.5	15.2
20791	422, 423	5.9	5.6	12.8	21.7	12.2	9.6	6.5	4.6	21.1
20792	424	34.6	15.8	16.0	11.1	5.8	3.5	2.5	.5	10.2
20793	426	20.4	12.2	14.4	14.0	8.8	7.5	4.0	.1	18.6
20794	566, 575	10.8	7.3	16.7	19.8	10.3	7.9	8.0	.1	19.1
20795	567	57.2	14.0	10.9	6.6	3.0	1.8	1.0	.1	5.4
20796	569	16.7	21.3	36.5	13.1	3.4	1.9	1.0	.1	6.0
20797	570	3.9	3.6	13.0	26.8	16.5	8.2	5.9	.4	21.7
20798	571, 572	7.3	6.5	10.4	15.1	10.7	9.1	7.0	.2	33.7
20799	576. 574	11.4	4.5	10.1	19.4	11.1	7.6	5.2	.6	30.1
20800	580	16.5	12.6	18.0	14.6	9.3	5.8	3.9	.5	18.8
20801	590	12.1	10.0	15.1	17.2	10.5	6.0	3.6	.7	24.8
20802	591	5.2	5.3	9.9	20.6	11.5	9.3	9.2	2.5	26.5

MICROSCOPIC EXAMINATION OF DUST PARTICLES.

Portions of the samples that passed the 240-mesh screen in the sizing tests were examined under the microscope by Reinhardt Thiessen, who prepared the accompanying photomicrographs (Pls. VI to VIII).

About 0.5 milligram of each sample examined was mixed or stirred into a drop of cedar oil, placed on a glass slide, and observed under the microscope. The results of Dr. Thiessen's observations, which have been given in Technical Paper 105, are repeated here.

GENERAL DESCRIPTION.

In size the dust particles varied from those averaging about 90 microns[a] down to particles that were invisible under the ordinary microscope. Some of the samples were made up chiefly of the smaller sizes, whereas others consisted mostly of the larger and intermediate sizes.

Although the larger sizes measured approximately 100 to 60 microns, there were some longer and narrower knifelike or needle-shaped particles. The smallest dimension measurable by the ordinary microscope is one-fourth to one-half micron. The smallest particle visible was about 20 millimicrons, or twenty one-thousandths of a micron.

PHYSICAL CHARACTER OF DUST PARTICLES.

In the main two kinds of particles are visible, one having a milk-white or porcelainlike look and the other being more glassy. The milky particles are generally thin, sharp edged, and irregular, often knifelike or spear point like in shape. The glassy particles are usually roughly polyhedral or spheroidal, although some are sharp edged or pointed. Of the other particles visible, some are perfect crystals, evidently calcite; others are sphalerite, galena, and pyrite. The particles are of almost every possible shape and form. The milk-white, sharp-edged particles are usually thin and scalelike; comparatively few are rounded or polyhedral. The glassy particles, on the other hand, are usually rounded or tetrahedral.

DESCRIPTION AND DISCUSSION OF PHOTOMICROGRAPHS.

In the following descriptions and discussions of the photomicrographs shown in Plates VI to VIII, and IX, *A*, the sample numbers used in Tables 4, 5, and 7 are retained.

Sample 215 (*Lab. No. 20790*).—The gradation in size is gradual. The largest particles measure about 85 microns (0.085 mm.). Sharp, angular, and irregular particles predominate.

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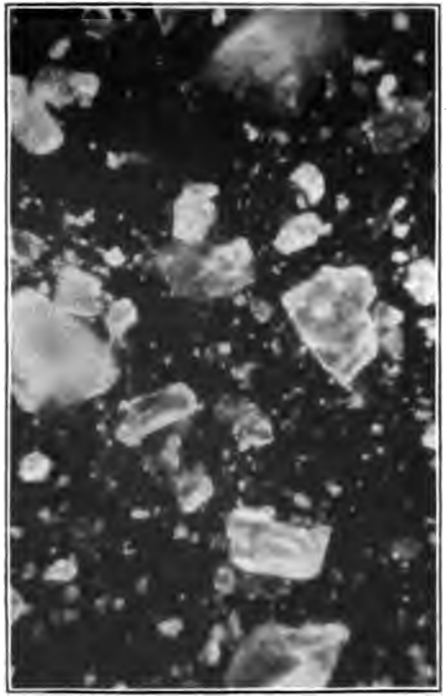
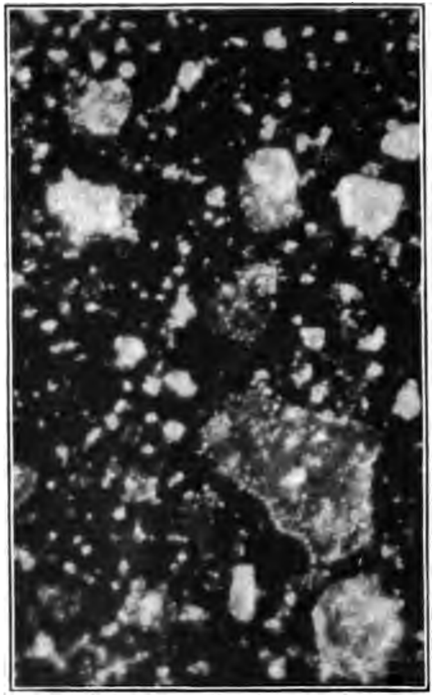
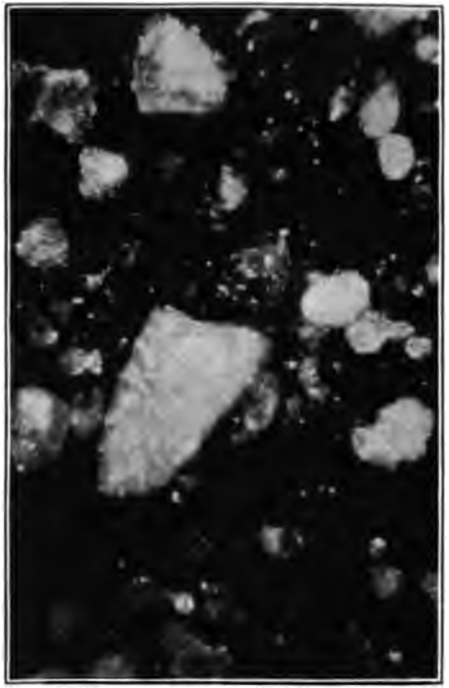
[a] 1 micron=one one-thousandth millimeter.



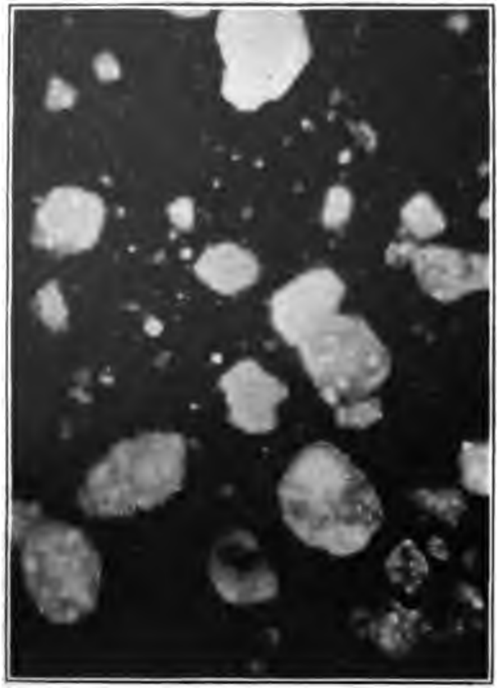
A. ROCK DUST FROM SAMPLE 215, PARTICLES MAGNIFIED 250 DIAMETERS.



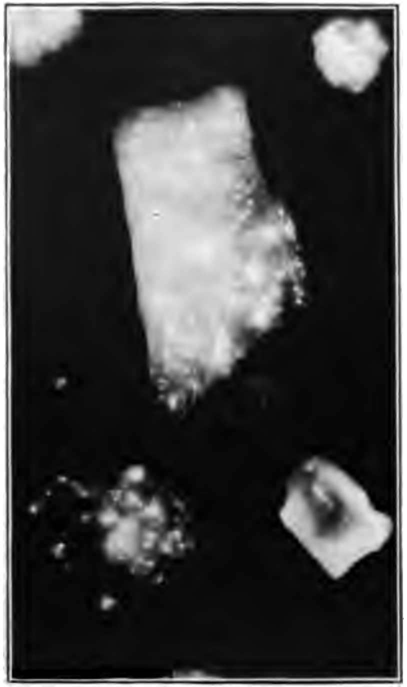
B. ROCK DUST FROM SAMPLE 215, PARTICLES MAGNIFIED 500 DIAMETERS.



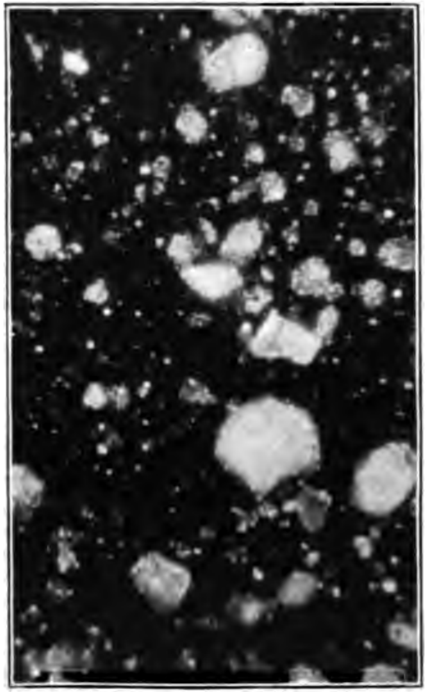
C. ROCK DUST FROM SAMPLE 424, PARTICLES MAGNIFIED 250 DIAMETERS.



A. ROUNDED AND POLYHEDRAL PARTICLES FROM SAMPLE 424, MAGNIFICATION 200 DIAMETERS.



B. PARTICLES FROM SAMPLE 566, MAGNIFICATION 500 DIAMETERS.



C. ROCK DUST FROM SAMPLE 591, PARTICLES MAGNIFIED 250 DIAMETERS.

In Plate VI, *A*, magnification 250 diameters, the particle at the right is approximately 85 microns long and 48 microns wide. The smaller particles, represented by spots, are less than 1 micron. In Plate VI, *B*, magnification 500 diameters, the largest particle is approximately 56 microns in diameter.

Sample 424 (*Lab. No. 20792*).—This sample has a large proportion of the coarser particles and relatively very few small particles. Both the milky-white, irregular, sharp-edged particles and the glassy, irregular, spherical particles are shown.

Plate VI, *C*, magnification 250 diameters, shows the abundance of the large particles, and some of the milky-white, sharp-edged ones, which are very thin and spadelike.

Plate VII, *A*, magnification 200 diameters, shows the rounded and polyhedral particles.

Sample 566 (*Lab. No. 20794*).—This sample is characterized by rather sharp, angular particles, with a large number of particles ranging from medium size down to invisible—that is, from 25 microns in diameter to about 1 micron—in which the milky-white or porcelainlike are the most abundant.

The porcelainlike particles vary widely in form. Besides the sharp knifelike or angular particles, rhomboidal, tetrahedral, and even globular forms, not entirely restricted to this sample, yet more common than in others, are present.

In Plate VII, *B*, magnification 500 diameters, the clear rhombohedral particle in the lower right-hand corner is about 24 microns long, 20 microns wide, and 10 microns thick. The thin platelike particle in the lower left-hand corner, almost clear on the left-hand half, but containing millions of minute inclusions in the right half, is about 36 microns long and wide, but probably less than 0.5 micron thick. The sharp-pointed particle in the middle has a knifelike edge on the left, but is rather thick on the right.

Sample 567 (*Lab. No. 20795*).—The gradation ranges from the larger particles down to those that are invisible. The smaller ultramicroscopic particles are more abundant than in any of the other photomicrographs shown. There are also more sharp-edged particles, with even sharper edges than in any other sample. In the other samples the scalelike, sharp-edged particles are confined to the larger and medium sizes, but in these samples grade into the invisible. Two kinds of particles predominate—the milky white, which comprise the bulk of the larger sizes, but are not confined to them, and the glassy particles, which are mostly rounded, although many are sharp-edged. The milky-white particles are almost always thin, bladelike, leaflike, pointed, and some even needlelike.

In Plate VIII, *A*, the finer particles were removed in order to show the larger ones, which are angular and sharp edged. All of the particles are porcelainlike. The length of the long slender particle is approximately 230 microns and its least breadth is little more than 8 microns.

Sample 576 (*Lab. No. 20799*).—In this sample there is an almost complete absence of the milky-white material, the particles being glassy. (See Pl. VIII, *B*.) There are comparatively few large particles, the gradation being uniform.

Plate VIII, *C*, magnification 400 diameters, shows one of the largest particles that could be found in the preparation. Its largest dimension is about 80 microns. This photograph shows the relatively large proportion of the finest dust particles.

Samples 590 and 591 (*Lab. Nos. 20801 and 20802*).—These samples are composed largely of rather spheroidal or roughly polyhedral particles varying from the largest to the smallest visible. There are few milky-white particles present and also few sharp or knifelike particles.

Plate VII, *C* (No. 591), shows the general distribution and comparative sizes. The largest one is approximately 60 microns in diameter. The spherical or oval form is also shown.

Plate IX, *A* (No. 590), shows the general nature of the particles. The groundmass is clear isotropic to milky white and contains numerous finer particles, the milkiness being due to a suspension of still finer particles.

SUSPENSION OF DUST PARTICLES IN THE AIR.

A large proportion of the dust particles are less than 1 micron in diameter. How long such small particles will remain suspended in dry still air can be roughly determined from Stoke's law, the formula for which is:

$$V=\frac{2r^2\,(s-s')g}{9z}$$

V=the velocity of fall in seconds per centimeter;
r=the radius of the particle in suspension;
s=the specific gravity of the particle;
s'=the specific gravity of the medium in which it is suspended;
g=the gravity constant;
z=the viscosity coefficient of the suspending medium.

If it be assumed that a certain round particle of quartz (specific gravity 2.62) is 1 micron (or 0.0001 centimeter) in diameter, and that the air (temperature 60° F.) is free from moisture, substituting values gives:

$$V=\frac{2\times 0.00005^2(2.62-0.0012)980}{9\times\frac{1970}{107}}=\frac{2\times 0.0000000025\times(2.62\times 980)}{9\times 0.0001970}=0.0072$$

centimeter per second=0.432 centimeter per minute=25.92 centimeters per hour=620.08 centimeters, or 6.2 meters (20.34 feet), per day.

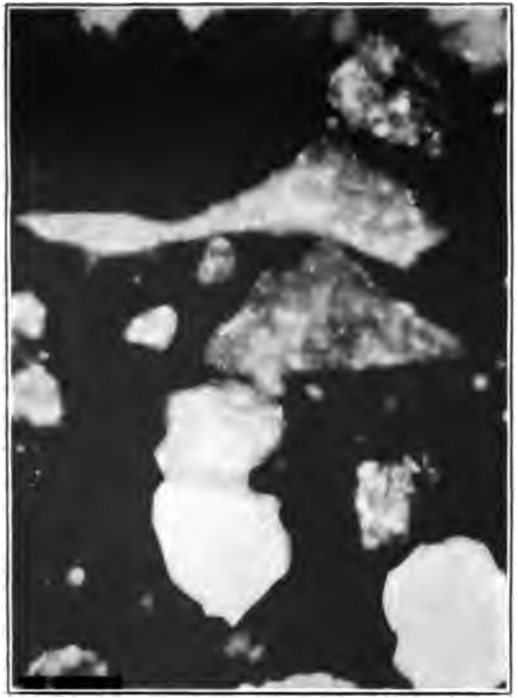
Suppose that a particle is 0.2 micron in diameter, then

$$V=\frac{2\times 0.000001^2\times 2.62\times 980}{9\times 0.000197}=0.000028$$ centimeter, or 0.00028 millimeter, per second=0.0168 millimeter per minute=1 millimeter per hour=24 millimeters (0.94 inch) per day.

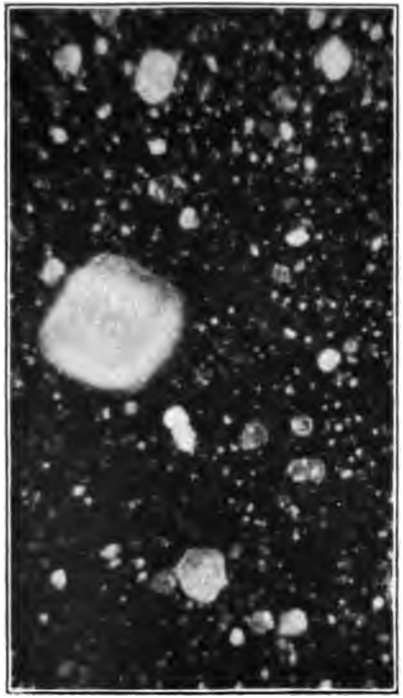
Although these theoretical calculations are of interest, it is obvious that in any attempt to determine the length of time dust particles will remain in suspension in mine air it is necessary to give proper consideration to air currents and to humidity.

MICROSCOPIC EXAMINATION OF THE PARTICLES CAUGHT ON SLIDES.

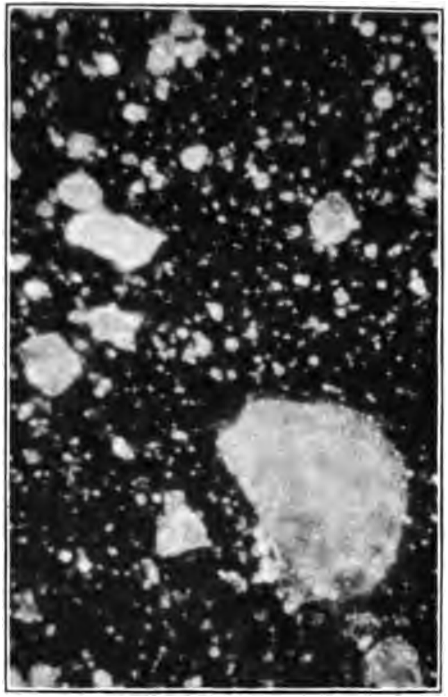
The samples that were obtained by catching dust in cedar oil on glass slides for the purpose of determining the sizes of the particles suspended in the mine air, and the length of time they would remain in suspension, were examined by Dr. Thiessen. Owing chiefly to the transparent nature of the particles, especially the smaller ones,



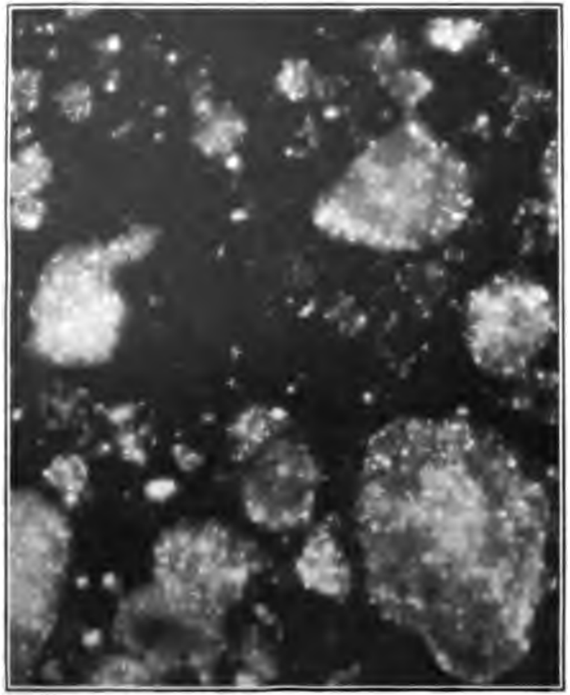
A. LARGER PARTICLES IN ROCK DUST FROM SAMPLE 567, MAGNIFICATION 250 DIAMETERS.



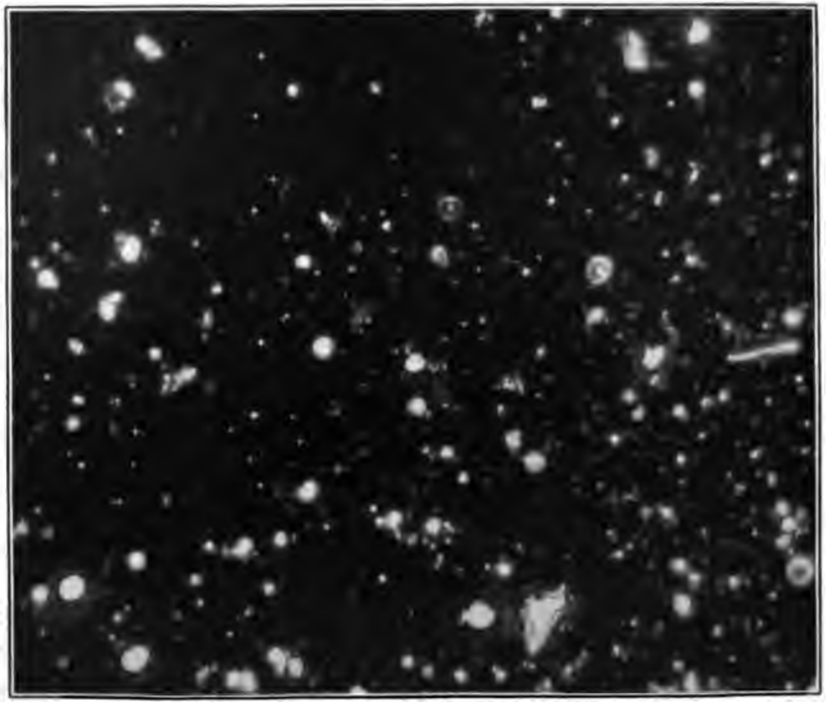
B. ROCK DUST FROM SAMPLE 576, MAGNIFICATION 250 DIAMETERS.



C. ROCK DUST FROM SAMPLE 576, SHOWING RELATIVELY LARGE PROPORTION OF FINER PARTICLES, MAGNIFICATION 400 DIAMETERS.



A. ROCK DUST PARTICLES FROM SAMPLE 590, MAGNIFIED 560 TIMES.



B. DUST FROM MINE AIR, CAUGHT ON BALSAM-COVERED PLATE, MAGNIFIED 500 TIMES.

the photomicrographs prepared from these slides, with the exception of sample 14, shown in Plate IX, *B*, were not distinct enough for the preparation of halftone reproductions. From Dr. Thiessen's report and a study of the photomicrographs, some data were obtained as to the size of the suspended dust particles and the length of time they will remain in suspension. This information is set forth in Table 6. One column shows the dimensions of the largest dust particle on the slide; in most of the slides there was only one nearly as large as indicated. Too much importance, therefore, should not be given to this figure.

TABLE 6.—*Results of tests to show size of dust particles in mine air and duration of suspension.*

Slide No.	Mine.	Date (1915).	Length of time exposed, minutes.	Cause of dust.	Distance sample was taken from source of dust, feet.	Lapse of time between production and sampling of dust.	Size of largest dust particle on slide, microns.[a]	Remarks.[b]
1	O	April 28	10	Drilling and shoveling	100	None	20 by 25	Few particles larger than 10 microns, majority less than 5 microns.
2	O	do	10	do	30	do	20 by 25	Same as No. 1.
3	O	do	18	do	30	do	20 by 17	Majority of particles less than 5 microns.
4	O	do	10	do	30	do	50 by 70	Larger percentage over 10 microns.
5	H	April 30	10	do	50	do	10 by 10	Nearly all less than 5 microns.
6	H	do	10	do	125	do	50 by 55	More larger particles than in Nos. 3 and 5.
7	H	do	10	do	125	do	50 by 40	Similar to No. 6.
8	H	do	10	do	125	do	10 by 10	Many particles between 5 and 10 microns.
9	G	May 1	10	Blasting		45 minutes	10 by 10	Few particles larger than 5 microns.
11	G	do	10	do		1 hour		Do.
12	G	do	10	do		1 hour 20 minutes	10 by 8	Do.
14	G	do	10	do		1 hour 40 minutes	10 by 5	Increasing number of particles from 1 to 2 microns.
16	G	do	60	do		2 hours	10 by 5	Similar to No. 14, but more particles of smaller sizes.
17	G	do	10	do		2 hours 30 minutes	9 by 6	Similar to No. 16.
18	G	do	20	do		3 hours	5 by 5	Gradual decrease in sizes.
19	G	do	60	do		do		Similar to Nos. 17 and 18, but more smaller particles.
20	G	do	20	do		3 hours 40 minutes		Large number of particles still smaller than in No. 19.
21	G	do	60	do		4 hours		Similar to No. 2, size decreasing and number greatly increased.
22	G	do	20	do		4 hours 20 minutes	5 by 3	Countless numbers of particles from 1 micron to submicroscopic size.

[a] 1 micron=0.001 millimeter.

[b] The number of microns refers to the greatest dimension.

The physical characteristics of the dust particles are practically the same as indicated on pages 25 and 26. In all cases the places where the samples were taken were poorly ventilated; air currents were sluggish or entirely lacking.

Slides 1 to 8 represent general conditions underground from 30 to 125 feet from the working face.

Slides 9 to 22 were taken at various intervals after blasts in order to determine the length of time the particles of various size would remain suspended in the mine air. The temperature where these exposures were made was 62° F. and the relative humidity ranged from 95 to 100 per cent. Most of the time there was visible water vapor in the air, and this doubtless hastened the settling of the dust particles. The photomicrographs show a gradual decrease in the size of the particles, and a corresponding increase in the percentage of small particles present toward the end of this experiment. None of the photomicrographs shows any great number of particles larger than 5 microns, those as large as 10 microns being only occasionally seen. On slides 17 to 22 there appeared countless numbers of particles 0.1 to 0.2 microns in diameter. It is worthy of note that samples obtained with the mouth-breathing apparatus at the time slides 21 and 22 were prepared showed no dust. This is not to be wondered at, as the dust particles were almost submicroscopic in size.

From these experiments it appears that dust particles with dimensions of 50 to 70 microns, and more, will settle within a few minutes. However, where mining operations that make dust are continuous, these larger sizes may be found in the air all of the time. One hour after the cessation of the work causing the dust, few particles larger than 10 microns will remain in suspension, if the air is comparatively still, the size gradually decreasing up to three hours, when the largest particle to be found will be about 5 microns in greatest dimension. After four hours few particles as large as 5 microns are to be found; a small percentage will range from 1 to 2 microns, but the bulk of the particles will range from 1 micron down to submicroscopic sizes.

SIZE OF DUST PARTICLES TAKEN INTO THE LUNGS.

In the course of the investigative work carried on by Dr. Lanza, prior and subsequent to his preliminary report,[a] many samples of sputum were collected from miners who reported for examination. These samples were centrifuged and small portions placed on glass slides, to be examined under the microscope for the presence of tubercle bacilli. The writer had an opportunity to examine a great number of these slides. Siliceous dust, in varying quantities, was present on all of them; the cherty flint particles were plainly in evidence, and in some cases scattering particles of zinc blende and galena were recognized.

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[a] Lanza, A. J., and Higgins, Edwin, Pulmonary disease among miners in the Joplin district, Missouri, and its relation to rock dust in the mines: Tech. Paper 105, Bureau of Mines, 1915, 48 pp.

Seven of these slides were examined by Dr. Thiessen. Following is only an abstract of that part of Dr. Thiessen's report concerning the siliceous dust on these slides:

> All of the slides contain, besides the rock dust, dust of organic origin, though relatively little. The rock dust is of the same nature and appearance as that gathered in tests of the suspension of dust particles. The sizes of the particles are also approximately the same as those discussed under "Suspension." There are comparatively few particles measuring more than 10 microns in diameter, but a large number measure between about 5 and 2 microns, with rapidly increasing numbers of smaller sizes, down to particles of colloidal dimensions. It is impossible to make a fair estimate of the proportion of the smaller particles to the larger, on account of the colloidal matter present in the preparations. The amount and size of particles on the sputum slides compare favorably with photomicrograph 14 (Pl. IX, *B*), of the suspension tests.

Thus it appears that comparatively few particles larger than 10 microns in diameter reach the lungs. Without question, however, much larger particles find lodgment in the nasal and other air passages leading to the lungs. When a dry hole is blown, or a drill hole is being started, the air about the hole is filled with large particles of dust, and even chips and splinters of rock. Under such conditions much of this coarse material gets inside the nostrils, or into the mouth. Miners working in very dusty places frequently can detect grit in the mouth.

WEIGHT OF DUST IN MINE AIR.

For the purpose of determining the weight of siliceous dust in mine air under varying conditions, a total of 222 samples were collected with the mouth-breathing device. These samples are described in Tables 7 and 8. The 39 samples included in Table 7 were taken during the preliminary investigation; those in Table 8, taken subsequently, are more completely described.

TABLE 7.—*Analyses of 39 samples of siliceous rock dust taken in sheet-ground mines of the Joplin district, Mo., in November, 1914.*

[All holes drilled horizontally; all samples, except where noted, taken close to nostrils of drill man or shoveler.]

Sample No.	Laboratory No.	Name of mine.	Remarks.	Weight of dust per 100 liters of air, milligrams.
202	20453	Ice Plant	Drilling breast hole, dry	6.57
203	20454	do	Taken after squibbing	4.86
209	20455	do	do	6.00
210	20456	Gopher	Shoveling	3.71
593[a]	20457	do	do	4.57
587	20488	do	do	4.86
211	20458	Little Princess	Drilling and shoveling, 10 feet away	4.00
212[a]	20459	do	do	5.14
212A	20460	Florine	Drilling and shoveling	5.14
216[a]	20461	do	do	5.71
215	20462	Oronogo Circle	Drilling, dry	3.71
222	20463	Century	Five men shoveling, 5 to 15 feet away	4.29
564[a]	20464	do	do	3.71
419	20465	Davey No. 3	Drilling breast hole, dry	6.86
420	20466	do	do	5.14
421	20467	do	Blowing hole, damp	4.29
424	20470	do	Drilling stope hole, damp	.57
426	20472	do	Taken after squibbing	4.29
423	20469	Coahuilla	Drilling breast hole, dry	4.00
574	20483	do	Shoveling	3.60
576	20485	do	do	1.14
425	20471	Wilson	Shoveling and roof trimming	2.29
573	20482	do	Shoveling	.57
571	20480	do	Drilling breast hole, dry	3.60
572	20481	do	Blowing hole, dry	7.71
563	20473	Schoolhouse	Taken 5 minutes after blowing hole, dry	3.14
567	20476	do	Taken after squibbing	7.16
567A[b]	20477	do	do	5.71
566	20475	Wingfield	Drilling breast hole, dry	3.14
575[a]	20484	do	do	3.14
569	20478	Bertha A	do	6.29
570	20479	do	do	6.00
580	20486	Otis	Drilling stope hole, damp	2.86
578	20487	do	Drilling breast hole, dry	6.00
590	20489	A. W. C. No. 1	Drilling and shoveling	6.57
596	20490	do	Shoveling	3.60
591	20491	Little Dorothy	Drilling in heading, dry	4.57
592	20492	Electrical	Taken one-half hour after five holes had been squibbed	4.00
583[a]	20493	do	do	4.86

[a] Duplicate of preceding sample.
[b] Taken 3 minutes after sample 567 was taken.

TABLE 8.—*Description of siliceous dust samples taken*

Sample No.	Bulb No.	Mine.	Date (1915).	Time.	Where taken.	Work progressing.	Position of bulb.
40	571	A	Feb. 18	*a. m.* 9.20	Face northwest of shaft.	Drilling breast hole	At drillman's head.
41	593	A	do	9.35	do	do	do
42	574	A	do	9.50	do	do	do
43	576	A	do	10.05	do	do	do
44	576A	A	do	10.20	do	do	do
45	222	A	do	10.25	do	do	1 foot to right and 1 foot below hole.
46	531	A	do	10.35	do	do	At drillman's head.
47	532	A	do	10.50	do	do	do
48	583	A	do	11.05	do	do	do
49	592	A	do	*p. m.* 2.30	30 feet from west face.	Drilling at face	5 feet from floor
50	540	A	do	2.45	do	do	do
51	539	A	do	3.00	do	do	do
52	538	A	do	3.15	do	do	do
53	541	A	do	3.30	do	do	do
54	537	A	do	3.45	do	do	do
55	203	A	Feb. 19	*a. m.* 9.00	Face northwest of shaft.	Drilling breast hole	At drillman's head.
56	596	A	do	9.15	do	do	do
57	572	A	do	9.30	do	do	do
58	580	A	do	9.45	do	do	do
59	566	A	do	10.00	do	do	do
60	587	A	do	10.15	do	do	do
61	573	A	do	10.30	do	do	do
62	294	B	Feb. 25	*p. m.* 2.45	Stope No. 3	Shoveling	At shoveler's head.
63	569	B	do	3.00	Stope No. 10	do	do
64	278	B	do	3.15	Stope No. 4	Shoveling and drilling.	At drillman's head.
65	324	B	Feb. 26	*a. m.* 9.15	Stope No. 12	Shoveling	5 feet from shoveler.
66	323	B	do	9.30	Stope No. 2	do	do
67	295	C	do	10.10	East face	Drilling breast hole	10 feet from drillman.
68	565	C	do	10.30	Northwest face	Drilling and shoveling.	At drillman's head.
69	530	C	do	11.00	do	do	40 feet from face
70	293	C	do	*p. m.* 2.15	No. 3 north heading.	Drilling breast hole	At helper's head.
71	536	C	do	2.30	No. 1 east heading.	do	At drillman's head.
72	292	C	do	2.45	East face	Shoveling	At shoveler's head.
73	209	A	Mar. 1	*a. m.* 9.45	Headframe	Dumping bucket	Beside hoistman
74	212	A	do	10.00	do	do	do
75	334	A	do	10.15	do	do	do
76	291	A	do	10.30	do	do	do
77	550	A	do	10.45	do	do	do
78	336	A	do	11.00	do	do	do

in sheet-ground mines of the Joplin district.

Distance of bulb from source of dust.	Air circulation.	Condition of working place.	Temperature.		Relative humidity.	Time consumed in taking sample.	Air taken.	Dust in sample.	Dust per 100 liters of air.	Remarks.
			"Wet bulb."	"Dry bulb."						
Ft.			*°F.*	*°F.*	*P. ct.*	*Min.*	*Liters*	*Mg.*	*Mg.*	
8	Poor..	Wet....	59.5	61	92	2	35	1.00	2.86	Samples 40 to 48 taken to show quantity of dust made drilling with piston drill and no water.
8	Poor..	Wet....	59.5	61	92	2	35	1.00	2.86	
8	Poor..	Wet....	59.5	61	92	2	35	.80	2.28	Changed drill steel.
8	Poor..	Wet....	59.5	61	92	2	35	.70	2.00	12-foot hole completed 10.10 a. m.
8	Poor..	Wet....	59.5	61	92	2	35	4.50	12.86	Started new hole 10.15 a. m.
1½	Poor..	Wet....	59.5	61	92	2	35	77.60	221.71	Taken to compare quantity of dust closer to hole.
8	Poor..	Wet....	59.5	61	92	2	35	.80	2.28	
8	Poor..	Wet....	59	61	89	2	35	.20	.57	
8	Poor..	Wet....	59	61	89	2	35	.20	.57	
30	Fair...	Dry....	58	61	84	2	35	.70	2.00	Samples 49 to 54 taken to determine average quantity of dust in air when no water was used. Compare with samples 79 to 85. 2 machines working at face; no shovelers near.
30	Fair...	Dry....	58	61	84	2	35	1.20	3.43	
30	Fair...	Dry....	58	61	84	2	35	1.30	3.71	
30	Fair...	Dry....	58	61	84	2	35	1.30	3.71	
30	Fair...	Dry....	58	61	84	2	35	.60	1.71	
30	Fair...	Dry....	58	61	84	2	35	1.00	2.86	
8	Poor..	Wet....	59.5	61.5	89	2	35	.50	1.43	Compare samples 55 to 61 with Nos. 40 to 48; conditions same except that water spray was used in hole at intervals when samples 55 to 61 were taken. New hole started 9.05 a. m.
8	Poor..	Wet....	59.5	61.5	89	2	35	.80	2.28	
8	Poor..	Wet....	59.5	61.5	89	2	35	.60	1.71	
8	Poor..	Wet....	60	61.5	89	2	35	.40	1.14	
8	Poor..	Wet....	60	61.5	94	2	35	.00	.0	Drill sticking at intervals.
8	Poor..	Wet....	60	61.5	94	2	35	.00	.0	Do.
8	Poor..	Wet....	60	61.5	94	2	35	.10	.29	Do.
3	Poor..	Wet....	60	61.5	94	5	100	.70	.70	Drill working in heading, 20 feet above point of sampling.
3	Poor..	Damp..	60.5	62	92	4	70	2.10	3.00	Some dust made by dirt coming down from heading.
6	Poor..	Wet....	60.5	62	92	4	70	......	....	Bulb broken.
7	Poor..	Damp..	60.5	62	92	5	90	.70	.80	
7	Poor..	Damp..	61	62	94	5	90	.50	.60	
(a)	Poor..	Dry....	58	61	84	2½	45	1.40	3.11	2 shovelers also working within 10 feet of sampler.
8	None..	Wet....	60	61.5	92	2½	45	1.10	2.44	Drill stopped frequently; 2 shovelers working within 10 feet of sampler.
(a)	Poor..	Wet....	60	61.5	92	2½	45	.50	1.11	
3	Poor..	Wet....	60	61.5	92	2½	45	.80	1.78	
8	Poor..	Wet....	60	61.5	92	2½	45	.20	.45	
3	None..	Damp..	59	61.5	89	2½	45	.70	1.55	
6	Good..	Damp..	35	37	83	3	45	.90	2.00	Samples 73 to 78 taken to determine quantity of dust made in dumping bucket at top of derrick. 2 men were breaking large bowlders on grizzly, 6 feet below point of sampling.
6	Good..	Damp..	35	37	83	3	45	.70	1.55	
6	Good..	Damp..	35	37	83	3	45	.50	1.11	
6	Good..	Damp..	35	37.5	87	3	45	.60	1.33	
6	Good..	Damp..	35	37.5	87	3	45	.60	1.33	
6	Good..	Damp..	35	37.5	87	3	45	.40	.90	

[a] Bulb held at various distances.

TABLE 8.—*Description of siliceous dust samples taken*

Sample No.	Bulb No.	Mine.	Date (1915).	Time.	Where taken.	Work progressing.	Position of bulb.
79	301	A	Mar. 1	*p. m.* 2.15	30 feet from west face.	Drilling at face....	5 feet from floor.....
80	325	A	...do.....	2.30	.....do.............	.....do.............	.....do.............
81	335	A	...do.....	2.45	.....do.............	.....do.............	.....do.............
82	303	A	...do.....	3.00	.....do.............	.....do.............	.....do.............
83	549	A	...do.....	3.15	.....do.............	.....do.............	.....do.............
84	302	A	...do.....	3.30	.....do.............	.....do.............	.....do.............
85	548	A	...do.....	3.45	.....do.............	.....do.............	.....do.............
86	958	D	Mar. 3	*a. m.* 9.15	Point A..........	General...........	5 feet from floor.
87	960	D	...do.....	9.30	Point B..........	.....do.............	.....do.............
88	297	D	...do.....	9.45	Point A..........	.....do.............	.....do.............
89	345	D	...do.....	10.00	Point B..........	.....do.............	.....do.............
90	957	D	...do.....	*p. m.* 2.15	Point A..........	.....do.............	.....do.............
91	954	D	...do.....	2.25	Face, near point A.	Blowing hole dry..	.....do.............
92	955	D	...do.....	2.30	Point B..........	General...........	.....do.............
93	959	D	...do.....	2.45	Point A..........	.....do.............	.....do.............
94	315	D	...do.....	3.00	Point B..........	.....do.............	.....do.............
95	951	D	...do.....	3.15	Point A..........	.....do.............	.....do.............
96	211	D	...do.....	3.30	Face, near point A.	Blowing damp holes.	.....do.............
97	304	E	Mar. 5	*a. m.* 9.30	South face, machine 5.	Drilling and shoveling.	15 feet from drillmen and shovelers..
98	305	E	...do.....	9.50	South face.........	Blowing damp holes.	5 feet from floor.
99	306	E	...do.....	10.15	125 feet from south face.	None.............	.....do.............
100	307	E	...do.....	11.00	North face........	Drilling breast hole.	3 feet from hole.
101	308	E	...do.....	*p. m.* 2.25	200 feet from north face.	None.............	5 feet from floor.
102	309	E	...do.....	2.30	North face........	Blowing wet stope hole.	.....do.............
103	313	E	...do	2.55	South face........	Starting stope hole	.....do.............
104	317	E	...do.....	3.00	60 feet from south face.	None.............	.....do.............
105	547	E	...do.....	3.20	250 feet from south face.	.....do.............	.....do.............
106	542	F	Mar. 8	*a. m.* 10.30	Face northeast of north shaft.	.....do.............	.....do.............
107	543	F	...do.....	10.45	Face northwest of north shaft.	Drilling breast hole.	5 feet from hole and floor.
108	544	F	...do.....	11.00	Face west of north shaft.	Shoveling.........	At shoveler's head..
109	545	F	...do.....	*p. m.* 1.00	.....do.............	.....do.............	.....do.............
110	546	F	...do.....	1.40	Machine No. 8.....	Drilling roof hole..	3 feet from floor.
111	953	F	...do.....	2.00	Machine No. 5.....	.....do.............	5 feet from floor.....
112	563	F	...do.....	2.15	.....do.............	.....do.............	.....do.............
113	590	F	...do.....	2.45	.....do.............	.....do.............	.....do.............
114	210	F	...do.....	3.00	Face northwest of north shaft.	.....do.............	.....do.............
115	985	G	Mar. 10	2.30	Drift 14 east.......	.....do.............	4 feet out from hole.
116	986	G	...do.....	2.45	Drift 9 east........	Shoveling.........	5 feet from floor.....
117	988	G	...do.....	3.05	Drift 5 east........	None.............	.....do.............
118	535	G	...do.....	4.30	Drift 2 north	.....do.............	.....do.............
119	316	G	...do.....	4.50	Drift 3 north......	Shoveling.........	.....do.............

in sheet-ground mines of the Joplin district—Continued.

Distance of bulb from source of dust.	Air circulation.	Condition of working place.	Temperature.		Relative humidity.	Time consumed in taking sample.	Air taken.	Dust in sample.	Dust per 100 liters of air.	Remarks.
			"Wet bulb."	"Dry bulb."						
Ft.			°*F.*	°*F.*	*P.ct.*	*Min.*	*Liters*	*Mg.*	*Mg.*	
30	Fair...	Damp..	60	61.5	92	2½	45	0.60	1.33	Compare with samples 49 to 54. Samples 79 to 85 were taken at same place and under same conditions, except that sprays were in use at all drills.
30	Fair...	Damp..	60	61.5	92	2½	45	.60	1.33	
30	Fair...	Damp..	60	61.5	92	2½	45	.50	1.11	
30	Fair...	Damp..	60	61.5	92	2½	45	.50	1.11	
30	Fair..	Damp..	61.5	62.5	94	2½	45	.70	1.55	
30	Fair..	Damp..	61.5	62.5	94	2½	45	.60	1.33	
30	Fair...	Damp..	61.5	62.5	94	2½	45	.40	.90	
(a)	Good..	Dry....	60	62	89	3	45	.80	1.78	Point A was 20 feet from face northeast of mill shaft; 2 drills and 4 shovelers working within 30 feet of sampler. Point B 100 feet south of A, 10 feet from face; 2 drills and 4 shovelers working within 40 feet of sampler. Samples taken to get average conditions during day.
(a)	Good..	Dry....	60	62	89	3	45	.40	.90	
(a)	Good..	Dry....	60	62	89	3	45	.90	2.00	
(a).	Good..	Dry....	60	62	89	3	45	1.20	2.67	
(a)	Good..	Dry....	60.5	62	92	3	45	.00	.00	
10	Good..	Dry....	60.5	62	92	3	45	102.80	228.45	
(a)	Good..	Dry....	60.5	62	92	3	45	.30	.67	
(a)	Good..	Dry....	60.5	62	92	3	45	.10	.22	
(a)	Good..	Dry....	60.5	62	92	3	45	.20	.45	
(a)	Good..	Dry....	60.5	62	92	3	45	.20	.45	
10	Good..	Dry....	60.5	62	92	3	45	5.20	11.55	
15	Poor..	Dry....	58	60	89	2½	45	.90	2.00	
15	Poor..	Dry....	58	60	89	2½	45	2.30	5.11	Holes wet with squirt gun, not effective.
125	Poor..	Dry....	58	60	89	2½	45	.70	1.55	Drilling and shoveling at face, 125 feet from sampler.
3	Poor..	Dry....	59	61	89	2½	45	16.10	35.78	Bulb held close to face, on same level with hole, 3 feet to right.
200	Poor..	Dry....	60	62.5	87	2½	45	1.10	2.45	Taken 5 minutes after squibbing at face.
15	Poor..	Dry....	60	62.5	87	2½	45	.60	1.34	
10	Poor..	Dry....	60	62.5	87	2½	45	6.10	13.55	
60	Poor..	Dry....	60	62.5	87	2½	45	.00	.00	Taken 60 feet back from point at which No. 103 was taken.
250	Poor..	Dry....	60	62.5	87	2½	45	.40	.90	Taken 250 feet from working face.
40	Poor..	Wet....	53.5	54	97	2½	45	.80	1.78	Machine working in heading above, 40 feet from sampler.
5	Poor..	Wet....	61	62	94	2½	45	.70	1.55	Hole 6 feet deep; sample taken directly in front of hole.
3	Poor..	Wet....	61	62	94	2½	45	.60	1.33	
3	Poor..	Wet....	61	62	94	2½	45	.60	1.33	
5	Poor..	Dry....	61	62	94	2½	45	11.50	25.55	Hole in 4 feet; sample bulb held 4 feet below and 4 feet to right of hole.
11	Poor..	Wet....	54	55	94	2½	45	16.60	36.90	Hole in 6 feet; sample bulb held 10 feet below and 3 feet to left of hole.
20	Poor..	Wet....	54	55	94	2½	45	2.10	4.67	Taken 20 feet back from point where No. 111 was taken.
75	Poor..	Wet....	54	55	94	2½	45	2.40	5.33	Taken 75 feet back from point where No. 111 was taken.
7	Poor..	Wet....	58	60	89	2½	45	3.30	7.33	Hole in 2 feet; sample bulb held 3 feet below and 3 feet to left of hole.
4	Poor..	Damp..	59	60	94	2½	45	1.40	3.11	Sampler stood in helper's position on ladder directly in front of hole.
40	Poor..	Dry....	59	60	94	2½	45	.80	1.78	Samples 118 to 124 taken after regular squibbing, at 4 p. m., which made considerable dust in places.
30	Poor..	Dry....	58.5	60	92	2½	45	.90	2.00	
30	Poor..	Dry....	59	60	94	2½	45	1.90	4.22	
30	Poor..	Dry....	59	60	94	2½	45	1.10	2.44	

[a] Bulbs held at various distances.

TABLE 8.—*Description of siliceous dust samples t*

Sample No.	Bulb No.	Mine.	Date (1915).	Time.	Where taken.	Work progressing.	Position of
79	301	A	Mar. 1	*p. m.* 2.15	30 feet from west face.	Drilling at face....	5 feet from f
80	325	A	...do.....	2.30	.....do............	.....do............	.....do...
81	335	A	...do.....	2.45	.....do............	.....do............	.....do..
82	303	A	...do.....	3.00	.....do............	.....do............	.....do..
83	549	A	...do.....	3.15	.....do............	.....do............	.....do..
84	302	A	...do.....	3.30	.....do............	.....do............	.....do
85	548	A	...do.....	3.45	.....do............	.....do............	.....d
86	958	D	Mar. 3	*a. m.* 9.15	Point A............	General............	5 feet
87	960	D	...do.....	9.30	Point B............	.....do............	.....
88	297	D	...do.....	9.45	Point A............	.....do............	.....
89	345	D	...do.....	10.00	Point B............	.....do............	.....
90	957	D	...do.....	*p. m.* 2.15	Point A............	.....do............	...
91	954	D	...do.....	2.25	Face, near point A.	Blowing hole dry..	...
92	955	D	...do.....	2.30	Point B............	General............	.
93	959	D	...do.....	2.45	Point A............	.....do............	..
94	315	D	...do.....	3.00	Point B............	.....do............	.
95	951	D	...do.....	3.15	Point A............	.....do............	.
96	211	D	...do.....	3.30	Face, near point A.	Blowing damp holes.	
97	304	E	Mar. 5	*a. m.* 9.30	South face, machine 5.	Drilling and shov eling.	
98	305	E	...do.....	9.50	South face.........	Blowing dam holes.	
99	306	E	...do.....	10.15	125 feet from south face.	None..........	
100	307	E	...do.....	11.00	North face.........	Drilling bre hole.	
101	308	E	...do.....	*p. m.* 2.25	200 feet from north face.	None.....	
102	309	E	...do.....	2.30	North face.........	Blowing hole.	
103	313	E	...do....	2.55	South face.........	Startin	
104	317	E	...do.....	3.00	60 feet from south face.	None	
105	547	E	...do.....	3.20	250 feet from south face.	.....	
106	542	F	Mar. 8	*a. m.* 10.30	Face northeast of north shaft.	...	
107	543	F	...do.....	10.45	Face northwest of north shaft.	I	
108	544	F	...do.....	11.00	Face west of north shaft.		
109	545	F	...do.....	*p. m.* 1.00	.....do.........		
110	546	F	...do.....	1.40	Machine No. 8..		
111	953	F	...do.....	2.00	[illegible]ine No. 5		
112	563	F	...do.....	[illegible]	[illegible]		
113	590	F	...do.....	[illegible]	[illegible]		
114	210	F	...do.....	[illegible]	[illegible] north [illegible]		
115	985	G	Mar. 10	[illegible]	[illegible]		
116	[illegible]	[illegible]	...do	[illegible]	[illegible]		
117	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		
118	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		
11[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		

[illegible]					Remarks.
[illegible]				[illegible]	Taken 300 feet from nearest work.
[illegible]				[illegible]	Taken 250 feet from nearest work.
[illegible]				[illegible]	Taken 50 feet from drills.
[illegible]					
[illegible]					
[illegible]					
[illegible]				[illegible]	2 machines and 3 shovelers working; taken 15 feet from nearest machine.
[illegible]					
[illegible]				[illegible]	Hole in 6 feet.
[illegible]				[illegible]	Men working at face 250 feet distant.
[illegible]					
[illegible]				[illegible]	No water used.
[illegible]				.44	Much water used from separate line; taken 10 feet to right of drill hole.
[illegible]					
[illegible]				12.45	Squibbed 12-foot stope hole, no water used.
[illegible]				1.10	Samples 133 to 139 taken to find quantity of dust made by blasting. All faces where samples were taken were shot at 8.30 p. m. Shovelers were working, and this had effect of keeping dust in motion to certain extent. All places had much dust, powder smoke, and water vapor in them.
[illegible]				7.78	
				3.11	
				7.11	
				6.00	
			[illegible]	4.67	
[illegible]			[illegible]	4.67	
[illegible]			.10	.22	
[illegible]			.90	2.00	Samples 140 to 153 taken to determine general conditions in mine. Point A was 30 feet from south face. Three water drills and 6 shovelers working within 100 feet of sampler. Wet under foot and air misty.
[illegible]			.80	1.78	
			.50	1.10	
[illegible]					
[illegible]		45	.60	1.33	Point B was 600 feet north of point A and 40 feet from north face. 5 solid steel piston drills and 7 shovelers working within 100 feet of sampler; wet underfoot. Water vapor in atmosphere at both points.
[illegible]					
[illegible]					
[illegible]					
[illegible]	2½	45	1.00	2.22	
[illegible]	2½	45	.30	.67	
	2½	45	.40	.90	
[illegible]	2½	45	1.00	2.22	
	2½	45	.50	1.10	
	2½	45	.80	1.78	
[illegible]	2½	45	.50	1.11	
	2½	45	.10	.22	
[illegible]	2½	45	.20	.44	
	2½	45	.40	.90	
[illegible]	2½	45	.70	1.55	Air hose filled with water before blowing.
[illegible]	2½	45	.70	1.55	
[illegible]	2½	45	1.30	2.88	Hole 8 feet from floor.
[illegible]	2½	45	6.10	13.55	Hole 10 feet from floor, coarse pieces flying.

bulb held at various distances.

TABLE 8.—*Description of siliceous dust samples taken*

Sample No.	Bulb No.	Mine.	Date (1915).	Time.	Where taken.	Work progressing.	Position of bulb.
120	318	G	Mar. 10	*p. m.* 5.00	Drift 5 north......	Shoveling..........	5 feet from floor.....
121	322	G	...do.....	5.15	300 feet south of air drift.	None..............	.....do...............
122	564	G	...do.....	5.25	250 feet south of air drift.	.....do...............	.....do...............
123	567	G	...do.....	5.35	Drift No. 11........	Drilling............	.....do...............
124	967	G	...do.....	5.45	Drift No. 13........	Shoveling..........	.....do...............
125	968	G	...do.....	6.00	Drift north of lay-by 4.	Blowing damp holes.	.....do...............
126	608	H	Mar. 11	2.00	Southwest face....	Drilling and shoveling.	.....do...............
127	609	H	...do.....	2.05	.....do.............	Drilling roof hole dry.	At drillman's head.
128	611	H	...do.....	2.20	250 feet from southwest face.	None..............	5 feet from floor.....
129	612	H	...do.....	2.30	Northeast face....	Starting stope hole	4 feet above hole....
130	613	H	...do.....	3.00	North face.........	Drilling roof hole..	4 feet from floor.....
131	615	H	...do.....	3.25	Northeast face....	5 minutes after squibbing.	.....do...............
132	617	H	...do.....	3.30	North face.........	Drilling roof hole..	.....do...............
133	601	G	Mar. 12	9.15	Drift No. 5.........	After blasting.....	.....do...............
134	602	G	...do.....	9.30	Drift No. 7.........	.....do...............	.....do...............
135	603	G	...do.....	9.40	Drift No. 8.........	.....do...............	.....do...............
136	604	G	...do.....	9.50	Drift No. 10.......	.....do...............	.....do...............
137	605	G	...do.....	10.00	Drift No. 11.......	.....do...............	.....do...............
138	606	G	...do.....	10.05	Drift No. 13.......	After blasting; drilling.	.....do...............
139	607	G	...do.....	10.15	Drift No. 18.......	After blasting.....	.....do...............
140	962	I	Mar. 15	*a. m.* 10.00	Point A...........	General...........	5 feet from floor.....
141	969	I	...do.....	10.20	.....do.............	.....do...............	.....do...............
142	973	I	...do.....	10.40	.....do.............	.....do...............	.....do...............
143	984	I	Mar. 15	*p. m.* 2.00	Point A...........	General...........	5 feet from floor.....
144	619	I	...do....	2.20	.....do.............	.....do...............	.....do...............
145	987	I	...do....	2.40	.....do.............	.....do...............	.....do...............
146	533	I	...do....	3.00	.....do.............	.....do...............	.....do...............
147	963	I	...do....	*a. m.* 10.10	Point B...........	.....do...............	.....do...............
148	992	I	...do....	10.30	.....do.............	.....do...............	.....do...............
149	995	I	...do....	10.50	.....do.............	.....do...............	.....do...............
150	618	I	...do....	*p. m.* 2.10	.....do.............	.....do...............	.....do...............
151	279	I	...do....	2.30	.....do.............	.....do...............	.....do...............
152	534	I	...do....	2.50	.....do.............	.....do...............	.....do...............
153	202	I	...do....	3.10	.....do.............	.....do...............	.....do...............
154	961	J	Mar. 18	*a. m.* 9.15	Face, 400 feet west of shaft.	Blowing wet hole..	.....do...............
155	980	J	...do....	10.15	Face, 600 feet west of shaft.	Drilling breast hole.	5 feet out from hole..
156	981	J	...do....	10.25	Face, 700 feet southwest of shaft.	.....do...............	At helper's head.....
157	982	J	...do....	10.45	Face, 800 feet southwest of shaft.	Drilling roof hole..	.....do...............

in sheet-ground mines of the Joplin district—Continued.

Distance of bulb from source of dust.	Air circulation.	Condition of working place.	Temperature.		Relative humidity.	Time consumed in taking sample.	Air taken.	Dust in sample.	Dust per 100 liters of air.	Remarks.
			"Wet bulb."	"Dry bulb."						
Ft.			° *F.*	° *F.*	*P. ct.*	*Min.*	*Liters*	*Mg.*	*Mg.*	
30	Poor..	Dry....	59	60	94	2½	45	0.80	1.78	
300	Good..	Dry....	58	60.5	86	2½	45	.70	1.55	Taken 300 feet from nearest work.
250	Fair...	Dry....	58	60.5	86	2½	45	1.40	3.11	Taken 250 feet from nearest work.
50	Poor..	Dry....	59	60	94	2½	45	.90	2.00	Taken 50 feet from drills.
6	Poor..	Dry....	59	60	94	2½	45	.80	1.78	
30	None..	Wet....	59	60	94	2½	45	.90	2.00	
(*a*)	None..	Dry....	61	62	94	2½	45	.70	1.55	2 machines and 3 shovelers working; taken 15 feet from nearest machine.
8	None..	Dry....	61	62	94	2½	45	.80	1.78	Hole in 6 feet.
250	Poor..	Dry....	61	62	94	2½	45	.70	1.55	Men working at face 250 feet distant.
4	None..	Dry....	60	61	94	2½	45	8.50	19.00	No water used.
12	None..	Wet....	55	56.5	91	2½	45	.20	.44	Much water used from separate line; taken 10 feet to right of drill hole.
20	None..	Dry....	58	59	94	2½	45	5.60	12.45	Squibbed 12-foot stope hole, no water used.
30	None..	Damp..	55	56	94	2½	45	.50	1.10	Samples 133 to 139 taken to find quantity of dust made by blasting. All faces where samples were taken were shot at 8.30 p. m. Shovelers were working, and this had effect of keeping dust in motion to certain extent. All places had much dust, powder smoke, and water vapor in them.
20	Poor..	Dry....	58	60.5	86	2½	45	3.50	7.78	
20	Poor..	Dry....	58	60.5	86	2½	45	1.40	3.11	
20	Poor..	Dry....	58	60.5	86	2½	45	3.20	7.11	
(*a*)	Poor..	Dry....	58	60.5	86	2½	45	2.70	6.00	
30	Poor..	Dry....	58	60.5	86	2½	45	2.10	4.67	
18	Poor..	Dry....	58	60.5	86	2½	45	2.10	4.67	
10	None..	Wet....	59	60	94	2½	45	.10	.22	
(*a*)	Poor..	Wet ...	61	61	100	2½	45	.90	2.00	Samples 140 to 153 taken to determine general conditions in mine. Point A was 30 feet from south face. Three water drills and 6 shovelers working within 100 feet of sampler. Wet under foot and air misty.
(*a*)	Poor..	Wet ...	61	61	100	2½	45	.80	1.78	
(*a*)	Poor..	Wet ...	61	61	100	2½	45	.50	1.10	
(*a*)	Poor..	Wet....	61	61	100	2½	45	.60	1.33	Point B was 600 feet north of point A and 40 feet from north face. 5 solid steel piston drills and 7 shovelers working within 100 feet of sampler; wet underfoot. Water vapor in atmosphere at both points.
(*a*)	Poor..	Wet....	61	61	100	2½	45	1.00	2.22	
(*a*)	Poor..	Wet....	61	61	100	2½	45	.30	.67	
(*a*)	Poor..	Wet....	61	61	100	2½	45	.40	.90	
(*a*)	Poor..	Wet....	56.5	58	91	2½	45	1.00	2.22	
(*a*)	Poor..	Wet....	56.5	58	91	2½	45	.50	1.10	
(*a*)	Poor..	Wet....	56.5	58	91	2½	45	.80	1.78	
(*a*)	Poor..	Wet....	56.5	58	91	2½	45	.50	1.11	
(*a*)	Poor..	Wet....	56.5	58	91	2½	45	.10	.22	
(*a*)	Poor..	Wet....	56.5	58	91	2½	45	.20	.44	
(*a*)	Poor..	Wet....	56.5	58	91	2½	45	.40	.90	
15	Poor..	Damp .	56.5	58	91	2½	45	.70	1.55	Air hose filled with water before blowing.
5	Poor..	Damp..	56.5	58	91	2½	45	.70	1.55	
8	Poor..	Damp..	56.5	58	91	2½	45	1.30	2.88	Hole 8 feet from floor.
10	Poor..	Damp..	56.5	58	91	2½	45	6.10	13.55	Hole 10 feet from floor, coarse pieces flying.

a Bulb held at various distances.

TABLE 8.—*Description of siliceous dust samples taken*

Sample No.	Bulb No.	Mine.	Date (1915).	Time.	Where taken.	Work progressing.	Position of bulb.
158	983	K	Mar. 18	*a. m.* 11.00	Southwest face	Various kinds	20 feet from drill
159	638	J	do	*p. m.* 1.30	Face, 400 feet west of shaft.	Drilling breast hole.	7 feet out from hole.
160	639	J	do	2.00	Face, 600 feet west of shaft.	Shoveling	5 feet from hole
161	640	J	do	2.30	Face, 800 feet southwest of shaft.	After squibbing	do
162	641	J	do	2.40	do	do	do
163	642	J	do	3.00	Face, 400 feet west of shaft.	Drilling breast hole.	10 feet out from hole.
164	643	J	do	3.05	do	do	2 feet out from hole
165	644	L	Mar. 22	12.50	Face, southwest of shaft.	None	20 feet from face
166	645	L	do	1.00	do	Drilling and shoveling.	3 feet from floor
167	646	L	do	1.50	Face, south of shaft.	do	4 feet from floor
168	647	L	do	2.15	Face, southwest of shaft.	Shoveling	3 feet from floor
169	648	L	do	2.30	do	do	do
170	649	L	do	2.45	do	do	do
171	620	L	do	3.00	do	do	do
172	621	L	do	3.15	do	do	do
173	622	L	do	3.55	do	None	4 feet from floor
174	623	L	do	4.05	Face, west of shaft.	After squibbing	do
175	624	L	do	4.15	do	do	do
176	625	L	do	4.25	do	do	do
177	633	L	do	4.40	do	do	do
178	637	L	do	5.00	do	do	do
179	869	M	Mar. 23	1.20	Northeast face	Various kinds	do
180	872	M	do	1.50	Station northeast of shaft.	do	do
181	873	M	do	2.00	do	do	do
182	874	M	do	2.15	do	do	do
183	889	M	do	2.25	do	do	do
184	890	M	do	2.35	do	do	do
185	892	M	do	2.45	do	do	do
186	898	M	do	3.00	do	do	do
187	899	M	do	3.15	do	do	do
188	634	N	Mar. 24	12.50	Face, 200 feet southeast of shaft.	Drilling	do
189	635	N	do	1.00	Face, 250 feet southeast of shaft.	do	do
190	891	N	do	1.25	Face, 350 feet east of shaft.	do	do
191	893	N	do	2.15	do	Blowing wet holes.	do
192	894	N	do	2.30	Face, 250 feet southeast of shaft	Drilling do	do do
193	895	N	do	2.45	do	do	5 feet from floor, 3 feet out from hole.

in sheet-ground mines of the Joplin district—Continued.

Distance of bulb from source of dust.	Air circulation.	Condition of working place.	Temperature.		Relative humidity.	Time consumed in taking sample.	Air taken.	Dust in sample.	Dust per 100 liters of air.	Remarks.
			"Wet bulb."	"Dry bulb."						
Ft.			°*F.*	°*F.*	*P.ct.*	*Min.*	*Liters*	*Mg.*	*Mg.*	
(a)	Poor..	Wet....	52	55	82	2½	45	0.80	1.78	3 drills and 5 shovelers working, taken 20 feet from nearest drill.
7	Poor..	Wet....	56	58.5	86	2½	45	1.40	3.11	Hole 12 feet in.
15	Poor..	Damp..	56	58.5	86	2½	45	.50	1.11	
15	Poor..	Damp..	56	58.5	86	2½	45	3.10	6.90	Taken 3 minutes after squib shot. Top layer of sugar blackened by smoke.
60	Poor..	Damp..	56	58.5	86	2½	45	1.30	2.88	Taken 13 minutes after same squib shot. Top layer of sugar blackened by smoke.
10	Poor..	Damp..	56	58.5	86	2½	45	.50	1.11	Compare with No. 164.
2	Poor..	Damp..	56	58.5	86	2½	45	.80	1.78	Compare with No. 163.
20	Poor..	Dry....	56	58.5	86	2½	45	.50	1.11	Taken 50 minutes after squib shot.
(a)	Poor..	Dry....	56	58.5	86	2½	45	1.00	2.22	2 shovelers 8 feet from sampler; 1 drill 12 feet.
(a)	Poor..	Dry....	56	58.5	86	2½	45	.70	1.55	Stood 3 feet from shoveler and 8 feet from drill.
3	Poor..	Damp..	54	58	77	2½	45	.60	1.33	Samples 168 to 172 taken at same point.
3	Poor..	Damp..	54	58	77	2½	45	.70	1.55	
3	Poor..	Damp..	54	58	77	2½	45	.50	1.10	
3	Poor..	Damp..	54	58	77	2½	45	.00	.00	
3	Poor..	Damp..	54	58	77	2½	45	.00	.00	
20	Poor..	Dry....	54	58	77	2½	45	.60	1.33	Taken 10 minutes after blowing 3 holes wet.
75	Poor..	Dry....	54	58	77	2½	45	1.80	4.00	Samples 174 to 178 were taken to determine effect of squibbing. 11 squibs were fired at 4 p. m. At 4.05, when first sample was taken, water had been turned into air line and place was being thoroughly sprinkled. Sprinkling was continued until 4.35; 4 hose were used.
75	Poor..	Dry....	55.5	56.5	94	2½	45	1.10	2.44	
75	Poor..	Damp..	55.5	56.5	94	2½	45	.70	1.55	
75	Poor..	Wet....	56	57	94	2½	45	.60	1.33	
75	Poor..	Wet....	56	57	94	2½	45	.60	1.33	
(a)	Good..	Wet....	57	59	89	2½	45	.50	1.10	2 shovelers 8 feet and 1 drill 50 feet from point of sampling.
(a)	Good..	Wet....	57	59	89	2½	45	.60	1.33	Samples 180 to 187 taken at point 20 feet from northeast face, 4 solid steel piston drills and 6 shovelers were working within 100 feet. Face, roof, and floor very wet; good air current. Squibbing and blowing holes frequently.
(a)	Good..	Wet....	57	59	89	2½	45	1.30	2.88	
(a)	Good..	Wet....	57	59	89	2½	45	.70	1.55	
(a)	Good..	Wet....	57	59	89	2½	45	.50	1.10	
(a)	Good..	Wet....	57	59	89	2½	45	.90	2.00	
(a)	Good..	Wet....	57	59	89	2½	45	.50	1.10	
(a)	Good..	Wet....	57	59	89	2½	45	1.30	2.88	
(a)	Good..	Wet....	57	59	89	2½	45	.80	1.78	
30	Poor..	Dry....	57	58.5	91	2½	45	.40	.90	Shovelers not working in the mine on day sample was taken.
25	Poor..	Dry....	57	58.5	91	2½	45	.90	2.00	
25	Poor..	Dry....	56	59	83	2½	45	.90	2.00	
20	Poor..	Dry....	56	59	83	2½	45	.40	.90	
25	Poor..	Dry....	57.5	58.5	94	2½	45	1.10	2.44	
3	Poor..	Dry....	57.5	58.5	94	2½	45	.2	2.67	

[a] Bulb held at various distances.

TABLE 8.—*Description of siliceous dust samples taken*

Sample No.	Bulb No.	Mine.	Date (1915).	Time.	Where taken.	Work progressing.	Position of bulb.
				p. m.			
194	896	N	Mar. 24	2.50	Face, 250 feet southeast of shaft.	Drilling	5 feet from floor, 10 feet out from hole.
195	897	N	...do....	3.00	.....do	.....do	4 feet from floor
196	900	G	Mar. 25	9.45	Heading No. 8	1½ hours after blasting.	5 feet from floor
197	866	G	...do.....	10.00	Heading No. 10	1½ hours after blasting.	.....do
198	863	H	Apr. 7	2.25	Southwest heading.	Drilling and shoveling.	.....do
199	864	H	...do.....	2.30	.....do	.....do	.....do
200	865	H	...do.....	2.40	250 feet from southwest face.	None	.....do
201	867	H	...do.....	3.20	100 feet east of north shaft.	Shoveling	At shoveler's head
202	675	H	...do.....	3.40	100 feet from northwest face.	Various places at face.	5 feet from floor
203	699	H	...do.....	3.45	.....do	.....do	.....do
204	696	H	...do.....	3.50	20 feet from northwest face.	.....do	.....do
205	670	O	Apr. 28	1.00	20 feet from north face.	None	.....do
206	699	O	...do.....	1.25	20 feet from northwest face.	Shoveling and drilling.	.....do
207	859	O	...do.....	1.40	20 feet from north face.	10 minutes after squibbing.	.....do
208	671	O	...do.....	2.00	20 feet from northwest face.	Shoveling and drilling.	.....do
209	672	O	...do.....	2.25	.....do	.....do	.....do
210	860	H	Apr. 30	2.40	50 feet from north face.	Shoveling and drilling at face.	.....do
211	861	H	...do.....	3.00	125 feet from north face.	.....do	.....do
212	857	H	...do.....	3.18	.....do	.....do	.....do
213	674	H	...do.....	3.30	.....do	.....do	.....do
214	966	G	May 1	4.50	Heading No. 11	None	.....do
215	989	G	...do.....	5.05	.....do	.....do	.....do
216	990	G	...do.....	5.25	.....do	.....do	.....do
217	991	G	...do.....	5.45	.....do	.....do	.....do
218	992	G	...do.....	6.05	.....do	.....do	.....do
219	993	G	...do.....	6.35	.....do	.....do	.....do
220	994	G	...do.....	7.10	.....do	.....do	.....do
221	995	G	...do	7.50	.....do	.....do	.....do
222	998	G	...do.....	8.30	.....do	.....do	.....do

in sheet-ground mines of the Joplin district—Continued.

Distance of bulb from source of dust.	Air circulation.	Condition of working place.	Temperature.		Relative humidity.	Time consumed in taking sample.	Air taken.	Dust in sample.	Dust per 100 liters of air.	Remarks.
			"Wet bulb."	"Dry bulb."						
Ft.			*° F.*	*° F.*	*P. ct.*	*Min.*	*Liters*	*Mg.*	*Mg.*	
10	Poor..	Dry....	57.5	58.5	94	2½	45	0.50	1.10	
25	Poor..	Dry....	57.5	58.5	94	2½	45	.60	1.33	
20	Poor..	Dry....	58	60	89	2½	45	2.00	4.45	Compare with samples 95 to 100. Fan was running in No. 7 shaft.
20	Poor..	Dry....	58	60	89	2½	45	.00	.00	
15	Poor..	Dry....	60.5	63.5	84	2½	45	.80	1.78	
5	Poor..	Dry....	62	64	90	2½	45	.20	.45	Much water in use.
250	Poor..	Dry....	60	63	84	2½	45	.90	2.00	
4	Poor..	Wet....	60	63	84	2½	45	.00	.00	Water running on dirt.
100	Poor..	Dry....	62.5	65	87	2½	45	.60	1.33	
100	Poor..	Dry....	62.5	65	87	2½	45	1.60	3.55	3 squibs fired at 3.45 p. m.
100	Poor..	Dry....	62.5	65	87	2½	45	10.30	22.89	Shows effect of firing the 3 squibs.
......	Poor..	Wet....	61	62	94	2	42	.20	.48	
30	Poor..	Wet....	61	62	94	2	42	.60	1.43	
20	Poor..	Wet....	61	62	94	2	42	1.20	2.86	
30	Poor..	Wet....	61	62	94	2	42	.90	2.14	
30	Poor..	Wet....	61	62	94	2	42	1.40	3.33	
50	None .	Dry....	60.5	62.5	89	2	42	.60	1.43	Much powder smoke and water vapor.
125	None .	Dry....	60.5	62.5	89	2	42	.20	.48	Sampler had to retreat because of squib shots.
125	None .	Dry....	60	63	84	2	42	.50	1.19	6 squibs fired at 3.07 p. m.
125	None .	Dry....	60	63	84	2	42	1.20	2.86	Dust working back from face.
15	Poor..	Damp .	60	60	100	2	42	.80	1.90	Much powder smoke and water vapor.
15	Poor..	Damp .	60	60	100	2	42	.00	.00	Do.
15	Poor..	Damp .	60	60	100	2	42	.10	.24	Do.
15	Poor..	Damp .	60	60	100	2	42	.60	1.43	Do.
15	Poor..	Damp .	60	60	100	2	42	.60	1.43	Do.
15	Poor..	Damp .	60	60	100	2	42	.70	1.66	Much water vapor.
15	Poor..	Damp .	60	60	100	2	42	.40	.98	Do.
15	Poor..	Damp .	60	60	100	2	42	.20	.48	Do.
15	Poor..	Damp .	60	60	100	2	42	.00	.00	Do.

In studying these tables, it must be borne in mind that the same operation may produce widely varying quantities of dust in different mines. As pointed out on a previous page, the rock varies in dust-making tendency in different parts of the sheet-ground district; again, the presence or absence of water in the working place is an important factor. Ventilation also has an important effect on the time that dust may remain in suspension in the working places.

Samples taken at different hours of the day, also those taken at night, in all parts of the mines, show varying quantities of dust. In many of the mines where it was customary to squib and blow holes excessively just before loading, a far greater amount of dust was produced in the period between 3 and 4 o'clock in the afternoon. Although some of the samples taken indicate this condition, it is not brought out as clearly as it should be. Owing to the necessity of leaving the face when holes are squibbed, there was often difficulty in sampling during this period.

Due consideration should be given the fact that in taking samples close to shovelers, or beside drillmen, the mine air in the vicinity contained a certain amount of dust produced by other work. For instance, in obtaining a sample of dust produced by drilling, a hole might be blown in a near-by working place, and the dust so produced would vitiate the sampling at the drill. Such samples were rejected. In some cases, where the interference was only nominal, the samples are included in the tables with explanatory notes.

The samples show that dust is produced in wet working places as well as in dry places. By holding a carbide lamp so that the eyes are shaded, dust may be seen with ease in the light rays. However, smoke from shots or visible water vapor in a stream of light much resembles dust, a fact that may lead miners and others to believe that a working place is much dustier than it really is.

BLOWING DRILL HOLES.

Sample 91, taken at a point 10 feet from a drill hole being blown dry, showed 228.45 milligrams of dust per 100 liters of air. There is no doubt that the blowing of dry holes causes more dust than any other detail of mining. That a squirt gun is not always effective in wetting the dust before blowing holes, and that damp holes will make considerable dust, is shown by samples No. 96 (11.55 milligrams); No. 98 (5.11 milligrams); and No. 16 (4.29 milligrams). Holes that are naturally wet, or that are thoroughly wet by using a hose and nozzle, make very little dust, as shown by samples No. 102 (1.34 milligrams), No. 154 (1.55 milligrams), and No. 191 (0.90 milligrams).

DRILLING.

Samples taken to determine the quantity of dust made by drilling varied widely. The chief factors in this variation were: The amount of water made or supplied at the face, the distance from the collar of the drill hole at which the sample was taken, and the dust-producing tendency of the rock. The most important facts brought out by the samples of dust produced by drilling were the following: The greatest quantity of dust is produced when the hole is being started, decreasing as the hole becomes deeper. In general, roof holes produce more dust than breast holes, and breast holes produce more than stope holes. The quantity of dust in the samples decreases rapidly with the distance away from the drill hole. Little or no dust is produced from wet holes. These points are demonstrated by most of the samples listed in Tables 9 and 10, which contain the drilling samples embraced in Table 8.

TABLE 9.—*Data regarding samples of dust produced by drilling without water.*

Sample No.	Mine.	Distance of sample bulb from hole.	Kind of of hole.	Depth of hole.	Dust per 100 liters of air.
		Feet.		*Feet.*	*Mg.*
40	A	8	Breast	10.25	2.86
41 [a]	A	8	...do...	10.75	2.86
42 [a]	A	8	...do...	11.50	2.28
43 [a]	A	8	...do...	12.00	2.00
44 [a]	A	8	...do...	.17	12.86
45 [a]	A	1.5	...do...	.25	221.71
46 [a]	A	8	...do...	.50	2.28
47 [a]	A	8	...do...	2.00	.57
48 [a]	A	8	...do...	2.50	.57
100	E	3	...do...	1.00	35.78
103	E	10	Stope	.25	13.55
107	F	5	Breast	6.00	1.55
127	H	8	Roof	6.00	1.78
129	H	4	Stope	.17	19.00
155	J	5	Breast	6.00	1.55
156	J	8	...do...	6.50	1.30
157	J	10	Roof	.33	13.55
159	J	7	Breast	12.00	3.11
163	J	10	...do...	6.00	1.11
164 [a]	J	2	...do...	6.00	1.78
188	N	30	...do...	4.00	.90
189	N	25	...do...	4.00	2.00
190	N	25	...do...	4.00	2.00
192	N	25	...do...	6.00	2.44
193	N	3	...do...	6.50	2.67
194 [a]	N	10	...do...	6.50	1.10
195	N	25	...do...	3.00	1.33

[a] Taken at same point as preceding sample.

TABLE 10.—*Data regarding samples of dust produced by drilling wet or damp holes.*

Sample No.	Mine.	Distance of sample bulb from hole.	Kind of hole.	Depth of hole.	Source of water.	Dust per 100 liters of air.
		Feet.		*Feet.*		*Mg.*
55	A	8	Breast	11.5	Spray	1.43
56 [a]	A	8	...do...	.17	...do...	2.28
57 [a]	A	8	...do...	1.00	...do...	1.71
58 [a]	A	8	...do...	1.50	...do...	1.14
59 [a]	A	8	...do...	2.00	...do...	.00
60 [a]	A	8	...do...	2.33	...do...	.00
61 [a]	A	8	...do...	2.00	...do...	.29
70	C	8	...do...	3.50	Damp face.	1.78
71	C	8	...do...	4.00	...do...	.45
107	F	5	...do...	6.00	...do...	1.55
111	F	11	Roof (high).	.50	Wet face	36.90
112 [a]	F	20	...do...	1.25	...do...	4.67
113 [a]	F	75	...do...	2.00	...do...	5.33
114	F	7	...do...	2.00	...do...	7.33
115	G	4	...do...	2.75	Damp face.	3.11
130	H	12	...do...	3.00	Spray	.44
132	H	30	...do...	1.50	Damp face.	1.10
A [b]	I	30	Various			1.43
B [c]	I	40	...do...			1.11
155	J	5	Breast	3.00	Damp face.	1.65
156	J	8	...do...	7.00	...do...	2.88
157	J	10	Roof	1.00	...do...	13.55
163	J	10	Breast	6.00	...do...	1.11
164 [a]	J	2	...do...	6.00	...do...	1.78

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[a] Taken at same point as preceding sample.

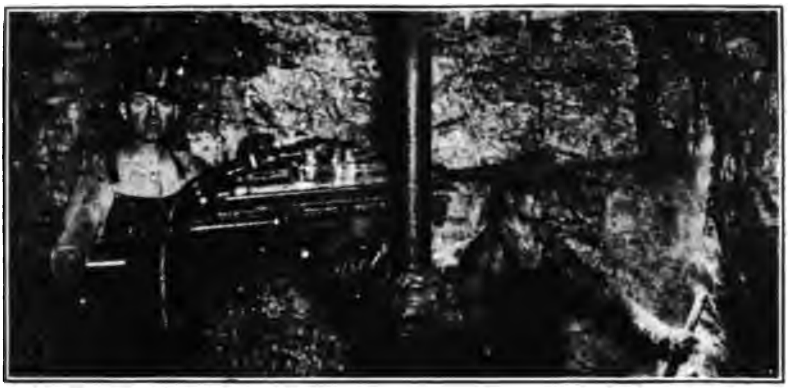
[b] Sample A was average of samples 140 to 146; shoveling and drilling with water drills in progress. (See Table 8.)

[c] Sample B was average of samples 147 to 153; shoveling and drilling with solid steel piston drills in progress; face very wet. (See Table 8.)

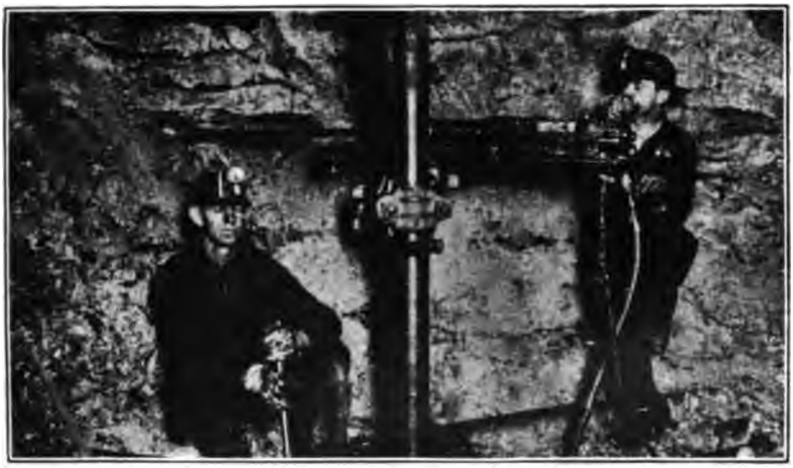
Samples 40 to 48 and 55 to 61 afford an interesting comparison as to the quantity of dust made by a piston drill working without water (see Pl. X, *A*) and the same drill working while water from a nozzle was played into the hole from time to time. The samples were taken at intervals of 15 minutes. The results have been arranged in the following table for convenient comparison:

Results of drilling with a piston drill with and without water in the hole.

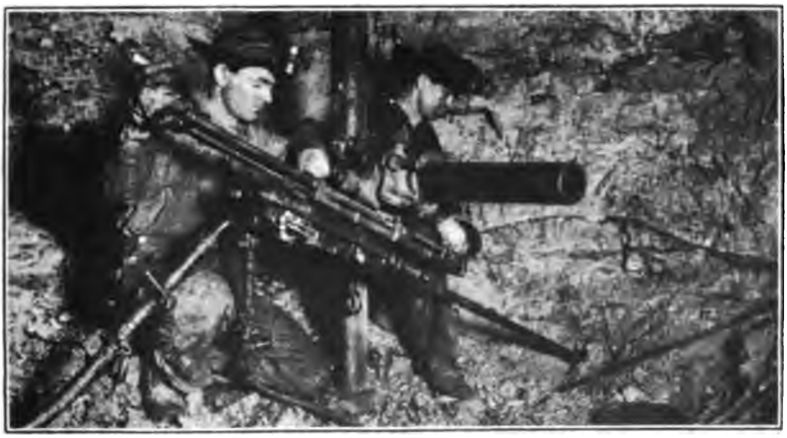
Sample No.	Without water.		Sample No.	Using water spray.	
	Depth of hole, feet.	Milligrams of dust in 100 liters of air.		Depth of hole, feet.	Milligrams of dust in 100 liters of air.
40	10.25	2.86	55	11.50	1.43
41	10.75	2.86	56	.17	2.28
42	11.50	2.28	57	1.00	1.71
43	12.00	2.00	58	1.50	1.14
44	.17	12.86	59	2.00	.00
46	.50	2.28	60	2.33	.00
47	2.00	.57	61	3.00	.29
48	2.50	.57	Average		.99
Average		3.29			



A. DRILLING DRY HOLE WITH SOLID STEEL PISTON DRILL. NOTE DUST ON HELPER'S SLEEVE AND ON ROCK; ALSO NOTE HAZE NEAR DRILL HOLE DUE TO DUST.



B. DRILLING BREAST HOLE WITH WATER-INJECTION DRILL. NOTE CLEARNESS OF VIEW AND ABSENCE OF DUST.



C. DRILLING STOPE HOLE WITH WATER-INJECTION DRILL. NOTE CLEARNESS OF VIEW AND ABSENCE OF DUST.

It will be noted that about three and one-third times more dust was produced in drilling without water. The results for samples 46, 47, and 48 would have been higher had not the exhaust from a near-by drill carried away some of the dust. The comparison would have been even more striking had it not been for the fact that the working place in both tests was comparatively wet. All of these samples were taken near the drillman's head, 8 feet from the collar of the hole. Sample 45, which showed 221.71 milligrams of dust, was taken 1½ feet from the hole when it was only 4 inches deep. (See Table 9.) This sample obviously contained many coarse particles.

Many other samples were taken in places where drilling and other mining work were in progress. The conclusions to be drawn from all of these samples are that drilling in wet places will make some dust, though not near as much as drilling in dry places; and that dust from drilling may be practically prevented by the use of drills that provide for introducing water into the hole through the core of the drill (see Pl. X, *B* and *C*), or by the use of a spray or stream of water directed into the hole.

SQUIBBING.

For convenience in studying the results of squibbing, the following tabulation is rearranged from Table 8.

Data on dust produced by squibbing in sheet-ground mines.

Sample No.	Number of holes.	Condition of working place.	Ventilation.	Distance of bulb from face, feet.	Time between squibbing and sampling, minutes.	Milligrams of dust per 100 liters in sample.
2	1	Dry	Fair	15	3	[a]4.86
18	1	Damp	do	15	2	4.29
27	1	Wet	Poor	15	2	7.16
28 [b]	1	do	do	15	5	5.71
38	1	Dry	do	10	30	[c]4.00
131	1	do	do	20	5	12.45
161	1	Damp	do	15	3	6.90
162 [b]	1	do	do	60	13	2.88
174	11	Dry	do	75	5	4.00
207	1	Wet	do	20	10	2.86

[a] Duplicate showed 6.0 milligrams.

[b] Sample taken at same place as No. 27.

[c] Duplicate showed 4.86 milligrams.

In addition to the above samples many of the general samples shown in Table 8 included dust made from squibbing. In comparing the samples taken after squibbing with those taken during drilling, shoveling, and other operations, there are two important allowances to be made—namely, the samples taken after squibbing were collected some minutes after the shot, and at distances of 15 to 75 feet from the face; whereas the samples during drilling, shoveling, etc., were procured while the operation was in progress, and usually at points closer to the source of dust. The time interval and the distance factor have an important bearing on the quantity of dust found in the air. It is plainly seen that squibbing, next to blowing dry holes, is the chief producer of dust in the sheet-ground mines.

In nearly all of the samples obtained after squibbing the sugar in the sampling bulb was blackened by powder smoke to a depth of one-sixteenth to one-fourth inch. Sample 174 (Table 8) was taken 75 feet from the face, 5 minutes after 11 holes had been squibbed. The sample showed 4 milligrams of dust per 100 liters of air. Immediately after this sample was taken water was turned into the air line and the working places were thoroughly wet, four hoses being used. This spraying was continued without interruption for 30 minutes. Samples taken at 10-minute intervals after sample 174 was collected showed 2.44, 1.55, 1.33, and 1.33 milligrams of dust, respectively.

Inasmuch as wet holes when squibbed produce much dust, although not as much as dry holes, the use of water will not avail to abate the dust. The only remedy is to stop squibbing while the shift is at work. This procedure is simple enough as regards squibbing for chambering holes, for that work can be done as the shift leaves the mine. The chief trouble is from squib shots fired to remove obstructions from the hole when the drill steel sticks. Many of the miners claim that the amount of squibbing can be reduced, but there are cases where it must be used in order to save a hole. After careful observation for a period of several months the writer believes that squibbing is largely a matter of habit, and that the sticking of the drill can usually be remedied by aligning the drill properly. In some of the mines the practice of squibbing has been entirely eliminated; in others, it is permitted only when the hole is deep and after every effort has been made to align the drill. In such cases the abandonment of the hole might mean the loss in labor of two men for one-fourth to one-third of an entire shift. Aside from the dust it causes squibbing is a costly practice. Every time a squib shot is fired all of the shovelers, drillmen, helpers, and others in the vicinity have to seek safety behind a pillar, thus curtailing the production. If to this cost be added the cost of the explosive and fuse used, it becomes apparent that the practice of squibbing, especially in mines where much used, may be a considerable item of expense.

BLASTING.

Samples of air taken after blasting show that blasting produces even more dust than does squibbing. As pointed out on a previous page, dust from blasting becomes a menace only when one shift follows closely on the preceding one. Good ventilation, or thorough sprinkling, and the regulation of shifts in accordance with the condition of the working places, will prevent inconvenience from dust caused by blasting.

SHOVELING.

Dust samples obtained where shoveling was in progress indicate that little or no dust is produced when the dirt is wet. As much as 1.55 milligrams per hundred liters were obtained where the dirt was damp; samples where the dirt was dry contained 1.78 to 4.59 milligrams of dust. It follows that dust produced by shoveling may be largely abated by spraying the dirt pile.

GENERAL CONDITIONS.

As will be noted in Table 8, many samples were taken while two or more kinds of work were being done. In some cases an average place in the mine was selected and samples taken at intervals throughout the day. In mine "A" samples 49 to 54 were taken at 15-minute intervals 30 feet from a comparatively wet face where drilling and shoveling were in progress. Samples 79 to 85 were taken at the same point the first day after the installation of water sprays. The following tabulation of results is interesting:

Effect of spraying on quantity of siliceous dust produced by drilling and shoveling.

Without sprays.		With sprays.	
Sample No.	Milligrams of dust in 100 liters of air.	Sample No.	Milligrams of dust in 100 liters of air.
49	2.00	79	1.33
50	3.43	80	1.33
51	3.71	81	1.11
52	3.71	82	1.11
53	1.71	83	1.55
54	2.86	84	1.33
		85	.90
Average	2.90	Average	1.24

Had this face been dry to start with the amount of dust would have been still more reduced. It was noted also that three squib shots were fired during the sampling while sprays were in use, which increased the quantity of dust in the air.

Samples 62 to 66 were taken from a naturally dry mine where water sprays were in use. This mine was remarkably free from dust, the samples showing from 0.6 to 0.8 milligram, with the exception of No. 63, which showed 3 milligrams. In this case the dust was caused by large bowlders being rolled down from a high heading. This mine was visited many times, but dust was not noticeable.

Mine "H," one of the dryest and dustiest mines of the district, was sampled before and after water sprays were installed. Twelve samples taken when water was not used ranged from 1.10 to as high as 22.89 miligrams of dust; this latter sample was taken 100 feet from the working face after several squib shots. Samples taken when sprays were used ranged from 0.00 to 1.78 milligrams. In this mine the samples were not taken at regular intervals for the purpose of obtaining an average, so that it is not possible to figure the exact effect of the use of water. However, it would probably be safe to say that the use of sprays in this mine reduced the amount of dust to one-sixth or one-eighth the amount formerly made.

Attention is directed to samples 101 (2.45 milligrams); 105 (0.90 milligram); 121 (1.55 milligrams); 122 (3.11 milligrams); and 200 (2 milligrams), all of which were taken in various mines at points 200 feet and more from the working face.

WHAT CONSTITUTES A DUSTY MINE.

The term "dusty working place" is only relative; 1 to 5 milligrams of dust per 100 liters of air will render a working place dusty; the place will be dusty, but of course more so if the air contains from 50 to 100 milligrams of dust per 100 liters. A classification such as slightly dusty, dusty, and very dusty places may be made. It seems desirable, however, to adopt a standard content for the dividing line between slightly dusty and dusty working places. Evidently such a standard can not properly be based on the minimum quantity of dust that will harm the miner's lungs; it can be said only that a small quantity of dust in the mine air will be less harmful than a larger amount. Practically all air, both inside and outside of mines, contains some dust. The most reasonable standard then appears to be one based on the quantity of dust that will remain in suspension after the best known methods have been put into use for its abatement.

It has been demonstrated in the sheet-ground mines of the Joplin district that by the proper use of water and the regulation of certain details of mining the quantity of dust in the mine air can be kept below 1 milligram per 100 liters of air; so it seems reasonable to use 1 milligram as a standard at least for the Joplin district.

In this connection it might be well to discuss further a matter that was mentioned in the preliminary report of the work in the Joplin district [a] from which the four following paragraphs are quoted:

> The weight of rock dust contained in mine air in South Africa, Australia, and England [b] was found in some cases to be considerably higher than those set forth in Table 5. (See Table 7 in this report.) Haldane [c] states, as regards metal mines in

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[a] Lanza, A. J., and Higgins, Edwin, Pulmonary disease among miners in the Joplin district, Missouri, a preliminary report: Tech. Paper 105, Bureau of Mines, 1915, pp. 31-32.

[b] Haldane, J. S., Martin, J. S., and Thomas, R. A., Report to the Secretary of State for the Home Department on the health of the Cornish miners. 1904.

[c] Haldane, J. S., The investigation of mine air, 1908, pp. 121-122.

> Cornwall, that in the air of an "end" or "rise," with a rock drill at work boring dry holes, the air commonly contains 5 to 10 milligrams of dust per 10 liters.
>
> As pointed out previously the area of the connected open ground (on the same level) in the mines of the Joplin district is far greater than in the average metal mine. Obviously the amount of dust in a given volume of air would be much greater in a confined area, such as the face of a drift, in a raise, or in a stope than in a more open space.
>
> Moreover in taking the dust samples in the Joplin district the sampling bulb was invariably held near the head of the miner at work, so that the results obtained are not comparable with sampling closer to the source of the dust. Obviously there will be more dust in the air near the source than at points 5 to 10 feet distant. It is conceivable that larger dust particles may be collected close to the source of the dust and that these larger particles will greatly outweigh the smaller particles that would be found in suspension farther away.
>
> The records of the countries referred to, in extreme cases, indicated as high as ten times more dust (in comparatively confined places) than was found in the Joplin mines. It is doubtless true that there will be more dust produced per volume of air in drifts, raises, and stopes than is produced in the Joplin mines. The writer does not believe, however, for the reason stated (regarding the point at which the samples were collected), that the amount would be five or ten times greater.

Since the above statement was written all the samples included in Table 8 were taken. A small percentage of the samples contained from 50 to 100 milligrams of dust per 100 liters of air, which corresponds to the 5 to 10 milligrams per 10 liters referred to by Haldane. A study of this table shows that samples taken close to the source of dust contain much greater quantities than those taken farther away. The most striking instance is in the case of samples 45 and 46. Sample 45 was taken 1½ feet from a drill hole just being started and shows 221.71 milligrams of dust per 100 liters of air; sample 46, taken immediately afterwards, 8 feet from the hole, shows 2.28 milligrams of dust. It is thus apparent that widely varying results may be obtained in taking dust samples. It is the belief of the writer that samples taken close to the nostrils of the miner, wherever he may be in performing his duties, afford the closest approximation of conditions under which he must work. Sample 46, which was taken close to the nostrils of the drillman, 8 feet from the drill hole, shows that this miner was breathing air that contained 2.28 milligrams of dust per 100 liters; consequently it would be misleading and erroneous to consider sample 45 (221.71 milligrams) as representative of this working place.

ATTEMPTS TO ABATE SILICEOUS DUST IN THE MINES.

When C. M. Harlan, W. W. Holmes, and I. L. Burch entered on their duties in 1913 as deputy State mine inspectors for the Joplin district they were instructed to do all they could to abate siliceous dust and to improve sanitary conditions generally in and about the mines of the district. Although the inspectors made every effort to better conditions they were only partly successful, as there was no law under which their recommendations could be enforced. A few operators installed water lines and adopted certain other precautionary measures, but in the main the mines remained as dusty as before. Even in mines where water lines were installed it was found difficult to induce the miners to use the water for spraying. In spite of the difficulties under which they labored, the mine inspectors were insistent in their efforts to abate the siliceous dust. When the representatives of the Federal Government began their work the mine inspectors were anxious to assist and cooperate in furthering the work that they had started.

COOPERATIVE WORK OF FEDERAL AND STATE OFFICIALS.

This cooperative work between the Federal Government and the State of Missouri began with the arrival at Joplin, on November 7, 1914, of representatives of the Public Health Service and of the Bureau of Mines. In the course of the campaign for the improvement of sanitary conditions in and about the mines, which was intensively prosecuted for about four months, the Government representatives were given the active and earnest support of the State mine inspectors, civic organizations, mine operators, and, toward the last, the miners. The outstanding features of the campaign were the organization of the Southwestern Missouri Mine Safety and Sanitation Association, the preliminary report of the Government representatives, the effects of publicity and educational work, passage of laws relating to sanitation in and about mines, and the final vigorous and successful attempts to improve sanitary conditions.

SOUTHWESTERN MISSOURI MINE SAFETY AND SANITATION ASSOCIATION.

Early in the investigation it was recognized that an organization of operators would be of great value in furthering sanitary work in and about the mines. At a meeting of prominent operators the project was discussed and the result was the organization, during the latter part of November, 1914, of the Southwestern Missouri Mine Safety and Sanitation Association. Shortly thereafter a large hall in Webb City was obtained by the association as a meeting place, where luncheons were served every Thursday, after which business sessions were held. A secretary was employed to look after the affairs of the association.

Before the end of April, 1915, the association had enrolled 31 active, 30 associate, and 17 honorary members. The luncheons were attended by from 70 to 90 members and guests. On various occasions the business meetings were enlivened by addresses on subjects of interest, made by men of prominence.

The association appointed a special committee on sanitation, the members of which gave much time and thought to the betterment of sanitary conditions about the mines.

PRELIMINARY REPORT OF THE FEDERAL REPRESENTATIVES.

On the completion of the preliminary investigation, December 6, 1914, a report with recommendations was made to the operators of the district, through the Southwestern Missouri Mine Safety and Sanitation Association. A preliminary report was then prepared for publication by the Bureau of Mines as Technical Paper 105,[a] abstracts of which were released to various technical periodicals and to the Joplin newspapers on February 12, 1915.

This report contained the following recommendations[b] with regard to the abatement of rock dust:

1. Provide a water supply for every working face by the laying of a separate water line.
2. Where drills are operated without water, attach to the hose leading to the face a 5 or 6 foot length of pipe with a nozzle from one-eighth to one-fourth inch in diameter. Make and enforce such regulations as will insure the use of this water spray for the purpose of wetting drill holes, the face, and the broken rock about the face. For the purpose of washing drill cuttings from drill holes, this hose may be attached to the long pipes now in use for blowing out drill holes.
3. Where there is in use some type of drilling machine that provides for water passing through the core of the drill steel into the drill holes, make and enforce regulations that will insure the spraying of the face and broken rock for short periods at such times as the drill may not be in operation. For this purpose the water hose must be uncoupled from the drilling machine; this inconvenience may be overcome, however, by the use of a tee and a short length of hose.
4. Make and strictly enforce rules against squibbing and bowlder popping while the shift is underground and against the blowing of dry holes at all times.
5. Improve ventilation by the opening of new shafts whenever practicable.

In addition to the above recommendations, the following provisions[c] were offered for consideration in the framing of State laws dealing with sanitation in and about the mines:

1. That all operators of dusty mines (the question as to what constitutes a dusty mine to be left to the State mine inspector) be compelled to run a water line, in addition to the air line, to each working face, and to provide water for the line.
2. That every miner be compelled to make use of the water so provided, for the purpose of wetting down the broken ore, the working faces, and the drill holes from which dust may be spread.
3. That the practice of breaking bowlders by means of powder or dynamite be prohibited, except when the shift has left the mine.
4. That the blowing of drill holes, without first thoroughly wetting them with water, be prohibited at all times.

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[a] Lanza, A. J., and Higgins, Edwin, Pulmonary disease among miners in the Joplin district, Missouri, and its relation to rock dust in the mines: Tech. Paper 105, Bureau of Mines, 1915, 47 pp.

[b] Lanza, A. J., and Higgins, Edwin, work quoted, p. 33.

[c] Lanza, A. J., and Higgins, Edwin, work quoted, p. 44.

5. That managers or operators of mines be prohibited from maintaining a common drinking device.

In addition to the above suggested provisions the writers are of the belief that a workmen's compensation act would go far toward improving safety and sanitary conditions in the mines in the Joplin district.

RESULTS OF EDUCATIONAL WORK AND PUBLICITY.

Early in the investigation it became apparent that, in order to effect a permanent betterment of sanitary conditions, it would be necessary to obtain the cooperation of both the mine operators and the miners; and that such cooperation could be obtained if sufficient publicity were given to the ill effects of unsanitary conditions, especially the prevalence of siliceous dust in the mines. The writer, in company with Dr. A. J. Lanza, of the Public Health Service, attended many meetings of mine operators and various organizations, and talked on the subject of mine sanitation, and the cause, effect, and abatement of siliceous dust. The matter was given much publicity in daily newspapers. Miners and their families, to the number of 2,700, were addressed at three moving-picture shows. At these gatherings, in addition to films pertaining to sanitation and safety, slides made from photomicrographs of rock dust were shown. By means of talks to groups of from 25 to 50 miners at change houses, practically every miner in the sheet-ground district was reached.

At the start few miners gave evidence of interest in better sanitary conditions. However, as they began to acquire a knowledge of the ill effects of siliceous dust their attitude changed, and the miners as a whole became interested in the abatement of siliceous dust and the general improvement of conditions underground and on the surface. There were many instances of miners quitting their working places if they were not supplied with means of allaying the dust.

Many of the mine operators, without waiting for the passage of the laws requiring improved sanitary conditions, inaugurated improvements in and about the mines, such as the building of new and commodious change houses (see Pl. I, *B*), the equipment of the mines with separate water lines and sanitary drinking devices, and the regulation of the practices of squibbing and the blowing of dry holes.

LEGISLATION LOOKING TO IMPROVED SANITARY CONDITIONS.

Shortly after the preliminary investigation the mine inspectors and the sanitation committee of the association framed certain bills looking to the abatement of siliceous dust and the correction of other evils in and about the mines. Actively supported by the mine inspectors and others, these bills were enacted into law on the last day of the Missouri State legislative session, at the end of the third week in March, 1915.

These laws, which became effective July 1, 1915, are submitted herewith.

MINING LAWS OF MISSOURI, PASSED BY THE FORTY-EIGHTH GENERAL ASSEMBLY, 1914.

MINES AND MINING: INSPECTION OF MINES AND SAFETY OF MINERS—PROVIDING FOR SPRINKLING IN LEAD AND ZINC MINES.

SECTION 1. *Amending article 2, chapter 81, R. S. 1909, by adding two new sections.*—That article 2, chapter 81 of the Revised Statutes of Missouri for the year 1909 be amended by adding two new sections to be known as "section 4869a" (8469a) and "section 4869b" (8469b), providing for the maintenance of water lines for sprinkling purposes in all lead and zinc mines generating dust, prescribing certain duties of owners and employees in operating such mines and providing a punishment for violation of the provisions thereof, which sections shall read as follows:

SEC. 8469a. *Water lines to be maintained for sprinkling in certain mines generating dust—duties of inspectors, etc.*—The State mine inspectors for lead mines, zinc mines, and other mines, other than coal, are hereby authorized, empowered, and directed to thoroughly inspect all underground excavations in all such mines, as often as the inspector or either of his deputies may deem proper, for the purpose of ascertaining or discovering in the air in any such mine or mines the presence of dust in such quantities as shall be injurious to the health of employees engaged in working in such underground excavation; and upon finding dust in the air of any such mine in such quantities as shall tend to injure the health of the employees of such mine, such inspector or deputy inspector shall immediately notify the owner, managing agent, or operator of such mine, in writing, specifying the underground excavation so found to contain dust particles as aforesaid in the air thereof, and such owner, agent, or operator of such mine shall within 15 days after receiving such written notice, provide, install, equip, and thereafter at all times maintain in such mine an independent water line, fully equipped and in good serviceable working order and repair, leading up to the face of any and all drifts where such dust is produced, or so close to the face of said drifts so that by the use of suitable hose extension or sprinkling attachments to be supplied by the owner or owners of said mine, the mineral or earth in and adjoining the face of the drift or drifts of such mine can be sprinkled or wet by water from said pipe line; thereupon and thereafter every person drilling, squibbing, or blasting in said mine shall keep the face, surface, and drill holes in said drift or drifts wet or moist by the use of water from said water line to such an extent and in such a way as shall prevent, as far as possible, any dust raising from the working of any such face or from the drilling, "blowing" or "shooting" of any hole or holes; and the ground boss in charge of the underground in any such mine, so equipped with a water line, shall require all ground or dirt after being shot or blasted to be thoroughly wet or sprinkled to such an extent as shall prevent, as far as possible, any dust from arising therefrom while the employees are at work therein.

SEC. 8469b. *Penalty.*—Every owner or part owner of any such mine and every employee of the owner of any such mine, who shall violate any of the provisions of this act, shall be deemed guilty of a misdemeanor and upon conviction thereof shall be punished by a fine of not less than $5 nor more than $50.

Approved, March 23, 1915.

MINES AND MINING: INSPECTION OF MINES AND SAFETY OF MINES—PROMOTING HEALTH AND SANITARY CONDITIONS OF LEAD AND ZINC MINERS—PROVIDING DRESSING ROOMS, ETC.

SECTION 1. *Amending Article II, chapter 81, R. S. 1909, by adding a new section thereto.*—That Article II, chapter 31 (81), of the Revised Statutes of Missouri for the year 1909, be, and the same is hereby, amended by adding a new section thereto, to be known as section 8469b, providing dressing rooms for employees of all owners and operators of lead and zinc mines, which section shall read as follows:

SEC. 8469b. *Dressing rooms to be provided—Equipment—Inspection—Penalty.*—It shall be the duty of every owner or operator of any lead or zinc mine in the State of Missouri to provide and maintain a room or building of sufficient size and dimensions and properly equipped, for the use of employees of said mines, as a dressing room and for the purpose of changing, keeping, and storing their clothes and dinner pails. Said room shall be equipped with lockers with lock and key, and said employees shall be permitted to store their clothing and dinner pails in said lockers. Sufficient washing conveniences shall be provided in said room or building for the use of said employees, and sufficient benches or seats shall be provided for the use of employees in said room or building; and said room or building shall at all times be properly heated and shall be kept in a clean and sanitary condition. It shall be the duty of the mine inspector to see that the provisions of this section are properly enforced. Any person, firm, or corporation operating a lead or zinc mine in this State failing to comply with the provisions of this section shall be guilty of a misdemeanor and upon conviction thereof shall be fined in a sum of not less than $5 or more than $25.

Approved, March 23, 1915.

MINES AND MINING: INSPECTION OF MINES AND SAFETY OF MINERS—SANITARY DRINKING DEVICES FOR USE OF EMPLOYEES.

SECTION 1. *Amending Article II, chapter 81, by adding a new section thereto.*—That Article II, chapter 81, Revised Statutes of Missouri, 1909, be, and the same is hereby, amended by adding one new section thereto, to be known as section 8469c, which said section shall read as follows:

SEC. 8469c. *Sanitary drinking devices.*—Every owner, agent, or operator of any lead or zinc mine in this State, employing 10 or more men, shall provide sanitary drinking devices for the use of their employees.

Approved, March 23, 1915.

MINES AND MINING: INSPECTION OF MINES AND SAFETY OF MINERS—INSPECTORS TO CLOSE MINES, WHEN.

SECTION 1. *Inspectors to close mines, when.*—The chief mine inspector and his assistants shall have the power, and it is hereby made their duty, to stop the operation of and close any mine or part thereof, where poisonous damps exist, where rotten ropes or unsafe cages are used, or where a safe escape way is not provided for all employees. Any person or persons violating the provisions of this section, and any member or stockhold (er) or officer of any company or corporation who shall violate the provisions of this section, shall be deemed guilty of a misdemeanor, and on conviction thereof be punished by a fine of not less than $25 nor more than $100, or imprisonment in the county jail not less than 30 days nor more than 90 days, or by both such fine and imprisonment, for each and every separate offense.

Approved, March 22, 1915.

MINES AND MINING: INSPECTION OF MINES AND SAFETY OF MINERS—PROMOTING HEALTH AND SANITARY CONDITIONS OF COAL MINERS—PROVIDING WASHHOUSES AND EQUIPMENT.

SECTION 1. *Washhouses and equipment—Sanitary conditions, etc.*—It shall be the duty of every person, corporation, or company, and of his or its agents, officers, representatives, or person or persons in charge of, owning and operating or operating as lessees a coal mine within the State of Missouri, wherein 10 or more miners are employed in digging coal to provide within six months after the approval of this act a suitable building to be used by said miners as a washhouse, of a size not to exceed 12 feet in width and 20 feet in length and one story in height, and to be located within a reasonable distance, not to exceed 300 feet, of the shaft house of the said mine, for the accommodation of the miners who desire to use the same; said washhouse to be equipped with a stove or other heating apparatus, and if a stove be furnished that the fuel to be burned in the same to be furnished by the operator of said mine; said washhouse shall be kept in a good and sanitary condition by the miners who use the same for the purpose of washing themselves and changing clothing when going to and returning from the mine. The room for the negroes shall be separate from that for the white race, but may be in the same building. An available supply of water shall be furnished by the operator and the same shall be carried to said washhouse by the miners; the miners shall furnish their own towels, basins, and soap, and shall exercise control over and be responsible for all property by them left therein.

SEC. 2. *Liability for loss or destruction of property.*—No person, corporation, or company, its agents, officers, or representatives, furnishing such a washhouse at his or its mines as required in section 1 hereof shall be legally liable for the loss or destruction of any property left at or in said washhouse.

SEC. 3. *Penalty.*—Any person, corporation, or company, its officers or agents, failing, neglecting, or refusing to comply with the provisions of section 1 of this act shall be guilty of a misdemeanor, and shall upon conviction be fined in any sum not less than $25 nor more than $100. Each week that such person, corporation, or company fails and neglects to comply with the provisions of said section shall constitute a separate offense: *Provided,* This act shall not apply to strip-pit or steam-shovel coal mining.

Approved, March 22, 1915.

SUMMARY AND CONCLUSIONS.

The principal facts relating to the prevalence and abatement of siliceous dust in the sheet-ground mines, and the conclusions arrived at as a result of the investigative work done are summarized as follows:

1. Siliceous dust is produced in all of the sheet-ground mines in varying quantities, regardless of whether the working places are damp or dry. Even very wet mines may be dusty if excessive squibbing is permitted.

2. Dust is produced underground by drilling, squibbing, blowing dry holes, blasting, shoveling, bowlder popping, tramming, and roof and pillar trimming. The greatest quantities of dust are produced by the blowing of dry holes; next in importance are blasting, squibbing, and bowlder popping. The dust produced by blasting, however, is not often troublesome, as it is the usual practice to blast as the shift goes off duty.

3. The dust produced is densest at the working faces, but permeates all parts of the mines.

4. The dust is composed of particles of cherty flint and contains usually more than 90 per cent siliceous residue. There are two distinct kinds of dust particles, one milky white and the other transparent and glassy. The milky white particles are generally thin, sharp edged, irregular fragments, often bladelike or knifelike in shape. The glassy particles are usually roughly polyhedral or sphe-

roidal, although some are sharp-edged or pointed. The dust particles are of almost every possible shape and form, but nearly all particles have one or more sharp edges.

5. The particles of dust in the mine air range in size from those having dimensions of 70 to 80 microns to those that are invisible under the ordinary microscope. The smallest particle measurable under the microscope was from one-fourth to one-half micron. The smallest particle visible was about 20 millimicrons, 20 thousandths of a micron. The number of these smaller particles is enormous.

6. Dust particles with dimensions of 50 to 70 microns settle within a few minutes. One hour after the cessation of the operation causing the dust, in comparatively still air, few particles larger than 10 microns remain in suspension. After three hours the largest sizes remaining in suspension have a greatest dimension of 5 microns. After four hours few particles as large as 5 microns are to be found; a small percentage will range from 3 to 4 microns, but the bulk of the particles will be less than 1 micron.

7. Most of the dust particles that reach the lungs range from 2 to 5 microns, with rapidly increasing numbers of the smaller sizes; comparatively few particles with any dimension more than 10 microns are to be found. Larger dust particles, however, find lodgment in the air passages leading to the lungs, even gritty particles being intercepted in the nose and mouth.

8. In 222 samples of dusty air, the greatest quantity of dust obtained was 221.71 milligrams per 100 liters of air. A large proportion of the samples show a content in the mine air less than 10 milligrams, and the average dusty atmosphere contained from 3 to 5 milligrams per 100 liters.

9. By proper precautions, the amount of dust in the air may be kept down to 1 milligram and less per 100 liters. The requirements and precautions necessary to accomplish this end are as follows:

Improve ventilation, where practicable, by installing fans, or by properly locating new shafts. This is an important consideration.

In mines wet enough to produce only thoroughly damp dirt, prohibit squibbing and blasting while the shift is underground. In all other mines the following precautions should be observed:

(*a*) At each working face provide a water supply of sufficient head to throw a stream at least 20 feet through a nozzle one-fourth inch in diameter. The best arrangement is a main pipe line from the surface or the water column, the water being carried from the main line to the face by 1-inch wire-wrapped rubber hose.

(*b*) The water so provided may be used with any type of water-injection drill. Provision can be made also for spraying the face and broken rock by inserting a tee connection at the machine and attaching a hose and nozzle. The dirt pile should be frequently wetted in order to prevent dust being raised by shovelers.

(*c*) Where drills using solid steel are employed, a pipe long enough to reach the back of the drill hole and having a nozzle formed at one end should be coupled to the rubber hose. The spray thus provided should be used often enough during drilling to keep the drill hole damp; it should also be used for spraying the working place and broken rock at sufficiently short intervals to keep them damp. The dirt pile must be kept damp.

(*d*) Prohibit the blowing of dry holes. When necessary, wash the drill cuttings from the drill holes by means of the spray.

(*e*) Prohibit blasting and squibbing while the shift is underground, or just before the time that the shift goes to work.

(*f*) Where more than one shift is worked, allow at least two hours (preferably more) between blasting at the end of a shift and the beginning of work by another shift.

(*g*) Prohibit the popping of bowlders while the shift is underground.

10. The campaign for the abatement of siliceous dust and the improvement of sanitary conditions in and about the mines was markedly successful. This outcome resulted from the educational work conducted, the cooperation of the miners, operators, and those carrying on the campaign, and the passage of State laws regulating the conduct of mining. Similar laws in other mining districts have proved ineffective. In the Joplin district the laws were effective because of the educational work, and because of both operator and miner being penalized in case of noncompliance.

NOTES REGARDING HEALTH CONDITIONS IN METAL MINES IN FOREIGN COUNTRIES.

As indicating the consensus of opinion, and to draw attention to what has been done in other countries by way of legislation, there is included here a set of notes bearing on health conditions in mines in the Transvaal, western Australia, Great Britain, and other countries.

REPORT ON HEALTH OF CORNISH MINERS.

In the report[a] to the secretary of state for the home department on the health of Cornish miners, the following summary and recommendations are set forth:

SUMMARY.

1. The death rate among miners living in Cornwall, which has always been very high in the case of men over about 40, has very greatly increased during the last few years among men of from 25 to 45.

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[a] Haldane, J. S., Martin, J. S., and Thomas, R. A., Report to the secretary of state for the home department on the health of Cornish miners, 1904.

2. The excessive death rate is shown by the statistics to be due entirely to phthisis and other lung diseases.

3. The recent increase in the death rate is also shown to be due to the deaths of men who have worked rock drills. The great majority of these deaths are attributable to the effects of rock-drill work in the Transvaal or elsewhere abroad; but a considerable number are attributable to work in Cornwall.

4. Nearly the whole of the deaths of rock-drill men were due to "phthisis," and of the cases examined at least 74 per cent were tuberculous.

5. The predisposing cause of the present excessive mortality among metalliferous miners from lung diseases is evidently the inhalation of stone dust.

6. The dust is produced in the drilling of dry holes by rock drills, in blasting, in handling the ore, and in other ways.

7. The dust produced by rock drills can easily be prevented by even a very small water jet.[a] The dust from blasting in close ends can be laid by a powerful jet of water and air; and can in any case be avoided by the men. The inhalation of dust produced in blasting on the stopes can also be to a large extent avoided, while the dust produced in handling the ore can be mostly prevented by keeping the workings damp.

RECOMMENDATIONS.

1. That the use of percussion rock drills in hard stone without satisfactory precautions for preventing the dust being inhaled by the men be prohibited in all mines.

2. That special rules be established under the metalliferous mines regulation act by the management of every metalliferous mine, subject to the approval of the secretary of state, for the carrying on of the work in such a manner as to reduce to a minimum the inhalation of dust by the various classes of men employed in the mine.

3. That special rules be also established under the same act, rendering it obligatory on the part of owners of metalliferous mines to provide and maintain in a suitable and cleanly condition a reasonable number of sanitary receptacles for the use of men in case of necessity underground, and also on the surface; and rendering it a contravention of the act unnecessarily to pollute any part of a mine with human feces.

The three main factors directly incidental to their occupation, which combine to produce diseases among the miners on the Witwatersrand are dust, infective processes, particularly tuberculosis, and in a lesser degree the fumes rising from the use (or abuse) of explosives.[b] This statement was made by Dr. Irvine in his evidence before the mining regulations commission in May, 1908, and with that statement there must be universal agreement.

Dust.—This constitutes the greatest danger which the underground worker has to combat. The sources of dust are: (*a*) Rock drilling, (*b*) blasting, (*c*) handling of rock. It must be acknowledged that the proper use of water in conjunction with rock drilling effectually allays the dust generated during the operation. In the face of this acknowledged fact, it is astounding that to-day this remedy has not, in many of our mines, been completely carried out. The proper use of water implies: (1) An adequate supply of clean water, (2) efficient application of the same.

With regard to this point, the law already has the power to insist on its enforcement (mines and worker's regulations, act 146, an amendment published in Government Notice No. 1278 of 1908.)

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[a] A jet with an orifice one-twentieth inch in diameter directed into the hole was recommended by the signers of the report. It may be pointed out, however, that this was prior to the time of drilling machines furnishing water through the core of the drill.

[b] Penlerick, S., Ventilation and health conditions in the mines of the Witwatersrand: Jour. Chem. Met. and Min. Soc. South Africa, vol. 10, 1910, p. 59.

MINING REGULATIONS IN SOUTH AFRICA.

The following extract[a] from an article in the Mining Magazine discusses provisions of the mines regulation act of 1912, that relate to rock-dust conditions in South Africa mines:

> The new mines regulation act embodied in act No. 12 of 1911, to consolidate and amend laws in force in the Union of South Africa, relating to the operations of mines, works, and machinery, and to certificates of competence, came into force on December 1, 1911. The old laws of the Provinces of Natal, Cape Colony, and Orange Free State are either wholly or partly repealed.
>
> The new regulations make a water spray or some other suitable method for allaying and removing the dust imperative.

MINING REGULATIONS OF NEW ZEALAND.

The mining act of 1906, No. 120, New Zealand (sec. 241) provides as follows:

> In every case where quartz or other substances are crushed in a dry state, or where rock drills are used, there shall at all times be used in and about the battery or place where such crushing or drilling is done such appliances as in the opinion of the inspector shall effectually keep the air pure and prevent dust circulating in the place where such operations are being carried on, and for this purpose an adequate supply of water shall be provided: *Provided*, That where either the owner or workmen's inspector is dissatisfied with the opinion of the inspector an appeal shall lie to the warden, whose decision shall be final.

The act of 1910 (No. 780), entitled "An act to amend the mining act of 1908," provides (art. 9):

> It shall not be lawful for the owner or manager of any mine, or for any person in charge of the mine to require any person who is employed in the mine, or applying to be employed, to be medically examined or to produce a medical certificate that he is in a good and sound state of health.

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[a] Editorial, Mining regulations in South Africa: Mining Mag., Feb., 1912, p. 123.

PHYSIOLOGICAL EFFECTS OF SILICEOUS DUST ON THE MINERS OF THE JOPLIN DISTRICT.

By A. J. LANZA.

INTRODUCTION.

A preliminary report dealing with the prevalence of pulmonary diseases among the zinc miners of the Joplin district has been published by the Bureau of Mines as Technical Paper 105.[a] In February, 1915, the investigation upon which the preliminary report was based was renewed and continued until the end of the calendar year 1915.

In the preceding chapter Mr. Higgins has given a description of the zinc mines of the Joplin district, the nature of the ground, the causes of rock dust and conditions under which it is found, and other technical data.

Soon after the investigation was renewed, under a cooperative agreement between the United States Public Health Service and the Bureau of Mines, an office and a laboratory were established in Webb City, Mo., in which to conduct physical examinations of miners, to make microscopic examinations of samples of sputum, and to collect pertinent data. A visiting nurse was attached to the office, who, on account of long acquaintanceship in the district and wide familiarity with the miners' families, was of great service in the collection of data on housing and on family histories. The office was kept open until the close of the year 1915.

The writer desires to thank the miners of the Joplin district for their readiness to cooperate in this work; the mine operators, the State mine inspectors, and the Jasper County antituberculosis society for their constant interest and many kind services; the physicians, who encouraged their patients to come to the office for examination, and other individuals who, by their constant moral support, furthered the cause of the investigation.

EDUCATIONAL WORK.

Among the recommendations of means that should be employed for the improvement of existing conditions outlined in the conclusion of the preliminary report was the following: "Through intensive educational campaigns in the public schools, and among the miners themselves, disseminate information as to the harmful effects of insanitary practices and conditions, such as crowded living quarters, overwork, exposure, dissipation, the breathing of air polluted by powder fumes and siliceous rock dust, and the use of common drinking devices."

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[a] Lanza, A. J., and Higgins, Edwin, Pulmonary disease among miners in the Joplin district, Missouri, and its relation to rock dust in the mines; a preliminary report: Tech. Paper 105, Bureau of Mines, 1915, 48 pp.

The first work undertaken by the investigators in February, 1915, was educational. Visits were made to the sheet-ground mines at the noon hour, and while the men were eating their dinner in the change houses they were addressed by both Mr. Higgins and the author. These short talks were on the causes of rock dust, its harmful effects, and how dust could be avoided, with emphasis upon the fact that, regardless of any rules or regulations in force, their salvation from pulmonary diseases rested largely with themselves. The men were urged to come to the office in Webb City for physical examination. They were encouraged to ask questions, and the extent to which they did this, together with their strict attention everywhere, evidenced their appreciation.

By direct invitation and by advertisement word was spread around the district that miners and their families would be examined free of charge at the office and would be advised as to their physical condition. In this manner there was collected a considerable amount of information as to the physical condition, not only of the men themselves, but also of their families. More than this, however, there was thus created an opportunity for doing considerable effective work among these people by direct personal advice and instruction. Every man examined was informed fully as to his physical condition. When necessary he was advised of the proper means for caring for himself and how to prevent the spread of infection to his family. He was encouraged to return for further examination or advice and was urged to bring his wife and family for examination when there was any reason to suspect that they might not be in good health. Educational literature was distributed freely at the office.

Addresses were made to the high schools in Webb City and Joplin, to the Joplin noonday lunch club, and at the weekly meetings of the southwest Missouri mine safety and sanitation association; also to the civic league of Carterville, and similar organizations. Addresses, illustrated with moving pictures and with lantern slides, showing magnified particles of rock dust obtained from air samples in the mines, were given at local theaters for the benefit of miners and their families. The local press cooperated whenever possible, and as a result of these varied efforts the attention of the community was thoroughly aroused to the points at issue.

In connection with the educational work, it is necessary to call attention to the fact that, as a result of the high prices of zinc ore that have prevailed for the past year, and the greatly increased activity that has resulted therefrom in southwestern Missouri, there has occurred an influx of miners from other places, until at the present time (January, 1916) it is probable that there are almost twice as many men working in the sheet-ground mines as there were a year previous. Many of these men are from coal mines, some are from metal mines, and others have never mined before. The men new to mining and the coal miners are largely ignorant of the effects of breathing hard-rock dust, and if the sanitary reforms already accomplished are to be maintained these newcomers must receive instructions similar to those outlined above. The State mine inspectors make this one of their duties, and it is probable that local sanitary organizations will be able to assist in this work.

PREVALENCE AND NATURE OF PULMONARY DISEASES.

In the preliminary report mentioned above is a table showing the deaths from pulmonary tuberculosis in Jasper and other Missouri counties for the years 1911, 1912, and 1913. Briefly, this table shows that, in respect to the disease mentioned, Jasper County leads St. Francis, Green (including Springfield), Jackson (including Kansas City), and Buchanan (including St. Joseph) Counties and the city of St. Louis, in the number of deaths per 100,000 of population. The difference between Jasper County and St. Francis County is significant: Each is a metal-mining county; each employs the same class of labor; and the general conditions are about the same in both, with this exception—hard-rock dust is not present in the southeastern Missouri mines.

A system of death reports alone does not give adequate information as to the status of a chronic disease like tuberculosis in any one place, as the disease may have been contracted elsewhere. The mining population in the Joplin district that does not represent residents of Missouri is recruited mostly from Kansas and from northern Arkansas. Most of the miners who have previously followed some other occupation have been farmers. When the victims of consumption are disabled, they are apt to return home, eventually dying there, and the cause of their death is charged, so to speak, against the wrong State or county.

To gain a more accurate idea of the amount of consumption among the miners in the Joplin district, it was decided to conduct a physical examination of a large number. It was not possible to go to any mine or group of mines and examine the entire personnel, as facilities for examining men at the mines were totally lacking. Consequently, it was necessary to fall back on the plan of having the men come to the Webb City office for examination.

During the time this office was open—May 15 to December 31, 1915—over 700 miners were examined, a number of them more than once. In each case an effort was made to determine whether the miner was suffering from chronic disease of the lungs, and whether such disease or disablement was due to tuberculosis or to miners' consumption (rock-dust consumption), or both. Other abnormalities of the lungs and heart were also noted. Before the results of these examinations are discussed the effects of hard-rock dust on the lungs may be briefly considered.

NATURE OF MINERS' CONSUMPTION.

The continuous inhalation of the exceedingly minute particles of sharp and insoluble flint that constitute the dust in the sheet-ground mines of the Joplin district produces a mechanical injury to the lungs, causing a disease peculiar to the mining industry, and known variously as miners' phthisis, miners' asthma, miners' consumption, silicosis, or, more accurately, pneumoconiosis due to siliceous rock dust. As the term "miners' consumption" has become widely prevalent in this country, and as it is descriptive of the condition itself, it will be retained throughout this report.

Mechanically produced, miners' consumption is neither contagious nor infectious. The lung, irritated, inflamed, and injured by the hard-rock dust, becomes fibroid—that is, there is developed throughout it a scarlike tissue, which interferes with the normal elasticity of the lung and prevents proper breathing. The disease may attack one or both lungs, but is usually evenly distributed on both sides. Miners' consumption has long been recognized in the mines of the Witwatersrand in South Africa, in the Bendigo district of Australia, in Cornwall, and elsewhere. As this form of consumption becomes established, there is a gradual impairment of working ability, owing to shortness of breath, with coughing, loss of weight, and weakness. There is also a disposition to "catch cold" easily, and the cold is apt to settle on the chest. The intensity of miners' consumption varies, all grades of severity being seen, depending on the nature of the dust and the length of time spent underground, or other extent of exposure. It may persist for years and tends gradually to produce total disability.

There is, however, a deeper menace than gradual disability, in that the injured lung, together with the lowered vitality of the whole body, presents an ideal site for the development of tuberculosis. When this occurs, not only is the miner threatened with an early death or, at least, a more hopeless future; but, having now a contagious disease, he becomes a source of danger to others—his fellow workmen and his family. A tuberculous infection may take place at any stage of miners' consumption, and it is the opinion of the writer that practically no cases die without its being present. At least such was the experience in southwestern Missouri. All cases examined that were obviously in a hopeless or dying condition showed tubercle bacilli in the sputum. In other cases, either where the men were still working, or where they were disabled, tubercle bacilli were sometimes present and sometimes not.

It is a frequent occurrence in the Joplin district, and one noticed a number of times by the writer, for a miner who for some time, perhaps several years, has had an increasing shortness of breath and an increasing impairment of working ability, suddenly to "blow up" and die after an illness of a few weeks or months. Such cases showed tubercle bacilli in the sputum. A tuberculous infection, spreading with great rapidity in a favorable soil, often helped by overwork and alcoholic dissipation, had run an acute course and brought the case of miners' consumption to a conclusion. Tuberculous infection would seem to occur most often after the mechanical disease is well established, but it does occur sooner or later in most cases.

In this connection Summons[a] says:

> It can safely be concluded that at the present time all Bendigo miners dying of their respiratory diseases die of tuberculosis.

The report of the miners' phthisis prevention committee of the Union of South Africa for 1912 states:[b]

> In at least the great majority of cases tuberculous infection becomes toward the end superimposed upon the preexisting silicosis.

These statements coincide with the experience of the writer during this investigation.

The concomitant occurrence of tuberculous infection becomes doubly important as regards hygiene and health to a community. Thus, when a miner is finally unable to work, and more or less bedfast, and on account of his advanced disease can no longer gain admittance to a sanitarium, or has been sent home from one as hopeless, and when finally the poverty consequent on long illness has resulted in forlorn and utterly insanitary surroundings for himself and family, then is the time when his sputum is loaded with tubercle bacilli, the danger of which to his wife and children is ignored by all concerned, because, say they, "He has miners' consumption, which is not contagious like the old-fashioned kind." As regards the public health, so far as the prevention of tuberculosis is concerned, both for the benefit of the family and of the other miners with dust-injured lungs, there should exist no distinction as to the treatment of a case of miners' consumption and of a case of tuberculosis, especially in regard to the careful disposal of the sputum in and around the home, as in any given case tubercle bacilli may appear at any time. From observation the writer is inclined to believe that certain cases may have tubercle bacilli in the sputum intermittently—that is, the bacilli may be absent at one time and present in quantities at another time. Hence, there is necessity for care at all times.

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[a] Summons, Walter, Report of an investigation at Bendigo into the prevalence, nature, causes, and prevention of miners' phthisis, 1907, p. 42; quoted in Report of Commission to Investigate the Prevalence of Miners' Phthisis and Pulmonary Tuberculosis on mines within the Union of South Africa, 1912, p. 6.

[b] Report of Commission to Investigate the Prevalence of Miners' Phthisis and Pulmonary Tuberculosis on mines within the Union of South Africa, 1912, p. 8.

Overwork, exposure, alcoholism, and poor living conditions are the general causes of tuberculosis among working people everywhere. When we add to these the hazard of rock dust, with its consequent injury to the lungs, we have an explanation of why so many metal miners die of tuberculosis. The conditions first named are often beyond the influence of any but the individuals concerned, but the sanitation of their working places and the elimination of rock dust can be controlled by mine operators through proper supervision, legislation, and the education of all concerned. Sanitary management of mines will be an accomplished fact when both the miners and their employers realize that there is just as much connection between poor sanitation and death through disease as there is between carelessness and death through accident; only the loss of life from disease far exceeds that from accident.

RESULTS OF PHYSICAL EXAMINATION OF 720 MINERS.

In all, 720 miners were examined, and as a result of the examinations they were grouped as follows:

Results of physical examination of 720 miners.

Classification.			Number.	Per cent.
Seemingly well			179	24.8
Suffering from miscellaneous diseases			26	3.6
Doubtful cases			43	5.9
Suffering from tuberculosis, seemingly not connected with siliceous dust injury to the lungs			39	5.3
	Number.	Per cent.		
First stage	2	0.2		
Second stage	13	1.8		
Third stage	24	3.3		
Suffering from miners' consumption			330	45.7
First stage	120	16.6		
Second stage	142	19.7		
Third stage	68	9.4		
Suffering from miners' consumption with a tuberculous infection			103	14.7
First stage	8	1.1		
Second stage	22	3.5		
Third stage	73	10.1		
			720	

As bearing on the value of the figures given above, it should be stated that usually men who were beginning to be seriously affected by shortness of breath and to be alarmed as to their condition were most apt to come to the office, accounting for the preponderance of men in the first and second stages of miners' consumption. When their disease was further established they often refused to be examined, dreading to be told that they were in a hopeless condition. When, however, they believed their condition was hopeless, they were inclined to come in, like drowning men clutching at a straw, hoping that they might get some relief.

It was difficult, often impossible, to get samples of sputum, for although the miners were anxious for physical examination, they did not want to be told they had "bugs" in their sputum. They themselves clearly differentiated between what they thought was miners' consumption and the "old-fashioned kind," and nearly always maintained, even on their deathbeds, when their sputum was loaded with tubercle bacilli, that they did not have "regular consumption." They were largely correct, only they failed to appreciate the dangers of infection from the "irregular" kind.

The table shows that out of 720, 472 were suffering from pulmonary disease; of these 404, or 56 per cent of the whole, were working at the time examined or up to within six months previous. There were 433, 60.4 per cent, suffering from disease of the lungs directly due to rock-dust exposure; of these, 375, or 52 per cent of the whole, were working at the time of examination, or up to within six months previous. However, these figures do not justify the statement that half of the miners in the sheet-ground district had consumption; but in view of the chronic nature of miners' consumption and of tuberculosis, it is safe to say that practically all of these were in a diseased state during the year 1914. The total number examined represents about one-fourth of all men working in sheet-ground mines up to the boom of 1915. Therefore, of all the sheet-ground miners at that time, 15 per cent had consumption. On a conservative estimate, the writer believes that the actual number of cases of miners' consumption was at least twice as great. At the present time (January, 1916) there are probably twice as many men, if not more, working in the Joplin district as formerly; and the extent of the prevalence of lung diseases now is mere conjecture.

CASES OF TUBERCULOSIS WITHOUT ROCK-DUST INJURY.

As regards the 39 men who were found to be suffering from pulmonary tuberculosis and whose lungs showed no evidence of rock-dust injury, their histories indicated that several of them were tuberculous before they started mining work.

The two men in the first stage are still working. Of the 13 men in the second stage, 10 were working at the time of examination and 3 had stopped in the previous two months. One has since become disabled. Of the 24 men in the third stage, 6 were still working and 8 had quit at various times within the previous six months; 5 have died up to the time of writing.

CASES OF MINERS' CONSUMPTION ONLY.

The table shows that 330 miners manifested definite signs of injury to the lungs from siliceous dust, but as far as could be determined were free from tuberculous infection. Of the 120 in the first stage, 107 were still working underground and 11 had quit in the previous two months. Of the remaining 2, one had not mined for a year, the other for four years. The 13 who had quit mining were all doing some kind of surface work. In this group was a former coal miner, who probably had anthracosis before coming to the Joplin district.

Of the 142 men in the second stage, 123 were still working underground and 10 had quit the ground in the previous six months. The remaining 9 had not mined for one to six years previously. One man has since been killed, and 6, as far as is known, have stopped underground work since their examination. Two men had seemingly contracted their disease in other mining districts before coming to Missouri, and another had anthracosis when he came. Among those who no longer mine there are a teamster, a shoemaker, a machinist, a farmer, a man who works in the harvest fields, and another who runs a shooting gallery.

Of the 68 men in the third stage, 19 were still working steadily underground and 5 were working intermittently—when they were able; 22 had stopped working within the previous six months. Of the remaining 22, 11 had quit in the past year and the other 11 had not worked for periods up to seven years. Two have stopped work since being examined and 4 have died. Three are now farming or ranching, 1 works in a shoe factory, 1 is a cobbler, and 1 is a night-watchman. On account of their pronounced tendency to shortness of breath, there is not much that these men can do when they leave the mines, and as a class they are entirely unfamiliar with sedentary occupations.

CASES OF MINERS' CONSUMPTION WITH A TUBERCULOUS INFECTION.

As the table shows, there were 103 men who, besides having miners' consumption, showed symptoms of tuberculosis or had tubercle bacilli in their sputum. Of the 8 in the first stage, 7 were still working underground and 1 had laid off in the previous month. Of the 22 in the second stage, 19 were still working underground and 3 had laid off within the previous two months.

Of the 73 in the third stage, 25 were still working underground, 5 worked intermittently, and 18 had quit mining within the previous six months; 12 had quit within the previous year, and the remaining

13 had not worked underground for periods ranging to eight years; 4 have stopped work since being examined, and 15 have died (as far as known). One is now a paperhanger, 1 a carpenter, and 1 drives a wagon.

DOUBTFUL CASES.

The doubtful cases include men who had symptoms resembling miners' consumption, but on examination were found also to have heart or circulatory troubles, and concerning whom, therefore, there was a legitimate doubt as to the real cause and nature of their disability. Of the 43 men in this class, 35 were still working, 6 had worked until some time in the previous six months, and the remaining 2 had not mined for some years. Two have died since examination.

MINERS SUFFERING FROM MISCELLANEOUS DISEASES.

Included among the miners examined were men who were found suffering from heart, kidney, or other chronic organic disease not involving the lungs. Of the 26 men in this group, 25 were still working and 1 had stopped two months previously.

MINERS SEEMINGLY WELL.

There were 179 men who were seemingly well, of whom 169 were still working underground when examined, 7 had laid off within the previous six months, and 2 in the previous two years.

For convenience, the cases of consumption enumerated above have been divided into three stages—first, second, and third. In the first stage of miners' consumption are included those cases in which the sufferers showed a slight or moderate tendency to shortness of breath and some diminished expansion of the chest. Generally there was some pain in the chest, coughing, and expectoration, but no marked impairment of working ability. The second stage includes cases in which the sufferers showed moderate or moderately severe shortness of breath on exertion, diminished expansion of the chest, and well-defined impairment of working ability; with or without other symptoms. Those sufferers classed as being in the third stage showed a severe shortness of breath, with more or less total disability and aggravated symptoms of various kinds.

GENERAL REMARKS ON PHYSICAL EXAMINATIONS.

Throughout these examinations, the symptoms presented were interpreted in a conservative manner, and the errors of diagnosis are in favor of the group "seemingly well." It was recognized that men who had worked underground only a short time often exhibited some shortness of breath and coughing, with irritation of the throat and the bronchial passages, owing to the unfamiliar work and the dust and powder smoke. Effort was made not to include these cases of acute irritation of the upper air passages with those of early lung trouble. Where there was doubt, the miner was classified as seemingly well and urged to return for reexamination. Some of these cases did not return and were lost sight of and so remain as first recorded. There are not enough of them to appreciably change the percentages.

The nature of the work done seemingly has little influence on the inception of miners' consumption. Underground the bulk of the workers are either shovelers or machine men and their helpers, and all of these are equally exposed to dust. Most of the men examined, especially those who had mined for several years, had done both machine work and shoveling. A few cases of miners' consumption were noted in millmen who had not worked underground, but had inhaled the rock dust set free by the crushers.

SANITARY CONDITIONS UNDERGROUND.

PREVALENCE OF SILICEOUS DUST.

The causes of siliceous rock dust, its nature, and methods of prevention have been set forth. During the past year many mines have installed separate water lines to the heading, and the recent laws providing for the prevention and laying of dust are being complied with more and more. The miners are becoming accustomed to using water, and the efficiency of dust prevention is increasing. As a result of the educational campaign there is not much objection to the use of water by the men, and when they do fail to use it where provided such failure is rather through carelessness than intent. Now a man "blowing a dry hole" is apt to meet decided objection from those working near him. Men who do not have to work strenuously sometimes wear respirators, but there are very few who can wear them comfortably, on account of the impediment to breathing they cause, and at best respirators seem to be of little or no value, owing to the fact that they do not filter out the finer dust particles.

In a few mines fans have been installed and have aided greatly in clarifying the air. In one mine a large suction fan was placed on the surface and in another a blower fan keeps a current of air moving along the headings.

Where separate water lines are established, the "dummy" (machine helper) plays the water against and into the drill hole while the machine man runs the drill. In blowing the hole, the blowpipe is fitted into the water line instead of the air line, as formerly, and the hole is blown out from the bottom by water. When not otherwise in use, the water is allowed to run over the dirt pile, so that the dirt (ore) is wet when shoveled. This practice minimizes the likelihood of injury from dust, both for the shoveler and for the men who work on the screen where the ore is dumped.

Drills of the type in which water is fed through hollow steels have not come into general use, but it is probable that they will gain in favor. It should be remembered that water drills will not prevent dust where promiscuous squibbing and bowlder popping are allowed. Under these circumstances there is bound to be dust in the air, regardless of the amount of water under foot or in pipes. Squibbing and bowlder popping throw into the air the finest of dust particles, the ones that have come to be recognized as most dangerous to health. Thus, a false sense of security caused by waterline equipment must be guarded against. As in other matters of public health, prevention is easier, cheaper, and more efficient than remedial measures.

Altogether, the outlook for dust prevention, as judged by what has already been accomplished, is very hopeful. On a conservative estimate there has been at least a 50 per cent improvement, and it is the opinion of the State mine inspectors that at the present rate of progress dust in the sheet-ground mines will be reduced to a minimum within a couple of years from the time the present laws were passed. There is still too much squibbing and bowlder popping, but miners who have always been their own judges as to the necessity, can not be expected to drop these practices in a short time.

It would seem that underground conditions during the night shift are not as good as they are during the day shift. There are liable to be more new men on the night shift, and their supervision is not always so strict as during the day; men on night shift are more exposed to dust caused by blasting, and although it may not be practicable to change the time of blasting, more strenuous effort should be made to lay the dust thus caused where men have to work soon afterwards.

SUPPLY OF DRINKING WATER.

Recent mining laws did away with common drinking devices. Most of the larger mines are piped with drinking water of good quality and sanitary drinking devices are usually installed. A simple device is to have an upturned pipe, with a union on the end too big to be placed in the mouth, and with a drainage valve underneath; the water comes out with sufficient force to prevent saliva accumulating in the cup.

HUMAN WASTE.

So far little progress has been made toward remedying the evil of promiscuous ground pollution in the mines. A few mines have installed chemical closets, but it is too early yet to comment on their value. In the larger mines a simple toilet system should be installed, providing sanitary privies underground, with proper removal and disposal of their contents. An efficient privy is described in Bureau of Mines Technical Paper 105.[a] Empty powder boxes are used as receptacles, and are removed daily and burned. There does not seem to be much danger of hookworm disease in these mines, owing to the comparatively low temperature, which averages 60° F., and the nature of the soil, flint and limestone. However, the present habit of ground pollution is an objectionable nuisance and an offense to decency.

METALLIC POISONING.

The possibility of miners who work in zinc and lead being poisoned by those metals was kept in mind during this investigation. As particles of both these metals are in suspension in the air of the mines in the district, it might be supposed that they would have some harmful effect. However, there was nothing at all observed to lead in any way to the conclusion that metallic posioning was responsible for diseases of miners in the Joplin district. Outside of a couple of men who had worked in smelters, no cases with a history of lead poisoning were seen. Inquiry of operators and miners who had been in the district for many years failed to establish any instance of metallic poisoning. The lead occurs as galena or lead sulphide; being insoluble, such particles as may find their way into the body are either eliminated unchanged, or, if they lodge in the lung, remain chemically inert.

EFFECT OF USE OF CARBIDE LAMPS.

As carbide lamps have an offensive smell, and as they came into common use about the time that the prevalence of miners' consumption attracted general attention, there has been a somewhat widespread feeling against these lamps, as being responsible for this disease. This opinion, though erroneous, is not of importance, except that it tends to create a feeling that dust prevention is not necessary, and on the part of a ground boss, for instance, might lead to carelessness in this respect.

Carbide gas is poisonous, but there is no more reason for not using a carbide lamp than for not using illuminating gas, which is also poisonous, for cooking or for lighting a house. When carbide lamps fail to work properly underground and require to be opened the gas may escape, but diffusion rapidly takes place, and in the large connected rooms of the Joplin mines the gas could not possibly be in sufficient concentration to cause disease or death. A carbide lamp does not use proportionately as much oxygen as a candle, but a slight difference in this respect is of little importance. Moreover, carbide lamps are in use all over the world, but have never been said to cause pulmonary disease elsewhere than in mines.

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[a] Lanza, A. J., and Higgins, Edwin, Pulmonary diseases among miners in the Joplin district, Missouri, and its relation to rock dust in the mines; a preliminary report: Tech. Paper 105, Bureau of Mines, 1916, p. 34.

COMPRESSOR AIR.

Compressor air is also blamed locally as being a cause of disease and partly responsible for consumption. There are on record accidents that have occurred in various places, owing to the improper use of oil in compressors. The oil became volatilized, and the volatile products were sent through the air lines, causing illness and death to the men at the other end. No such accident has ever occurred in the Joplin district, as far as could be ascertained. During the writer's stay there the men at one mine became ill on one occasion, soon after going to work on the day shift, with suspicion being cast on the compressor air. Investigation placed the blame on powder gas released by shoveling and stirring the dirt piles blasted down on the previous night shift; samples of compressor air were analyzed and the air found to be normal. Various samples at other places showed the oxygen content to be practically always normal.

OVERWORK.

The writer has discussed the subject of overwork among the miners in the Joplin district in Bureau of Mines Technical Paper 105[a] previously mentioned. As ascertained by Mr. Higgins, the average ore production for all sheet-ground mines is 22 tons per 8-hour shift per shoveler. There is seemingly no prospect of any change in the piecework system, as it suits all concerned. There is one undesirable factor that can be eliminated, probably without a great deal of trouble, and that is the 20-minute dinner period. The State mining law, providing an hour for the noonday meal, should be enforced. Overwork and consequent exhaustion are the great contributing causes of tuberculosis among working people, and in this case there is no good reason why the miners should not have an adequate rest in the middle of their shift.

CHANGE HOUSES.

A notable improvement in regard to change houses has taken place during the past year (1915). Formerly the change house, or, as it was more aptly termed, the "dog house" (Pl. XI, *A*), consisted of a frame shack with a dirt floor, containing a stove, a couple of benches, and occasionally a few lockers. There were no facilities for washing, and garbage, refuse, and dirty and worn-out clothes littered up both the inside and the outside. Miners rarely changed clothes coming off shift, but went home, regardless of the weather, in their dirty and wet working clothes.

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[a] Lanza, A. J., and Higgins, Edwin, work quoted pp. 38-40.

Now the mines have erected, or are erecting, change houses with concrete floors and concrete or galvanized-iron walls, with plenty of windows and adequate ventilation. (See Pl. XI, *B*.) These change houses have shower baths with hot and cold water, wash basins, and individual lockers for clothes, are ventilated, and are heated by steam pipes. Also most of them have a separate room, with benches and tables, for eating (the men usually come out of the mine to eat).

The new change houses have been received with enthusiasm, and most of the men use them in the manner intended. A recent visit to a change house showed nearly every locker containing clean street clothes as evidence that the men changed when coming off shift.

HOUSING CONDITIONS.

Data on housing conditions were collected in and around Webb City by a series of house-to-house visits. Generally speaking, among miners in Webb City home conditions were fair or good, whereas among those living in the outskirts of town, or on mining land between towns (Pl. XII, *A*), they were bad. Taken all in all, when the wages of the miners in southwestern Missouri are considered, home conditions are far below par as far as sanitation and comfort are concerned. The situation in this respect is remarkable, because it is so needlessly bad. The miners make $3.50 to $5 a day, and even more at times, and they do not migrate to the extent observed in other mining communities. The chief obstacles in the way of improvement are a failure to appreciate better living conditions and, possibly to a lesser extent, the fact that many families live on mining land upon which nothing but temporary shacks can be built. Typical one-room, two-room, and three-room shacks are shown in Plates XIII, *A* and *B*, and XIV, *A*, and miners' homes of the best class are shown in Plate XIV, *B*.

The following tables were compiled from visits to 726 homes and 35 boarding houses. In the total of 761 premises there were 2,348 adults and 1,464 children, as far as could be learned. However, the total number of adults was considerably greater, because the number of boarders, both in boarding houses and in homes, could not be accurately obtained. The number of rooms to the house in 651 premises was as follows:

Number of rooms.	Number of premises.	Number of rooms.	Number of premises.
1	6	7	2
2	107	8	15
3	183	10	4
4	208	12	8
5	67	18	1
6	51	20	1

The houses with seven rooms and over were boarding houses, being two-story frame structures, with the exception of one concrete and one brick building. Practically all the homes were one-story frame buildings, and most of the one, two, and three room homes were "shacks."

The cleanliness in 694 premises was as follows:

Good	217
Fair	318
Bad	159

In the premises rated as "bad" are included most of the two and three room shacks situated outside of town and generally in the most squalid surroundings.

Data regarding the method of heating in 658 premises follow:

Natural-gas stoves	332
Coal stoves	276
Gas and coal	37
Other means	12

Houses in town or on the outskirts are supplied with natural gas, and those situated more remotely have to use coal. Usually the same agent is used for cooking as for heating.

The water supply of a great number of homes is rather unique. Water of good quality is obtained from deep wells and is *peddled* around the district in water wagons (Pl. XII, *B*) and sold by the barrel wherever there are no water pipes. Wells are scarce, and in the majority of homes the water barrel suffices for cooking and personal needs. Data regarding water supply in 658 premises follow:

Water obtained from barrel supply	443
City water piped to house	184
Wells	31

The outhouses are wretched, a feature in which southwestern Missouri resembles a great part of the rural communities of the United States. In 680 premises there were 644 insanitary privies, which consisted of the simplest kind of a box structure over a shallow pit dug in the ground. There were sewer connections in 36 premises. In view of the prevalence of filthy privies all over that part of the country, the scarcity of wells is fortunate, and undoubtedly the fact that water is peddled from a pure source is the greatest factor in preventing widespread typhoid fever and other intestinal disorders.

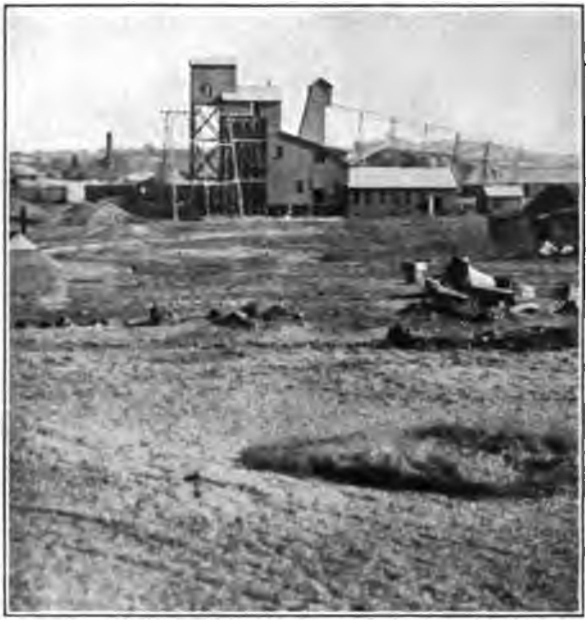
The method of garbage disposal in 610 premises was as follows:

Burned	352
Thrown on the ground around the house	216
Placed in cans and removed	19
Fed to the chickens or other animals	23

In none of these homes was there a bathtub or bathing facilities other than could be obtained from a pan of water on the kitchen

A. TYPE OF CHANGE HOUSE FORMERLY COMMON AND TERMED "DOG HOUSE." DIRT FLOORS, NO LOCKERS, NO BATHING OR WASHING FACILITIES, DIRTY INSIDE AND OUT.

B. CHANGE HOUSE OF NEW TYPE IN COURSE OF CONSTRUCTION. NOTE CONCRETE FOUNDATION AND PROVISION FOR ADEQUATE VENTILATION. CONTAINS VENTILATED LOCKERS, CONCRETE FLOOR, AND SHOWER BATHS WITH HOT AND COLD WATER.



A. SCENE NEAR WEBB CITY, MO. NOTE TENT ON LEFT AND SHACK ON RIGHT, TYPES OF THE FLIMSY STRUCTURE IN WHICH SOME MINERS LIVE, ON MINING LAND.

B. THE WATER WAGON ON ITS ROUNDS.

floor. In 281 premises there were 92 cases of tuberculosis and 120 cases of miners' consumption, representing office cases. In the other places visited information of this kind could not be accurately obtained.

It is impossible to give any figures as to ventilation. Many houses were kept tightly closed all the time, but even then the wind is likely to blow through them. Men suffering from miners' consumption soon find that they can not sleep without an abundance of fresh air, and in the homes of these sufferers doors and windows are usually left open when the weather will permit.

There did not seem to be much evidence of pulmonary tuberculosis among women and children, as a result of the crowded conditions in miners' homes. It is well known, however, that tuberculosis is largely a disease of children, and that in childhood its germs are lodged in the body to become awakened to activity in young adult life, when the stress of living is greatest. The conservation of the children in the Joplin district by taking care of adult consumptives so that they will not become a menace to their families is on a par with rock dust prevention—a matter of vital import. There is a serious need, both in and out of the public schools, of education regarding hygiene.

CARE OF THE SICK.

Recently Jasper County voted bonds to the amount of $100,000 for the erection of a county sanatorium for the care of consumptives. Such an institution is urgently needed, for at the present time there is no local hospital that will take these cases. The State sanatorium for tuberculosis at Mt. Vernon is an admirable institution, but limited by law to incipient or early cases, and consequently few miners are admitted there, as they seldom stop work while still able to get around. Not only for their own comfort, but for the protection of their families and of the community at large, is it necessary that there should be an institution where early cases may learn proper methods of living to combat their disease, and where advanced cases may have a decent and clean place to spend their final days.

There should be, preferably in Webb City, the center of the sheet-ground district, a dispensary where miners could go for physical examination and advice as to lung diseases. There should also be a children's dispensary, with visiting nurses attached, to observe the children in consumptive families, and by remedying the easily corrected defects so common in children, especially nose and throat defects, and by subjecting these children to physical examinations, to minimize the harmful effects of exposure to pulmonary disease. The writer desires to emphasize again that while miners' consumption is not infectious nor contagious, yet one can never know when a tuberculous infection may occur, and consequently in all public-health work miners' consumption should be treated with the same caution as is pulmonary tuberculosis.

It is to be hoped that local hospital and sanitary organizations will be able, in the near future, to organize a dispensary for both adults and children, with a visiting-nurse service. At the present time the Jasper County antituberculosis society and the Metropolitan Life Insurance Co. both maintain efficient visiting nurses, but it is probable that these different lines of endeavor, together with the cooperation of the miners, could be combined in connection with a dispensary service to the benefit of all concerned.[a]

The desired treatment of miners' consumption is purely hygienic—rest, fresh air, and good food. No medicine can restore those parts of the lung destroyed or damaged by rock dust. As with the great bulk of our tuberculosis sufferers, it is the financial inability to take sufficient rest and to change the occupation that is the most formidable obstacle to recovery.

Early cases of miners' consumption recover, as a rule, on removal from the cause, the inhalation of stone dust. Later cases may recover, but once the lung is seriously injured, not only is the breathing capacity permanently impaired, but a tuberculous infection is almost inevitable, and when that takes place in the already damaged lung the prospects for recovery are slight.

The writer has seen three or four sufferers from advanced miners' consumption who are seemingly recovering, after resting from all work for a number of months. They all look well and feel well, and two of them now do light work, although they can not do any hard work because of shortness of breath; the significant point is that they are improving. These cases are so exceptional that they stand out in a crowd of over 700. Still the fact remains that with a prolonged rest sufferers from miners' consumption may recover.

The question may arise as to the advisability of sending patients in the early stages of miners' consumption to a tuberculosis sanitarium for rest. As far as the patient is concerned, there is no sound objection to such a proceeding. Tuberculous infection is often a matter of doubt in these cases, but the patients will be in less danger in a properly regulated institution, where care is taken to prevent the spread of infection, than they would be at large.

SICKNESS INSURANCE.

Although sickness insurance has not been established on a firm basis in this country as yet, the experience of foreign countries, where this system has been eminently successful, indicates that

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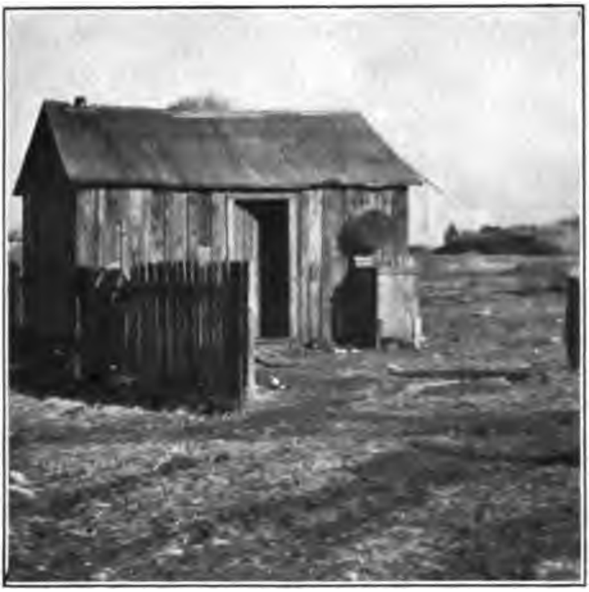
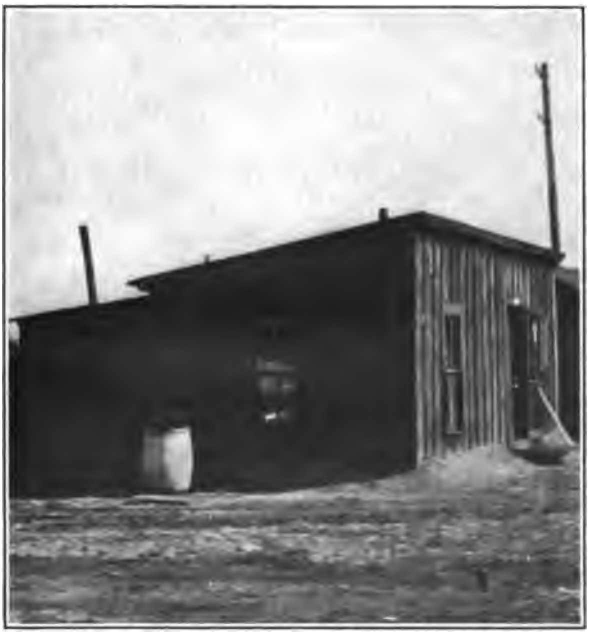
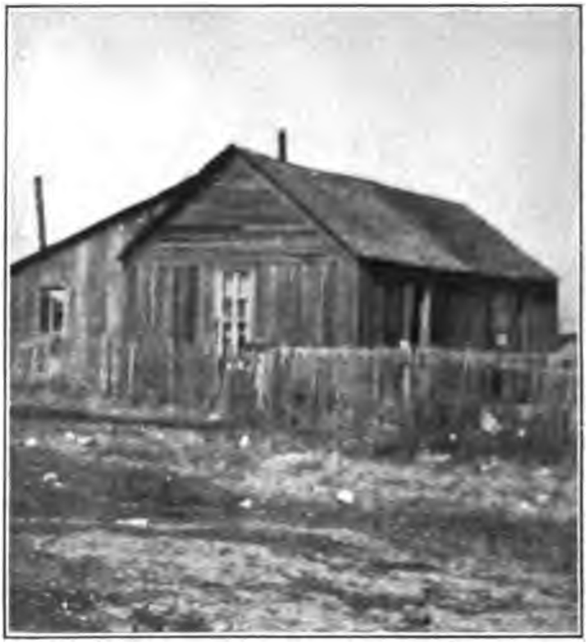
[a] Since this report was drafted, such a dispensary for adults and children has been opened and is being maintained in Webb City through the cooperation of the Jasper County antituberculosis society of the southwest Missouri mine safety and sanitation association.



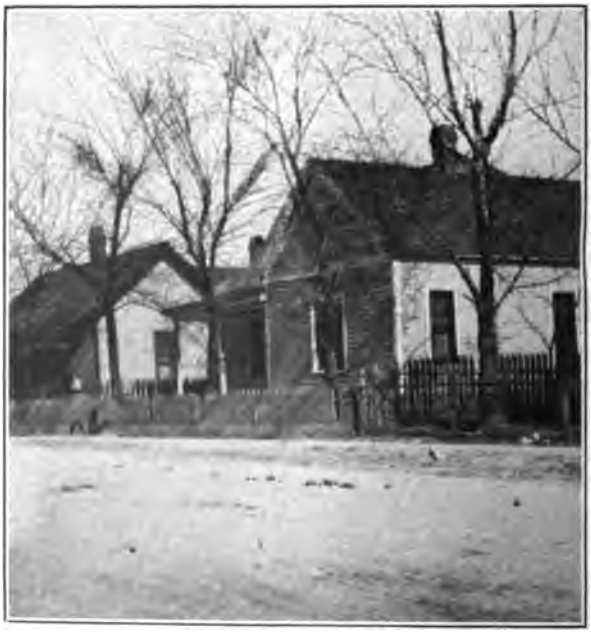
A. ONE-ROOM SHACK, OCCUPIED BY TWO PEOPLE. MINER HAS MINERS' CONSUMPTION.



B. TWO-ROOM SHACK, OCCUPIED BY TWO ADULTS AND TWO CHILDREN. ONE OF ADULTS HAS MINERS' CONSUMPTION. NOTE WATER BARREL.



A. THREE-ROOM BOARDING HOUSE; TWO ADULTS AND THREE CHILDREN IN FAMILY AND SIX BOARDERS. ONE INMATE HAS MINERS' CONSUMPTION AND ONE CHILD IS TUBERCULOUS.



B. MINERS' HOMES OF THE BEST CLASS.

sickness insurance must finally be instituted in southwestern Missouri if adequate relief from the burden of lung diseases is ever to be obtained.

It is not within the scope of this report to outline any scheme of sickness insurance with its somewhat complex details. It is desired to call the matter to the attention of the State department of health and charities, county, municipal, and mine-owners' organizations, and others interested, in the hope that by far-sighted cooperation the establishment of such a system may be undertaken. In the event that State assistance can not be procured, it may be possible to arrange among employers and employees a beneficial organization of the type that has proved so successful in some of the big mining and manufacturing districts in this country.

Workmen's compensation laws are already definitely established, and will probably be in force in Missouri before very long. The loss of time and of life from industrial sickness is far greater than from industrial accidents, and is quite as amenable to prevention and remedial legislation.[a]

CONCLUSIONS.

1. There is much pulmonary disease due to rock dust among the miners of the Joplin district, affecting probably as high as 30 per cent of sheet-ground miners.
2. The amelioration of dust-producing conditions is progressing at a reasonable rate.
3. Living conditions are generally poor, and often needlessly bad.
4. Although miners' consumption and tuberculosis of the lungs are entirely different diseases, as regards the public health no distinction should be made as to precautions against spread of infection.
5. The prevention of the spread of tuberculous infection, especially to children in the homes of consumptives, is of paramount importance.
6. The combination of hard work with a short dinner period is injurious and unnecessary.

RECOMMENDATIONS.

Instructions should be given from time to time, especially to new men, regarding the harmfulness of rock dust and regarding methods for its prevention. Educational leaflets should be distributed by the State bureau of mines, or local sanitary societies.[b]

There should be established a dispensary, or similar organization, to afford the miners the opportunity for physical examinations.

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[a] Warren, B. S., Sickness insurance, its relation to public health and common welfare: Public Health Report, No. 250, U. S. Public Health Service, January, 1915.

[b] The State mine inspectors do as much educational work as their duties permit.

There should also be medical inspection of schools and a visiting-nurse service, especially for the children of consumptive parents.

Continued care and vigilance should be maintained to abate squibbing and bowlder popping, and especially should attention be directed to lessening these abuses during the night shift.

An effort should be made, through cooperation of all concerned, to prescribe a maximum daily tonnage for shovelers.

It has been the experience of other communities and other countries that workmen's compensation laws and sickness insurance offer practical and equitable relief from the effects of various industrial conditions that are, in themselves, not entirely remediable. It is the opinion of the writer that such a system will eventually prevail in the Joplin district, and must prevail before present abuses will be reduced to a satisfactory minimum.

HISTORICAL REVIEW OF SILICOSIS.

By George S. Rice.

Silicosis, sometimes known as miners' consumption, is a specific occupational disease. It was seemingly known to the ancients, as is clearly shown in a paper by De Fenton,[a] read before the Chemical, Metallurgical and Mining Societies of South Africa, in May, 1916.

Hippocrates,[b] in his "Epidemics," speaks of the metal digger who breathes with difficulty, and is of a pale, wan complexion.

Pliny the Elder,[c] in his "Natural history," speaks of the use of respirators to avoid dust inhalation, as follows:

> Those employed in the works preparing vermilion (?) cover their face with loose bladder skins that they may not inhale the pernicious powder, and yet may see through the skin.

Loehniss,[d] in a book on mines and mine workers, published in 1690, says:

> The dust and stones fall upon the lungs, so that the men have lung disease, breathe with difficulty, and at last take consumption.

Diembroek[e] found in men's lungs such heaps of sand that in running the knife through the pulmonary vesicles he thought he was cutting some sandy body. This, by the way, was the experience of De Fenton, who, in the early days of the Rand, chipped many a razor blade while endeavoring to cut sections from dust-congested lungs.

Agricola (1494 to 1555), in his most famous work, "De re metallica," published in 1556, the year following his death, gives a considerable description of miners' diseases of the respiratory organs and their causes, a splendid translation of which, by Hoover and Hoover,[f] appeared in 1912. These diseases, as described by Agricola, De Fenton classified under two heads, as follows: (1) Those diseases which arise from stagnant air, or want of ventilation; and (2) those which are caused by the corrosive action of dust on the lungs of the miner. He finds four references under the first class, which he translates as follows:

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[a] De Fenton, J., On some diseases of the respiratory organs incidental to miners, as portrayed by Dr. Agricola, in A. D. 1550: Jour. Chem., Met., and Min. Soc. South Africa, May, 1916, pp. 223-227.

[b] Littré, E., Oeuvres complètes d'Hippocrate, avec le texte grec en regard, vol. 5, 1846, p. 147.

[c] Plinius Secundus, C., Historiae naturalis, lib. 33, sec. 11.

[d] Loehniss, G. E., Bericht von Bergwerchen, 1690, p. 56.

[e] See Ramazzini, B., A treatise of the diseases of tradesmen, translated by Dr. James, London, 1705, p. 163.

[f] Hoover, H. C., and Hoover, L. H., Georgius Agricola, De re metallica, London, 1912, 640 pp.

> Because the miners are liable to be killed, (*a*) sometimes by the pestilential air which they take in by breathing; (*b*) sometimes stagnant air, remaining either in a shaft or in a tunnel, produces a difficulty in breathing. The remedy for this evil are the ventilating machines.
>
> (*c*) If a shaft be very deep, and no tunnel or drive from another shaft reaches to it; or, if a tunnel be of great length, and no shaft connects with it, then the air does not replenish itself. In such a case it weighs heavily upon the miners, and they breathe with difficulty. Sometimes, indeed, they are suffocated, and burning lights are extinguished. And so one needs those machines which both Greek and Latin call "breathing," though they give forth no sound: for they enable the miners both to breathe easily and to complete the work in hand.
>
> (*d*) Air, indeed, remains stagnant both in tunnels and in shafts. This will occur in a deep shaft if it be by itself, i. e., if no tunnel connects with it and it is unconnected with any other shaft by a drive. * * * For this reason the vapors become heavy and like mist, and smell of moldiness like a vault, or some underground cellar which has been entirely closed for many years. Wherefore the miners are unable to continue work in these places for long (even though the digging may abound in gold and silver), or if they do they are unable to breathe freely, and suffer from headaches. This happens the more often if they work in such places many together and employ many lamps, which then supply them with but a feeble light. For the vapors which both men and lamps give out produce in their turn vapors which make the previous atmosphere fouler still.

Regarding the second class of diseases De Fenton extracts as follows:

> On the other hand, some diggings are so dry that they are entirely deficient in water, and this dryness produces even greater harm for the workmen; for the dust which is stirred up and put in motion by digging, penetrating right into the windpipe and lungs, engenders a difficulty in breathing and the illness which the Greeks call asthma. And if dust happens to have corrosive powers it eats away the lungs and engenders consumption in the body. Hence, in the mines of the Carpathian Mountains one finds women who have been married to seven husbands, all of whom this terrible disease has carried away by a premature death.

The Hoover translation of this reference is as follows:

> The critics say, further, that mining is a perilous occupation to pursue, because the miners are sometimes killed by the pestilential air which they breathe; sometimes their lungs rot away; sometimes the men perish by being crushed in masses of rock; sometimes, falling from ladders into the shafts, they break their arms, legs, or necks; and it is added that there is no compensation which should be thought great enough to equalize the extreme dangers to safety and life.

De Fenton criticizes the Hoover translation as being too general, whereas his own translation is more literal, and he states, "Agricola's phrasing is most pregnant, and pithily and accurately conveys to our mind the nature of the disease intended to be expressed. Remembering that the writer was a medical man with some 29 years' experience as a mine doctor, it behooves us to pay the greatest attention to the text."

In trades other than mining the danger of breathing certain kinds of mineral dusts has been indicated. Oliver,[a] in "Diseases of occupation," states:

> As far back as the days of Ramazzini (1670) the lung troubles of the potter had been recognized and described. With the terms "potters' rot" and "potters' asthma" the public are quite familiar. It is this liability to lung disease on the one hand and to lead poisoning on the other that has placed pottery manufacture high up the list of dangerous trades.
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> In firing china the articles are separated by means of ground flint, which is used over and over again. The particles of flint are sharp and angular and when breathed cause irritation and coughing.

Stonemasons, it has long been known, are subject to lung diseases. The French slate workers are exposed to breathing large amounts of schistose dust. Oliver [b] says:

> Dr. Hamaide, of Fumay, who knows the slate miners well, says that they begin to suffer from bronchitis between the ages of 35 and 40 years, and that once this becomes fixed and is followed by structural alterations of the lungs and bronchi, the malady proves fatal in 5 to 10 years. The mean duration of the life of the Fumay slate miner is 48 years. He has seen men who are the subjects of slate miners' phthisis expectorate fragments of hard stony material like schist, accompanied by profuse bleeding from the lungs.

It is thus seen that it is not alone underground workers who are subject to the malady, and that the disease is caused by dust rather than impure air; although the latter may in some cases be a factor in intensifying the trouble. Statistics in various countries show that coal miners and iron-ore miners are not subject to pulmonary troubles more than the average of men in the general occupations. In fact, the mortality among coal miners from pulmonary tuberculosis and pneumonia is less, so that it is thought that there is some protective influence exercised by coal dust. The fresh coal dust is sterile, and the particles do not irritate the lungs, in spite of the fact that the lung tissue of workers may be blackened by coal dust. Pennsylvania anthracite dust, however, although not causing miners' consumption, does appear to produce asthma, possibly owing to the hardness and sharp edges of the particles.

As regards the relation of dust to lung diseases Oliver [c] says:

> It was left to Peacock to establish the relationship between dust and lung diseases by demonstrating both chemically and microscopically the identity of the dust found in the lungs after death with that of the atmosphere in which the patient had worked. In the various pneumokonioses, or lung diseases caused by the inhalation of dust, the lesions in the lungs are, practically speaking, identical, quite irrespective of the nature of the dust. The lung substance, instead of remaining soft and spongy, becomes converted into a hard, unyielding substance, composed for the greater part of fibrous tissue. Thus structurally altered, the lungs are unable to discharge their function as organs of aeration.

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[a] Oliver, Thomas, Diseases of occupation, London, 1908, p. 307.

[b] Oliver, Thomas, work quoted, p. 304.

[c] Oliver, Thomas, work quoted, p. 280.

That it was the character of the dust which led to excessive pulmonary diseases among metal miners was early recognized in Great Britain. The statistics of the deaths from lung diseases and other causes in 1849–53 show that although the coal miners of Staffordshire, South Wales, and Durham had a lower annual death rate than the average of all the males of England and Wales, the average death rate of the Cornish miners mining tin and copper, after the miners had reached the age of 35, was much higher than that of the average occupied male. Between the ages of 35 and 45, the number of deaths per 1,000 was 8.5 for miners and 5.3 for all occupied males; between the ages of 45 and 55 the respective rates were 24.3 and 6.8; and between 55 and 65 were 44.5 and 9.6.

In 1901 Drs. Robertshaw and Leggi investigated the cause of excèssive death rate from dust inhalation among ganister miners. A royal commission was appointed in 1902 consisting of Dr. J. S. Haldane, J. S. Martin, and R. A. Thomas, to make an inquiry into the health of Cornish miners. Its report[a] showed the prevalence of lung disease among these miners as indicated above.

From an official standpoint the matter then rested in Great *Britain* until 1914 when a report was made by the Royal Commission on Metalliferous Mines. This report deals with miners' phthisis.

This commission quotes the results of experiments by Prof. Beattie at Sheffield University on the relative importance of certain mineral dust as a possible cause of fibrosis of the lungs. He found that the dust of some minerals, such as coal, clay, and cement, were not shown by experiment to be injurious, but other dusts, such as silica, quartz, flint, sandstone, carborundum, and emery were, and that fine crystalline silica was especially injurious. The commission agreed with these findings, but in the course of its report stated that the cause of miners' phthisis was the inhalation of dust of crystalline silica upon which is superimposed tubercular infection, and that fibrosis is not necessarily dangerous to life.

SILICOSIS IN SOUTH AFRICAN GOLD MINES.

The enormous development of gold mining on the Rand was accompanied by great mortality among the natives as well as the Europeans who worked in the mines. The vein rock is a conglomerate of the hardest kind, difficult to drill and still more difficult to blast. Unless extraordinary precautions are taken, enormous amounts of dry dust are produced, and if the air is kept agitated the atmosphere is charged with fine siliceous dust. The matter became so serious that it was brought to public attention shortly after the close of the Boer war.

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[a] Haldane, J. S., Martin, J. S., and Thomas, R. A., Report to the secretary of state for the home department on the health of Cornish miners, 1904.

As Oliver[a] states, "a few years previously young miners in the bloom of health had left their northern homes for South African gold fields, and after working there for four to six years returned to Northumberland and elsewhere broken in health."

In consequence, in 1902 the British Government appointed a commission to inquire into and report on the subject of phthisis in the mines of the Transvaal and to make recommendations as to preventive and curative measures. The report was published in 1903.

In the Rand mines, the great depth and consequent high temperatures, with chilling of the body when men were raised to the surface, according to Oliver, were responsible for bronchial catarrh and bronchitis, which intensified the condition produced by inhalation of dust. It was noted on the Rand that miners' phthisis started without previous outward symptoms, but once the disease gained a hold conditions might cause marked physical symptoms, one of the first being shortness of breath. The victim does not feel ill until the disease is far advanced, unlike tuberculosis and phthisis. Oliver[b] further comments:

> Loss of flesh and strength is so gradual that the miner continues to follow his calling, for the cough is slight and there is little or no expectoration. By degrees a sense of weakness impresses itself upon the patient, but it is the shortness of breath that obliges him to give up work. When the disease is fully established, it is observed, in addition to the dullness in the chest on percussion, that the breath sounds are coarse and tubular, but in the absence of accompanying bronchial catarrh moist râles are not usually heard. There is often pain in the chest, owing to the presence of a limited pleurisy over the patch of fibrotic lung, which explains the adhesions that are found after death binding the lungs to the chest wall internally. There is, as a rule, neither the rise of temperature nor the evening sweating which are present in tuberculosis phthisis, and "blood spitting" is an extremely rare event. So markedly absent is expectoration in uncomplicated cases that it is difficult to obtain sputum for bacteriological examination. In the early part of the illness, and sometimes throughout the whole of it, as in some of my own patients, the expectoration is free from tubercle bacilli. In other instances, toward the end of life tubercle bacilli are found in the sputum. This circumstance, the physical signs and symptoms, as well as the course of the disease, point to Rand miners' phthisis as being in the first instance a purely local affection of the lungs, the result of irritation by dust, and without tubercle. When the disease has become tuberculous, it is in consequence of superadded infection.

The report of the first commission (1902–3) showed that the disease was common among rock-drill men, and the conclusion was that dust inhalation was the cause of the disease. Some of the commission's important findings are given in the following extracts:

> Miners' phthisis is not peculiar to machine miners, for although it is most typically and commonly seen in them, it frequently develops also in course of time amongst miners who have never used rock drills at all. Most cases of this sort, however, have been mining for long periods, e. g., over 30 years, and it may be taken that the ordinary miner has a considerable advantage in length of life over the rock-drill miner.

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[a] Oliver, Thomas, work quoted, p. 279.

[b] Same, p. 281.

The disease undoubtedly falls most heavily on rock-drill miners. Out of the living rock-drill miners whose cases we have investigated, the majority of those who have been working rock drills for over 5 years were found to be suffering more or less from chest symptoms; the average time that those who were admittedly so affected had been employed on rock-drill work was from 7 to 9 years (ranging from 6 to 15) out of a mining life of 18 years. They died at the average age of 35.

It is true that men may be found who have worked for 10 years or more with rock drills and whose health is not perceptibly impaired; on the other hand others have become affected under the average time. Probably in this respect a great deal depends on the class and type of mining engaged in, and its locality. Wet mines have a much better reputation than dry mines. The quartzite of the gold-bearing beds of the Witwatersrand is admittedly very hard, and the mines are practically all "dry." There is also no doubt that persons who inherit chest weakness are more liable to be attacked by disease of this nature; but while this fact may explain some individual variations, it does not explain the enormous incidence of chest disease on miners as a class. The same observation may be made with regard to personal habits of intemperance and mode of living.

All the statistical evidence goes to show that developing, i. e., "raising" and "driving," is more dangerous to health than working in stopes. The most striking case in point which the statistics furnish is that of two brothers, who worked rock drills for four years only, doing nothing but "raising" all the time, and who died thereafter of chest disease at the age of 28 and 29, respectively.

RESULTS OF CLINICAL EXAMINATION.

Clinical investigation and examination show that the disease is an extremely insidious one, of very gradual development, having in its earlier stage little or no obvious effect on the general health, so that by the time the working efficiency of the miner becomes seriously impaired the disease is, as a rule, already well advanced.

The usual symptoms in a typical case are in the earlier stages recurrent and obstinate bronchial "colds" which may attract little attention, for the patient being for a long time quite able to work is apt to regard his earlier symptoms as of little moment. Gradually as the disease progresses shortness of breath on exertion, cough, and spit, a more frequent liability to contract colds and a greater difficulty in throwing them off, and sometimes flitting pleuratic pains, are the symptoms which usually first attract serious notice.

Gradual loss of weight and strength and more urgent breathlessness, which may be of an asthmatic type, follow, as an increasing area of breathing space becomes disabled. Expectoration may or may not be a prominent symptom; hemoptysis is exceptional and when present is usually slight; night sweats are not as a rule noticeable, and pyrexia may be altogether absent throughout. Three of the characteristic signs of tubercular phthisis are thus frequently absent.

Finally toward the end the patient rapidly loses ground in all directions.

The most striking feature brought out by the physical examination in a typical case is the diminished expansion of the chest, which in many cases is very striking, and is accompanied by rigidity of the chest wall, so that breathing becomes mainly abdominal. Percussion and auscultation often give very indefinite results owing to the diffuse nature of the disease. The disease is bilateral, but one lung is frequently affected to a greater degree than the other.

It is not common to find the typical physical signs of tubercular phthisis present, and this observation, together with the confirmatory fact that out of a series of over 30 sputa from cases of disease of the lung of miners examined by a member of your committee only 2 or 3 were found to contain tubercle bacilli, leads us to conclude that, while in some cases a true tubercular phthisis may coexist or may be superadded, the conjunction is only seen in a minority of cases.[a] * * * The disease is primarily a local one, and is at first confined to the respiratory organs; secondary disorders of the heart, liver, stomach, and kidneys are late accompaniments. The heart is not as a rule dilated, but the pulse rate may be accelerated.

In the more chronic forms, seen perhaps more typically in the miners of many years' standing, who as a rule are not rock-drill workers, chronic bronchitic and asthmatic symptoms are perhaps more prominent, and in them dilatation of the heart and accelerated pulse rate are more common.

We may therefore perhaps usefully distinguish from a clinical standpoint three types of "miners' phthisis."

(1) The pure fibroid nontubercular type, the commonest and most characteristic form of the disease.

(2) The mixed fibroid and tubercular type, where the two processes coexist.

(3) The very chronic type seen in miners of many years' standing, where the fibroid change in the lungs, although more chronic is less extreme, and where secondary changes in the heart and kidneys are more prominent.

In speaking of the Transvaal investigation, Oliver[b] says:

> In mining it is not the coarse particles but the fine, impalpable dust which is the source of danger, and which, in consequence of the arduous nature of the work in an overheated atmosphere, is drawn by the miner into the recesses of his lungs owing to deeper breaths he is obliged to take.
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> The amount of dust raised varies from 0.083 to 0.185 grain per cubic foot of air, so that a miner breathing his average 21 cubic feet of air per hour would run the risk of inhaling about 2.38 grains of fine dust per hour; hence the large quantity of silica found on chemical analysis of the lungs of gold miners after death, amounting frequently to 24.4 per cent of the total weight of the organs. Of 1,377 rock-drill miners employed in the Witwatersrand mines prior to the South African campaign, 225, or 16.34 per cent, died during the 2½ years immediately preceding the war.

This commission of 1902–3 seemed inclined to the view that gaseous impurities in the mine air predispose miners to miners' phthisis; the impurities arise from fumes of explosives. Its report disclosed a most serious situation regarding the presence of phthisis.

In 1907 another Rand commission was appointed to report on the working of the mines, work, and machinery regulations. This commission also gave much attention to miners' phthisis. Its report, published in 1910, sets, for the first time in South Africa, a definite standard of metal mine ventilation, based on a maximum allowable percentage of impurity as determined by chemical analysis of samples.

In 1911 a commission was appointed by the Union of South Africa, known as "The Miners' Phthisis Commission," to inquire into: (*a*) The extent to which miners' phthisis and pulmonary tuberculosis are prevalent among persons employed in certain mines; (*b*) the degree to which and the stage at which any person employed in such mines becomes incapacited by miners' phthisis and pulmonary tuberculosis from performing work; (*c*) the number of men who may be expected to become incapacitated annually; (*d*) the probable life of a sufferer after having become so incapacitated; (*e*) the provisions for compensation or insurance against phthisis.

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[a] In more recent reports the South African commission has changed its views on this question.

[b] Oliver, Thomas, Diseases of occupation, London, 1907, p. 282.

The findings of this commission, published in 1912, covered an examination of 3,181 miners, of whom 1,009 were afflicted with phthisis.

Another commission, the Miners' Phthisis Prevention Committee, was appointed in 1912 by the Union of South Africa, "to inquire into by experimental or other investigations and to report from time to time upon the improvement of methods for the prevention of miners' phthisis in the Witwatersrand Gold Mine, and to advise upon the introduction of a systematic and uniform policy and the amendments to the mining regulations which may be necessary for combating the disease."

This commission issued a preliminary report in 1912, an interim report in 1913, a special report dealing with the percentage of injurious particles in mine dust and on the counting of the particles of dust in air, in 1915, and a general report in 1916.

One of the first matters taken up was sampling air for *dust contents*, and the commission continued this work three and one-half years, taking a total of 2,809 such samples in the mines and on the surface. Samples were weighed and when it was considered desirable the number of particles was estimated. Wet and dry bulb readings were taken in conjunction with the sampling, and 100 samples of mine air were taken to determine percentages of deleterious gases present.

The Medical Commission in a report issued in 1912,[a] gave the following definition of the disease:

> (1) Miners' phthisis is a chronic disease of the lungs, characterized by progressive fibroid changes in the lung tissue and pleura, and accompanied by chronic catarrhal processes in the air cells and respiratory passages. The disease is thus primarily a fibrosis of the lung, and the essential factor in producing this condition is the more or less continuous inhalation over long periods of fine rock dust. All true cases of miners' phthisis are thus primarily cases of silicosis; silicosis is the feature common to them all.
>
> (2) In the latter states tuberculosis becomes commonly or invariably superimposed upon this condition, and the type of the disease becomes that of a tuberculous infection in a fibroid lung.

The report suggests that repeated exposure to the fumes of explosives in small quantities is a subsidiary factor in developing the disease. The definition states the important fact that ordinarily miners' phthisis does not run its course and terminate in death as the result of a single pathological process, but begins with silicosis and ends in an infective process.

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[a] Report of a Commission on Miners' Phthisis and Pulmonary Tuberculosis, Pretoria, 1912, p. 10.

A great majority of the cases in the early stage of the disease are those of silicosis uncomplicated by tuberculosis, and if the sufferer leaves underground work he may remain free from tuberculosis. On the other hand, if underground work is continued tuberculosis will nearly always follow. The following description of the structure of the lungs is important for the layman to remember in studying the effects of the disease.[a]

In order to reach the lungs the air which we breathe has to pass through the "air passages." First, it is conducted through the mouth or nose, the back of the throat, and the larynx, to the windpipe. The windpipe finally divides just above the heart into two large tubes, the right and left bronchi, which conduct the air respectively into the right and left lung. Each main "bronchus" immediately gives off branches, and these again smaller and still smaller branches, by which means the air is conducted to every part of the lung. Corresponding to this tree-like branching of the air tubes is the similar branching system of the arteries, which convey throughout the lung the blood which comes to it to be purified in breathing, and of the veins, which carry back to the heart the purified blood. These three branching systems, bronchi, arteries, and veins, run together throughout the lung.

Each minute tubular end branch or twig of the bronchial tree (called a "bronchiole") ends by dividing into several very small passages, round each of which is disposed a cluster of "air vesicles" or "alveoli." Each "air vesicle" is a roughly hemispherical tiny sac, the inner side of which is open to the air-containing passage in which the minute bronchus ends. Its outer side is covered by a close network of capillary blood-vessels. The wall of the "air vesicle" is extremely thin, in order to permit the interchange of gases between the blood and the air which goes on in breathing. The gases must pass through the thin walls of the air vesicle to reach the blood, and vice versa. Similarly, in order to penetrate into the lung, fine dust or pigmented matter must first pass through the fine walls of these little sacs.

The group of clusters of air vesicles, into which any one of the terminal ramifications of the bronchial tree leads, is called a "lobule," and there are in either lung thousands of such lobules, separated off from each other by fine partitions, derived from the supporting fibrous framework which runs throughout the lung.

The surface of each lung is covered by a smooth membrane, called the "pleura" or "pleural membrane," and this membrane is continuous at the "root" of the lung with a similar membrane lining the inner surface of the chest wall. Between the pleural membrane which covers the lung and that which lines the chest wall, a thin layer of fluid is normally interposed, an arrangement which facilitates the free movement of the lungs in breathing. When pleurisy (or inflammation of the pleural membranes) occurs, the two layers may become adherent, and the movement of the lung is then correspondingly hampered.

The air passages throughout are lined by a membrane which is kept moist by the secretion of "mucus" from its surface. This moist membrane tends to arrest dust or infective particles entering with the inspired air.

The interior of the nose also is so constructed that the organ under ordinary circumstances acts as an efficient dust arrester.

Further, the lining membrane of the lower air passages, namely, the windpipe, bronchi, and bronchial tubes, down to the very smallest of these, is provided with a multitude of excessively minute whiplike processes, called "cilia." The "cilia" are always in motion and maintain an outward movement of the mucus or "phlegm" away from the smallest tubes, and towards the windpipe and larynx. When the mucus reaches the latter, its presence irritates the very sensitive surface of the larynx and induces a cough, by which the phlegm is expelled into the mouth.

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[a] General Report of the Miners' Phthisis Prevention Committee, Pretoria, 1916, pp. 10-12.

These details are important, because the air passages furnish, in the manner described, a protective mechanism against the entry of foreign particles, such as dust, into the air vesicles.

Very much of any dust which is inhaled is entangled on the moist surfaces of the nose and throat, and if it is not present in overwhelming amounts, the portion of it which reaches the "bronchial tubes" is gradually worked upwards again in the phlegm and expectorated.

But when dust is continually inhaled in large quantities, this protective mechanism, adequate as it is under ordinary conditions, fails to completely fulfill its function. Many, especially of the finer particles, pass the protective barriers and gain access to the air vesicles.

The presence of irritant mineral particles in the bronchial tubes sets up a process of inflammation or "catarrh," which, by damaging the lining membrane, impairs the protective action of the "cilia," reduces the efficiency of the whole protective mechanism, and accelerates its breakdown. Similar catarrhal changes take place in the air vesicles also, when dust gains entry to them.

It is most important, therefore, for those employed underground to keep the protective mechanism, with which nature has provided them, in a state of efficiency. Exercise in the open air after working hours is to be strongly encouraged. Whether such exercise is taken or not, the systematic practice of simple breathing movements, which anyone can readily learn, will also help greatly to maintain the normal power of chest expansion, stimulate the circulation in the lungs, and promote the healthy activity of the whole breathing mechanism. Dust will thus be got rid of which otherwise would have remained behind to do further damage.

A regular annual holiday away from the inimical influences of underground work is also a very important preventive measure.

When dust is inhaled day by day in large quantity the protective mechanism provided by the air passages proves, as has been said, inadequate. Many of the very fine particles of the inhaled dust are carried, not only to the bronichal tubes, but to the air vesicles or alveoli, in both of which situations they set up catarrhal changes. This element of "catarrh" is one factor in miners' phthisis, and may from time to time be rendered acute by recurring "colds" of infective origin. Exposure to even minute quantities of nitrous fumes will also, when it occurs, provoke and maintain catarrh, and thus further weaken the resisting power of the lung against attack by dust. The catarrhal change in the alveoli is characterized by multiplication and shedding of the cells which line the walls of these tiny sacs. These cells readily take up dust.

The next step is that, ultimately, in many parts of the lung, a proportion of the very fine silicious particles gains entrance through the walls of the air vesicles to the surrounding connective tissue of the framework of the lung, chiefly through the agency of the shed alveolar cells, which act as carriers. Once in this situation the dust particles are carried along the lymphatic channels, and some lodge at various points in the course of the latter. At these points their irritant properties produce a slow reactive inflammatory change, characterized by the production of new fibrous tissue. This new fibrous formation is at first found around the smallest ramifications of the air tubes and blood vessels of the lung, principally in a finely nodular form, but it also leads to thickening of the fibrous framework of the organ and frequently of the pleural membrane which invests the lung. The air vesicles in the neighborhood of the areas of fibrous formation, and sometimes more generally, are observed to be in a state of "catarrh."

The new fibrous tissue appears first in small discrete areas. These gradually become more numerous throughout the lung and at the same time increase in size and extent at each point affected, so that they ultimately tend to coalesce with similar neighboring areas.

In time this fibroid change leads to a considerable amount of obliteration of the fine air tubes and blood vessels (or portions of them) and of air vesicles throughout the lung. The result of this is to reduce the reserve of air capacity, available in breathing, which exists in the normal lung. The thickening which takes place in the fibrous framework, and very frequently in the pleural membrane, aggravates this condition and impairs the capacity of the organ to expand during breathing. The silicotic process may, in time, lead to the formation of more or less extensive areas of dense fibroid consolidation, which render the portions of the lung so affected practically airless. The cardinal symptom of silicosis, shortness of breath on exertion, thus becomes readily intelligible. First, there is an increasing effort in breathing deeply; later on there is actual incapacity to do so.

A lung in a condition of silicosis is, as has been said, liable to intercurrent ineffective invasions throughout its course. Thus, attacks of bronchial or pulmonary catarrh or localized pleurisy are not uncommon, and are a factor of aggravation in the disease. Pneumonia may also occur as a complication at any stage and may cause death. If recovery takes place, this disease is characteristically slow to clear up in a silicotic lung.

But the most serious invasion to which such a lung is liable is tuberculosis. When tuberculosis invades a silicotic lung, destructive and obliterative changes become much more marked. More or less extensive areas of consolidation, which is frequently of a very dense fibroid character, result. The extensive areas of dense fibroid consolidation which are characteristic of cases of advanced miners' phthisis appear, indeed, most commonly to arise in this way, and to have a mixed silicotic and infective origin. In addition to the changes in the lung, the ordinary constitutional effects of the infection become apparent. The disease alters its type, and the characteristic symptoms of a simple silicosis are aggravated by, and to a large extent merged in, those of an acute or chronic pulmonary tuberculosis.

The Medical Commission stated that, as a result of the examination of 3,136 underground workers, 26 per cent of the general body of miners at work were found to show definite signs of silicosis, while a further 5.5 per cent were classed as probable but not definite cases. The disease especially affected the rock-drill miners, and successively in the order of risk, trammers, hammermen, and timbermen, but no group of underground workers were found to be free from serious risk of attack. Even the supervisory staff was not exempt. The Medical Commission advocated a strict system of selection of recruits for underground work. A man best suited for underground work was "one of good, sound physique and with a good reserve of respiratory capacity." They considered that it was not wise to accept a man over 45 years of age, because of a tendency to increasing reduction of the expansile power in the chest walls and a proneness to bronchitis and emphysema.

All having sign of tuberculosis taint should be rigidly excluded.

The risk of conveyance of tuberculosis from infected to healthy persons underground, the commission considered, is by no means negligible. Out of 205 specimens of sputum collected underground

15.2 per cent contained the tubercle bacillus. Infection is more likely to arise from contamination of hands and subsequent ingestion. All water for sprays, as well as drinking, should be pure. There should be disinfection of ladder and traveling ways, also of change houses and living rooms.

Dr. McCrae found that dust from lungs of deceased miners consisted principally of extremely minute particles, none larger than 12 microns in diameter, and the majority less than 1 or 2 microns.[a]

In a miner's lungs containing 20 grams of silica the total number of particles, if they averaged 2 microns in diameter, would be six million million.

Character of dust found in mines on the Rand.

Constituents.	Original ore.	Dust from blasting.
Silica (SiO_2)	87.4	69.8
Alumina (Al_2O_3)	9.8	22.5
Iron (Fe)	1.75	4.5
Basic oxides (CaO, MgO, FeO_3)	Traces.	3.2
Sulphur (S)	1.1	(a)

a Sulphur drawn off in heating dust.

The mining operations which result in dust being formed and projected into air were divided by the commission into four classes: Blasting, drilling, transport, breaking and crushing rock.

The first two are the important causes underground. The effect of blasting was tested at one time by systematic sampling of the main return air near the exhaust fan. The velocity of the air was 1,500 feet per minute.

Results of sampling air for dust.

Time of sampling.	Dust in each cubic meter. Mg.
9 a. m. to 2.40 p. m.	3.5
2.40 p. m. to 3.15 p. m.	1.0
Blasting done at 3.15 p. m., smoke appeared at fan at 3.55 p. m.	
3.15 to 4.40 p. m.	9.0
4.40 to 5.45 p. m.	12.5
5.45 to 6.40 p. m.	9.0
6.40 to 7.20 p. m.	5.5
8.45 to 9.37 p. m.	4.5

In different methods of drilling the amounts of dust produced were as follows:

Amounts of dust produced.

	Mg.
Dry drilling with piston type drills	68 to 201
Dry drilling with hammer drills	34 to 100
Drilling with water sprays in "hand stopes"	4.5
Drilling in "machine" stopes	3.8

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[a] A micron equals 1/1000 of a millimeter, or about 1/25000 of an inch; the diameter of a red corpuscle is 8 microns.

The commission found by tests that respirators would not stop the fine dust, and as they did catch the coarse dust, allaying irritation of nose and throat, their effect was misleading.

The use of water is a great gain. In the Simmer Deep Mine, while drilling without sprays, in September, 1911, before blasting 80 to 280 milligrams of dust per cubic meter of upcast air were found; after blasting (fan stopped), 100 milligrams. With sprays in use, in September, 1912, the amount of dust before blasting was 0.2 to 3.3 milligrams per cubic meter; after blasting, 14 to 17.1 milligrams per cubic meter.

Belger fluid, chiefly calcium chloride with glue and water, was tried. It was found to have no advantage as a spray, but owing to the deliquescence of calcium chloride was thought to be advantageous for use in airways or passages in which the air was below saturation, to keep the surfaces moist and thus hold the dust.

As a result of the work of the several commissions mentioned above, drastic regulations were adopted in the South African mines, among which are the requirements in the mines regulations act (Transvaal) that—

> No person shall in the drilling of holes use or cause or permit to be used any percussion machine drill unless a water jet or spray or other means equally efficient is provided and used so as to prevent the formation of dust by drilling, and unless the floor and sizes of the working place to a distance of at least 25 feet from the face be kept constantly wet.
>
> No person shall in any part of the mine move any broken rock or ground or cause or allow the same to be moved if such rock or ground is in a dusty condition unless and until it and the floor, roof, and sides of the working place to a distance of at least 25 feet have been effectively wetted and kept wet so as to prevent the escape of dust into the air during removal.
>
> The ganger shall in all development faces (except in the case of a winze being worked single shift) immediately after lighting up put into action the water blast which he shall have previously tested. If as a result of such test the water blast be found to be not in order no blasting shall take place.
>
> The manager of a mine shall:
>
> (*a*) Provide or cause to be provided an adequate and constant supply of water which is clear and odorless at every working place which is not naturally wet. Such supply shall be sufficient for effectively wetting the broken ground and for preventing formation of dust caused by drilling operations. Such water shall be supplied in metal pipes not less than 1 inch in diameter at a pressure of not less than 30 pounds to the square inch at each working place when all sprays and jets supplied from the same pipe are working. Such pipes shall reach to within 50 feet from the face and from there a sufficient length of hose shall be provided and used to bring water up to the face.
>
> (*b*) Cause the surface in all working places, traveling ways, and shafts which are not naturally wet to be kept wet as far as practicable.
>
> (*c*) As far as practicable cause such old stopes as tend to short-circuit or add dust to the ventilating current to be closed off and cause the ventilating current to be circulated along the working face.

(*d*) Provide in such shafts which are not wet ring sprays at suitable points to prevent dust being carried with the air current.

(*e*) Provide a water blast as near as practicable to all development faces (except winzes being worked on single shift.)

Among other improvements that have been introduced on the Rand are change houses, so that miners hoisted from the hot, humid atmosphere of the mines may have opportunity for bathing and for putting on dry clothing before going out into the cool, open air.

Owing to the hot atmosphere in the Transvaal mines, water used in wetting the mines tends to dry up rapidly, so that a few hours later clouds of dust may be raised by the concussion of blasting. Accordingly, either the development places should be resprayed or, as has been more recently proposed, continuous sprays may be used.

An alternative plan that has been tried[a] is the use of a mixture "consisting of a 50 per cent solution of molasses and water. * * * This mixture is sprayed on the roof and sides of development places, where it forms a slightly sticky coating. All dust settling on this surface is retained in a damp condition, thus preventing the usual cloud of dust raised by the concussion of blasting, whereas if water alone had been used the dust would have dried up after an interval of only a few hours, and been left to be disseminated at the next blast."

SILICOSIS IN THE QUARTZ MINES OF AUSTRALIA.

Statistics have shown that miners' phthisis is prevalent in the quartz mines of Bendigo, Australia. In a report on the health of miners there, Summons[b] gives the following figures regarding the annual deaths of miners from miners' phthisis, estimated as per 10,000 living at all ages: 1875–1879, 48.5 out of a total of 179.8; 1905–6, 129.6 out of a total of 270 deaths.

These figures show that since machine drills have been in use the number of men affected has increased enormously. Seemingly, the conditions are not so acute in the Bendigo mines as in those on the Rand, as the average number of years a miner worked in the Bendigo mines is 22.

In 1910 investigation was made into the prevalence of pulmonary disease among miners in western Australia. The report of the commissioner, Dr. J. H. L. Cumpston, does not bear out the conclusions previously held of the rarity of miners' phthisis.

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[a] Gullachsen, B. C., The prevention of dust in underground workings: Jour. Chem. Met. and Min. Soc. of South Africa, January, 1914, p. 340.

[b] Summons, Walter, Report of an investigation at Bendigo into the prevalence, nature, causes, and prevention of miners' phthisis, 1907, p. 5.

PHTHISIS IN GERMANY.

Although Agricola, as cited in the beginning of this chapter, specifically refers to the presence of consumption in the Hartz mines, with which he was acquainted, the disease was probably due largely to poor ventilation, and as modern methods were adopted it is probable that with the better ventilation the health of the workers improved and they became less susceptible to the relatively small quantity of dust produced before the introduction of percussive machines. In recent years the Hartz mines have been practically exhausted, and extensive metal mining in Germany is confined largely to iron and zinc ores, in which the dust problem is not serious.

However, as to the effect of coal dust upon the lungs, two opposite opinions are held—one that the dust is not only harmless but even conducive to health, and the other that it is a danger to health. As pointed out in a German journal,[a] these discordant opinions are due to the fact that some are exposed to the inhalation of pure coal dust and others to the inhalation of shale dust with the coal dust, and frequently shale dust contains much free silica.

It has been found in Germany that industrial dusts have had a very serious influence. The Prussian statistics show that in 1909 persons suffering with tuberculosis had been admitted to the Prussian hospitals to the number of 82,576, which was 7.8 per cent of the total number of patients. In 1909 the total number of deaths in Prussia was 60,871, or 0.15 per cent of the population. The number of cases in Germany is between 800,000 and 1,300,000 in one year.

Sommerfeld[b] points out that of 1,000 persons 2.39 per cent died of pulmonary tuberculosis in occupations in which little dust is encountered (particularly agriculture); 5.64 per cent in work evolving organic dust; and 5.84 per cent in work producing metallic dust.

The disease was particularly manifest among workers of porcelain factories, where the rate was 14 per 1,000. Hirt[c] states that the average life of glass grinders is only 42½ years, and if they begin to work at 15 years of age they seldom live beyond the age of 30. The cause of such early death is the dangerous character of the dust produced in grinding, the dust being exceedingly fine and sharp. These data indicate that a fine sharp dust is an important factor in the causation of consumption.

PREVALENCE OF SILICOSIS AMONG MINERS OF THE UNITED STATES.

It has for many years been known to American engineers and operators that, in certain mining districts in the United States, as for example in the quartz mines on the Mother Lode of California, and in the sheet-ground zinc mines of Joplin, Mo., there were most serious conditions, causing an excessive number of deaths of miners from pulmonary diseases. It was known that workers in metal mines in various parts of the country had been affected, but no comprehensive survey had been made, although the subject was alluded to from time to time in mining journals.

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[a] S. B. B. Zeitung, 1911.

[b] Sommerfeld, Theodor, Handbuch der Gewerbekrankheiten, Berlin, 1898, p. 20.

[c] Hirt, Ludwig, Die Krankheiten der Arbeiter, Breslau, 1873, p. 245.

In the earlier mining and tunneling in the United States, as in other mining districts of the world, hand drilling with relatively small charges of explosive was used in breaking ground. With such conditions dust was a less serious problem. Formerly the number of miners engaged in quartz and other mines which had siliceous ores was small as compared with the number of miners engaged in mining coal, iron ore, and other ores with nonsiliceous gangue, and owing to the drifting of the miners from one point to another miners' consumption was not particularly noted. When, however, compressed-air percussive drills were introduced extensively into the quartz and siliceous rock mine, sminers' lung troubles became more common. This was intensified with the increasing depth of working with consequent rise in temperature of the mine air, making the men more susceptible to pulmonary troubles, which then came more conspicuously before the public.

In the past few years a new factor has become prominent in the mining problem in the quartz and schist mines—the development of new drills. There was a demand for a drill that would automatically throw a stream of water into the hole being drilled, thus preventing dust. As a result there was devised a water-injection drill which when properly used admirably serves the purpose of preventing dust from drilling operations. The disadvantage is that in drilling "uppers" the drill men are splashed with water, and where the air is cool they sometimes do not use the water, so that then, of course, the drilling operation makes dust. Nevertheless the introduction of this type of drill has been an important factor in lessening the production of dust in many mines.

Two other drills—the "stoper" and the "jack hammer"—recently come into use in the United States add to the difficulties. These drills are light and easily moved around, and their use has proved highly effective, the "stoper" for use in stopes, and the "jack hammer" for drilling block holes. Owing to the purposes for which the drills are used, internal water jets may be inconvenient for the jack-hammer type and not be attached.

The use of respirators has been proposed from the first. Self-contained oxygen breathing apparatus, although it would be effective, would be too cumbersome and expensive for regular use; and the miners will not use the ordinary respirator unless the conditions are unusually bad. In fact, as pointed out by the British commissioners, Haldane and Martin, the protection afforded is very imperfect; the respirators leak around the edges, particularly after the filtering material has become wet from the moisture content of the expired air. Moreover, respirators are hot and uncomfortable, and as at the end of each respiration the open space is filled with expired air containing about 6 per cent of carbonic acid, increased depth of respiration is necessary. Men who are wearing respirators can not freely communicate with one another. Consequently in many districts it is difficult to get the men to use the respirators. The recent South African commission found that the respirators it tested were ineffective.

Various methods of lessening rock-drill dust have been tried, such as surrounding the mouth of the drill hole with a bag through which the drill passes; although the arrangement is effective when properly placed, it is rather difficult to adjust and keep in place, and when offered to the miners they in general will not use it. Such devices were also considered ineffective by the South African commission, and the final remedy adopted was the general use of water.

The United States Bureau of Mines immediately after its establishment, made a rough preliminary inquiry into the question. Subsequently, in May, 1911, the Bureau of Mines and the Public Health Service were jointly requested by the secretary of the national association for the study and prevention of tuberculosis to investigate the subject in accordance with the following resolutions adopted by the board of directors of that association:

> Whereas the royal commission appointed by the governor general of Australia to inquire into the subject of miners' lung diseases has ascertained a truly alarming state of affairs, resulting from the extensive use of rock drills underground, not provided with spraying apparatus to diminish the production of health-injurious dust; and
>
> Whereas a large proportion of our mining population are exposed to conditions quite similar to those reported upon adversely in the Australian Commonwealth; and
>
> Whereas the actual extent of the occurrence of lung diseases among metal miners in the United States is at present unknown:
>
> *Resolved by the board of directors of the national association for the study and prevention of tuberculosis*, That we recommend to the President and the Congress of the United States that a thorough investigation into the whole subject of the sanitary conditions surrounding metal mining under ground, with special reference to diseases of the lungs, be made by the United States Bureau of Mines, the Public Health and Marine Hospital Service, and the appropriate State authorities.

Prior to this the Public Health Service and Bureau of Mines had cooperated in some investigations as to the prevalence of contagious and infectious diseases among miners; and it was then decided to make a specific investigation of the prevalence of miners' lung diseases in metal mines. Accordingly, Passed Asst. Surg. S. C. Hotchkiss began a survey of metal mines in various places, and found considerable silicosis in certain kinds of mines. However, the investigation had hardly been started when Dr. Hotchkiss's sudden and untimely death led to the loss of the data collected. Subsequently it was decided by the Director of the Bureau of Mines to make a detailed investigation of the prevalence of pulmonary diseases in the Joplin district of Missouri, for the special purpose of obtaining the life history of the miners, that district being selected because the great majority of the miners are American born, and have worked almost exclusively there. At the request of the Director of the Bureau of Mines the Surgeon General of the Public Health Service detailed Dr. A. J. Lanza, passed assistant surgeon, to cooperate with Edwin Higgins, mining engineer of the bureau, to make a survey of the district, especially of the sheet-ground mines, in which flint dust was believed to be the largest source of trouble in the large number of fatalities from lung diseases in the Joplin district. The soft-ground mines of the district are for the most part in limestone and clay formation and it was not expected that siliceous dust would be found in dangerous quantities, a supposition that was confirmed by the subsequent investigations with which this report deals.

Before the inquiry was finished, owing to the serious conditions found, it was deemed important to publish a preliminary report, and this appeared as Bureau of Mines Technical Paper 105.[a] The investigators, Messrs. Lanza and Higgins, were accorded splendid cooperation by the Missouri State mine inspectors, by the mine operators of the district, and by the miners, and reforms were immediately begun. Statistics gathered in the immediate future may not show great improvements, owing to the fact that many persons were already seriously affected. Ultimately, however, there is likely to be a great change for the better.

Simultaneously with carrying on the field work an investigation was conducted as to the character of the dust produced in the mines, which in turn led back to a study of the origin of chert deposits in which the zinc and lead of the sheet ground is found. The study of the samples of chert was conducted by F. B. Laney, geologist of the United States Geological Survey, whose report is presented herewith.

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[a] Lanza, A. J., and Higgins, Edwin, Pulmonary disease among miners in the Joplin district, Missouri, and its relation to rock dust in the mines; a preliminary report: Tech. Paper 105, Bureau of Mines, 1915, 48 pp.

THE CHERT OR FLINT OF THE JOPLIN DISTRICT.

By F. B. Laney.

INTRODUCTION.

In the hard-ground or sheet-ground mines of the Joplin district, the principal rock associated with the ores is locally known as flint. It is dense, extremely hard, gray or black, and consists almost wholly of silica. Variations in color are largely due to carbonaceous matter and small amounts of iron oxide. According to modern usage, this rock may properly be called chert, the term used by the United States Geological Survey, and it is so designated in this report.

On the basis of color and texture there are in the Joplin district two distinct types of chert. One is light to bluish gray, exceedingly dense and fine grained, and very brittle; the other is very dark or black, of much coarser texture, and slightly less brittle. In point of age and conditions of deposition or formation, there are believed to be three distinct types, as follows: An original light to bluish gray chert, probably contemporaneous in deposition with the limestone in which it occurs in beds of varying thickness; a younger, secondary chert similar in color and texture, but occurring as a replacement of the limestone and in fractures in the original chert; and, finally, the secondary black chert or jasperoid. In other words, it is believed that the light to bluish gray chert is of two periods of deposition or formation and is the result of two separate and distinct processes, and the black chert or jasperoid is of one extended period of deposition and the result of one type of process. The gray chert is older than the ores, and although its surface may be coated with the ores of zinc and lead and they may fill fractures in it, they are never found disseminated throughout the mass of the rock. On the other hand, the black chert or jasperoid occurs as fracture filling in the older chert, and makes up the matrix for large masses of breccia, the fragments of which consist of sharply angular pieces of the older chert. The ore minerals are also widely disseminated throughout the body of the black chert. It is therefore clear that the jasperoid is younger than the gray chert and probably contemporaneous with the deposition of the ores.

Both types of the rock break with sharp conchoidal fracture into numerous splinters and particles with exceedingly sharp points and edges that cut almost like razors. This mode of fracture is more pronounced in the dense, older chert than in the jasperoid. Different observers have noted that when either type of the chert, but more especially the older, is broken the fracture faces are so curved that the fragments can not be made to fit together perfectly, thus indicating that the rock is under an internal stress or strain. This fact, to a certain degree, may account for the brittleness of the rock and its tendency to break into minute sharp-edged fragments.

Chert, including jasperoid, forms by far the most important gangue material in the Joplin ore deposits. No mines are entirely free from it, and in some it is practically the only rock intimately associated with the ores. An idea of the enormous quantities of chert that have to be broken, hoisted, and crushed in mining and milling the ores may be gained by a glance at the thousands and thousands of tons of tailings and "chats," both of which consist almost wholly of chert.

All writers on the Joplin district have necessarily devoted much space to the chert, and hence the literature on the subject is considerable. All writers agree fairly well in classifying the material into three divisions or generations—an older, fossiliferous chert; a still older, nonfossiliferous, gray chert; and a younger, black variety. Buckley and Buehler [a] make the following statement on the subject:

> There are at least three generations of flint associated with the limestone and the ore bodies. The first is a white variety, which is thought to be original, since it does not contain fossils common to the limestone. The second is a white, gray, or blue variety containing fossils and thought to be a replacement of the limestone. The third is a black flint, which evidently resulted from either the silicification of arenaceous mud or the precipitation of silica from underground waters, perhaps both. This flint contains no fossils and serves as a matrix, in which are embedded fragments of the white flint, forming what is commonly known as a breccia. In places both the white and the black flints have been decomposed. When decomposed, the white flint is known to the miners as "cotton rock," on account of its white, soft character, while the decomposed black flint is called "cod rock."

For the purposes of this report it is not considered necessary to discuss separately the two classes of older chert, which are much alike so far as their relation to the ores is concerned. The jasperoid is so different from the other chert and is so much more intimately associated with the ores that it merits separate consideration.

GRAY CHERT.

GENERAL CHARACTERISTICS.

The older, gray or bluish-gray chert is closely associated with the ores, both as beds of varying thickness—a few inches to several feet—of irregular lateral extent and as ellipsoidal nodules usually less than 6 inches in diameter. Many of the beds or sheets of chert have a wavy or irregular surface, and in a few the lamination planes of the limestone extend into and through the nodules. The rock varies much in color, owing to impurities, iron staining, and different degrees of weathering or decomposition. The weathered material has a flat-white and chalky appearance, except for iron stains of varying shades of yellow, and is so soft that it can be cut readily with a knife. Much of the chert, especially that in heavy beds, is shattered or broken into small, sharply angular fragments, which in many localities are cemented by the younger, black chert or jasperoid into beautiful breccias. The rock when fresh is very hard and brittle and breaks readily with nearly perfect conchoidal fracture into fragments of varying shapes and sizes, all with sharp points and razorlike edges. In fact, perhaps the most characteristic feature of the fresh rock is its tendency to break into sharp-edged fragments.

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[a] Buckley, E. R., and Buehler, H. A., The geology of the Granby area: Missouri Bureau of Geology and Mines, vol. 4, 2d ser., 1906, pp. 30-31.

Much of the chert shows small vugs or openings; some of these are lined with minute, well-terminated quartz crystals. Some of these openings contain blende and other ore minerals in more or less well-developed crystals, but the sulphides are rarely or never disseminated through the mass of the rock. Much of the rock also contains fractures, many of which have been more or less perfectly healed either by the deposition of younger, gray chert, often in the form of minute, anhedral quartz grains, or by the deposition of jasperoid. The rock is invariably so dense and so fine grained that little or nothing of its texture or mineralogical composition can be determined by the unaided eye, or even by a hand lens. Much of it is highly fossiliferous, the fossils almost invariably being those that originally had calcareous shells, or supports, such as corals, brachiopods, crinoids, or bryozoa. In some of the chert the fossils are only partly silicified, and in places the chert grades, by decrease of silica and corresponding increase of calcite, into nearly pure limestone, thus showing its origin. This subject is treated in greater detail in the discussion of the origin of the chert.

CHARACTERISTICS AS SHOWN BY THE MICROSCOPE.

In thin section, under the microscope, the gray chert is seen to be composed largely of exceedingly fine-grained irregularly interlocking quartz, the grains of which vary from 0.05 mm. to a size so small that the microscope all but fails to resolve them; in fact, most of this chert is typically cryptocrystalline. Although the texture is always very fine and dense there is considerable variation, as is well shown in Plates XV, *A*, XV, *B*, and XVI, *A*, which are reproductions of photomicrographs of typical gray chert.

The microscope shows that in much of the chert a great deal of the silica is present as chalcedony. In one thin section studied, the proportion of chalcedony was at least 30 per cent; but the amount varies greatly, and in some specimens examined chalcedony, if present, could not be identified. This peculiarity of the Joplin cherts was noted and described long ago by Hovey.[a] The chalcedony is usually slightly yellowish, probably owing to varying but small amounts of hydrated ferric oxide. It is transparent or slightly translucent in ordinary light, but between crossed nicols it shows beautifully the typical black cross or aggregate polarization. Its relation to the cryptocrystalline quartz so characteristic of these rocks indicates pretty clearly that the quartz has largely resulted from the crystallization or devitrification of the chalcedony, and that at one period in its history the chert was largely, if not wholly, in the colloidal or chalcedonic condition.

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[a] Hovey, E. O., Lead and zinc deposits of Missouri: Missouri Geol. Survey, vol. 7, 1894, p. 728.

Another interesting feature of the gray chert is that in many samples the grains of quartz show undulatory extinction, indicating that the rock is under stress or strain. There is no indication that the Joplin region has ever been subjected to such intense dynamic metamorphism as would be necessary to develop these "strain shadows" in the quartz. It is believed, therefore, that the phenomenon is due to internal stresses, probably resulting from the change from colloidal to crystalline condition. This internal stress probably has an important effect on the tendency of the rock to break so readily into sharp fragments.

Much of the rock is fossiliferous and the fossils, as has been stated, are largely those that originally were calcareous. They show all degrees of silicification, from pure limestone to pure chert. Such silicifications are almost invariably much more coarsely crystalline than the matrix in which they are embedded. The microscope also reveals what appears to be the remains of sponge spicules. Most of these consist of small spindle-shaped areas more coarsely crystalline than the rest of the rock. Some, however, are trident-shaped, as is shown in Plate XVI, *A*. The presence of the remains of sponge spicules is important as indicating the origin of the chert and will be considered in some detail in the discussion of that subject.

ANALYSES OF SAMPLES OF GRAY CHERT.

The results of chemical analyses of samples of gray chert follow.

Results of chemical analyses of the gray chert of the Joplin district.[a]

[E. A. Schneider, analyst.]

Constituent.	Percentage in sample—						
	A	B	C	D	E	F	G
SiO_2	98.17	98.92	98.71	99.46	99.23	98.60	99.13
Al_2O_3 and Fe_2O_3	.83	.48	.43	.29	.22	.52	.16
MgO	.01	.02	Trace.	Trace.	Trace.	.01	.24
CaO	.05	.03	.03	.04	.02	.10	Trace.
Loss on ignition	.78	.42	.50	.34	.50	.40	.20
	92.84	99.87	99.69	100.13	99.97	99.62	99.50

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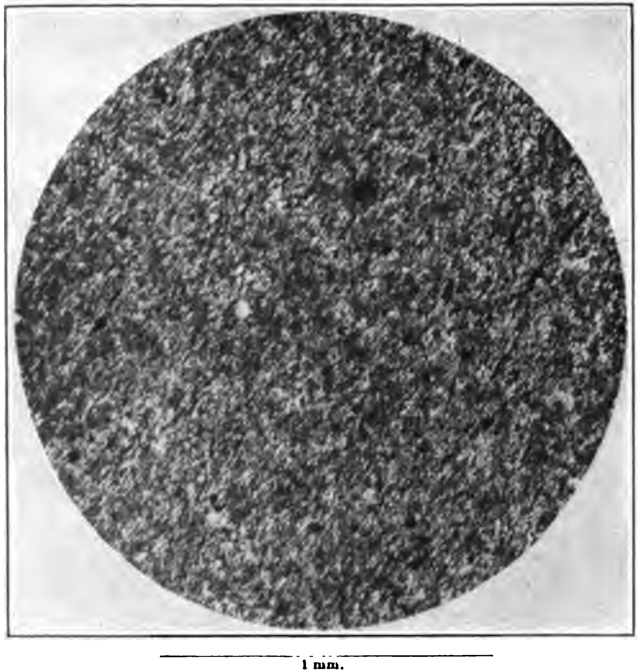
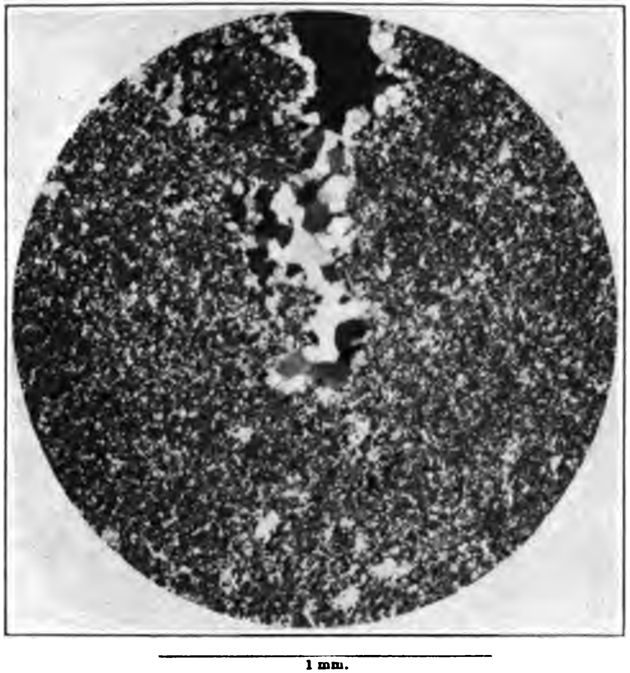
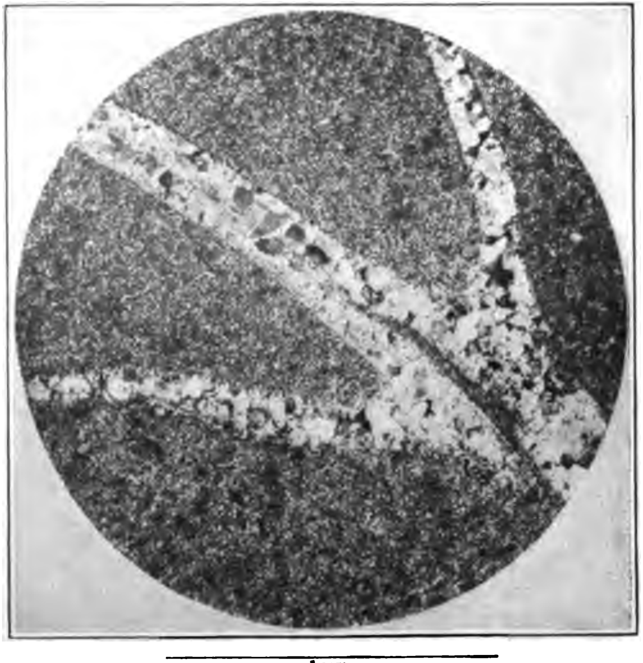
[a] Clarke, F. W., Analyses of rocks and minerals from the laboratory of the United States Geological Survey, 1880-1908: U. S. Geol. Survey Bull. 419, 1910, p. 186.



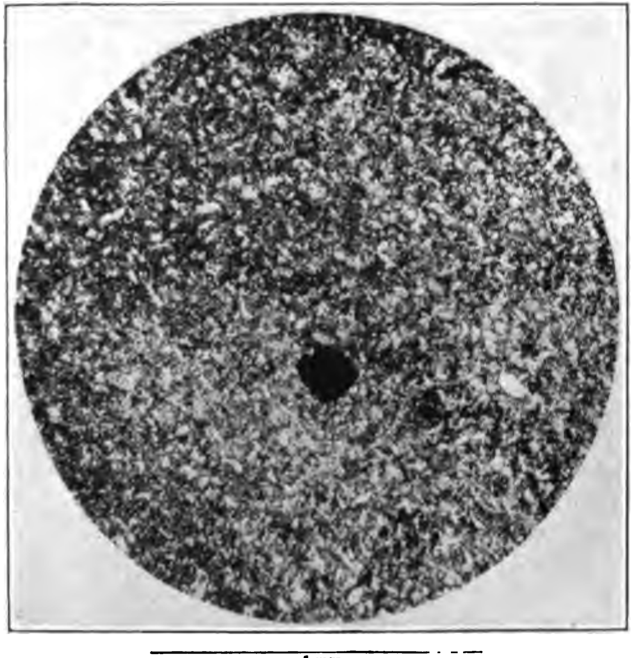
A. PHOTOMICROGRAPH, CROSSED NICOLS, OF THIN SECTION OF DENSE GRAY CHERT. TEXTURE IS TYPICALLY CRYSTALLINE, AND MUCH OF THE ROCK IS SO FINE GRAINED THAT THE MICROSCOPE FAILS TO RESOLVE IT.



B. PHOTOMICROGRAPH, CROSSED NICOLS, OF THIN SECTION OF THE COARSER GRAY CHERT. GRAINS ARE MUCH LARGER AT BORDERS OF SMALL OPENING OR VUG, IN UPPER PART OF SECTION, THAN ELSEWHERE.



A. PHOTOMICROGRAPH, CROSSED NICOLS, OF THIN SECTION OF FINE-GRAINED OLDER GRAY CHERT, SHOWING REMAINS OF TRIDENT-SHAPED SPONGE SPICULE.



B. PHOTOMICROGRAPH, CROSSED NICOLS, OF THIN SECTION OF JASPEROID. TEXTURE SHOWN, THOUGH CONSIDERABLY COARSER THAN THAT OF THE AVERAGE GRAY CHERT, IS FINER THAN MUCH OF THE JASPEROID.

The samples were collected in the Joplin district by W. P. Jenney, and represent localities as follows:

Sample A.—Unaltered chert, East Hollow, Belleville, Mo.
Sample B.—Partly altered chert, from the same locality as sample A.
Sample C.—Altered to "cotton rock," from the same locality as sample A.
Sample D.—Surprise mine, Joplin, Mo.
Sample E.—Blue chert, unaltered, Bonanza shaft, Galena, Kans.
Sample F.—From the same locality as sample E.
Sample G.—Altered chert, from the same locality as sample E.

These analyses of representative specimens of the gray chert show clearly that, whatever the origin and physical condition of the chert, it is at present made up almost wholly of silica. The small loss on ignition also indicates that by far the greater part of the silica is in the form of quartz. Field observations by the writer in many places in the Joplin district and in other localities in Missouri afforded conclusive evidence that much of this rock represents a replacement of the limestone by silica. The analyses, supplementing as they do the microscopic examinations, show how completely parts of the limestone may be silicified.

The analyses, when compared with those of the younger black chert or jasperoid, show 3 to 5 per cent higher silica content, less iron and alumina, and less loss upon ignition, as well as no carbonaceous matter.

ORIGIN OF GRAY CHERT.

It may not be out of place to make a few statements in regard to the probable origin and manner of deposition of the chert. As has been stated, it is believed that the gray chert is of two periods of deposition or formation, one contemporaneous with and one later than the deposition of the limestone in which it occurs. Some of the heavy beds of chert are continuous over wide areas and occur at definite horizons in the limestone. These are believed to have been deposited contemporaneously with the limestone and to have been formed from siliceous material deposited with the limestone.

Another part of the chert is clearly a replacement of the limestone in which it occurs. Limestone and chert are found in every gradation from pure limestone to pure chert. It is also well established that the alteration of limestone into chert has taken place on a large scale in many places in the Mississippi Valley, and as a result great masses of limestone have been replaced by chert.

Much of the nodular chert is also believed to be a replacement of limestone and consequently of secondary deposition. There are no means of determining how much of the chert is of primary nor how much is of secondary deposition, but it is believed that a large percentage of the total is of secondary origin and represents a replacement of the limestone.

The ultimate origin of a large part of the original chert is believed to have been siliceous materials, such as tests, shells, and spicules of sponges, which were deposited contemporaneously with the calcareous material that formed the limestone. That at least a part of the chert is of organic origin is conclusively shown by the fragments of sponge spicules frequently found in much of the chert. Although it is believed that much of the chert is of organic origin, the writer recognizes the possibility that a part, possibly the greater part, may represent chemically precipitated silica derived from clay and other silicates in solution in the sea water during the deposition of the limestone. Probably, both processes were active, but there are no means of ascertaining which was the more important. The form and texture of the cherts indicate that, whatever the source of the silica, a large part of it was probably precipitated in the colloidal condition. Perhaps the masses of siliceous tests and shells formed centers around which the dissolved silica collected, there being at the same time more or less solution of the tests and shells themselves.

The secondary gray chert, although in composition, texture, and general appearance closely similar to the original gray chert, is much younger and was deposited under entirely different conditions. That it is a replacement of the limestones and dolomites by silica is clearly shown by the fact that the calcareous rock is found in every stage of replacement from pure dolomite or limestone to pure gray chert. Moreover, the chert contains an abundance of originally calcareous fossils such as crinoids, brachiopods, and corals, that are now entirely silicified. Similar replacement of limestones and dolomites by chert is common throughout the whole Mississippi valley, whole beds of calcareous rock having been completely replaced by chert.[a] Much of the nodular chert is also known to be secondary, and such nodules, so far as the writer is aware, can in no way be distinguished from the nodules of the original and older chert. There are, therefore, no satisfactory means of determining definitely what proportion of the Joplin cherts represents replacement of calcareous rocks and what is of original deposition and therefore contemporaneous with the limestone.

The source of the silica of the secondary chert may have been any siliceous rock exposed to the action of solvent waters. Much of it has been dissolved, carried, and deposited by the ground water, which in this region always carries in solution an appreciable amount of silica. The conditions determining silicification are not known, but they assuredly are varied. Much silicification has taken place at considerable depths underground and seems to be in no way connected with the agencies of weathering. On the other hand, as shown by Bain and Ulrich,[a] much of the chert "as it now occurs is largely the segregation of siliceous matter under conditions of slow subaereal decomposition of the limestone." As regards the Joplin district, this last mode of formation, although applicable to the chert in natural outcrops or near surface, is certainly not applicable to that associated with the ores. As these theories of the origin of the chert involve processes that are still active, it is altogether probable that replacement is going on and that the calcareous rock is now being altered to chert.

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[a] Ball, S. H., and Smith, A. F., The geology of Miller County: Missouri Bureau of Geology and Mines, vol. 1, 2d ser., 1903, pp. 145-147.

JASPEROID OR BLACK CHERT.

GENERAL CHARACTERISTICS.

Aside from the carbonate rocks, the most prominent gangue material in the so-called "sheet ground" and much of the "open ground" is a dense, fine-grained, dark colored or black, hard, siliceous rock which closely resembles chert. This material has been designated by various names, such as ozarkite, cherokite, secondary chert, and jasperoid. The term jasperoid was first applied in 1907 by Smith and Siebenthal[b] in their folio on the Joplin district, and is at present the name used by the United States Geological Survey.

Jasperoid occurs extensively in all the sheet-ground deposits, in which it usually forms the matrix for more or less of the ore. It also serves as a matrix for much of the chert breccia in all types of mines, and in many places layers of it are intercalated in the dense older gray chert. Occasionally it shows a decided rhythmic banding due to concentration of certain of its constituents in narrow alternating bands. The greater part of the jasperoid, however, is dense, fine-grained, fairly uniform, and black, without indications of banding. The rock, although somewhat coarser in texture than the older gray chert, presents practically all the physical properties of true chert, being hard and brittle and breaking with pronounced conchoidal fracture into knife-edged and sharp-pointed fragments, although possibly not quite as readily as the true chert. These are the characteristics that make the dust from the rock so injurious to the lungs of miners.

CHARACTERISTICS AS SEEN BY THE MICROSCOPE.

In thin section under the microscope the purest or most highly siliceous phases of the jasperoid are seen to be composed of a very fine grained groundmass of approximately equidimensional interlocking grains of quartz that vary in size from 0.02 mm. in diameter down to sizes that the microscope fails to resolve. Irregularly distributed throughout this groundmass are numerous larger elongated and roughly rectangular grains of quartz that vary much in size, being 0.03 to 0.08 mm. in longest direction, which is usually two to three times that of the shortest grains, so that the texture of the chert roughly suggests the so-called ophitic texture of certain igneous rocks. Except a few small specks or grains of dolomite or limestone and particles of sphalerite and pyrite, the only other feature revealed by the microscope is the presence of innumerable minute black, opaque particles which chemical tests show to be carbonaceous matter. This is rather evenly distributed throughout the rock and accounts for its dark color. Although there is much exceedingly fine-grained material in the jasperoid, the rock as a whole has a decidedly coarser texture than the older gray chert. The microscope also corroborates the macroscopic evidence that in many, if not most, instances the jasperoid is a replacement of the limestone or dolomite. Specimens showing almost every step from calcareous rock to jasperoid are easily obtainable. Much of the jasperoid contains, in addition to the ragged sphalerite grains, numerous rhombohedral crystals of dolomite. Smith and Siebenthal[a] say:

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[a] Bain, H. F., and Ulrich, E. O., The copper deposits of Missouri: U. S. Geol. Survey Bull. 267, 1905, pp. 27-30.

[b] Smith, W. S. T., and Siebenthal, C. E. Geol. Atlas U. S., Joplin district folio (No. 148): U. S. Geol. Survey, 1907, p. 14.

> At numerous places dolomite occurs in the jasperoid in rhombohedrons, usually with clear-cut crystal boundaries. Many of these contain as inclusions, minute crystals of quartz and rarely sphalerite is found as a rule in well-defined crystals. Sphalerite occurring as one of the constituents of jasperoid usually presents a smooth outline where adjacent to dolomite, but where bordered by quartz its boundaries, even where it shows general crystal outlines, almost invariably yield to those of the quartz so as to produce minute irregularities at least.

The general appearance of the section of the finer-grained jasperoid as seen under the polarizing microscope is shown in Plate XVI, *B* (p. 103). The thin section from which this photomicrograph was made was from the fine-textured phase of the rock, and shows much finer grain than is found in the average jasperoid.

ANALYSES OF JASPEROID.

The following chemical analyses of jasperoid were made in the laboratory of the Geological Survey and show the composition of the purer phases of the rock.

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[a] Smith, W. S. T., and Siebenthal, C. E., Geol. Atlas U. S., Joplin district folio (No. 148), U. S. Geol. Survey, 1907, p. 14.

Results of analyses of jasperoid from the Joplin district, Mo.[a]

Constituent.	Percentage in sample—			
	A[b]	B[c]	C[c]	D
SiO_2	95.26	95.77	97.33	94.72
Al_2O_3	.57	1.84	1.89	4.00
Fe_2O_3	None.			
FeO	.69			
MnO	None.			
MgO	.05	.24	.09	Trace.
CaO	.25	.54	.11	1.18
Water		1.17	.77	
Organic matter	Large.			

[a] Bain, H. F., Van Hise, C. R., and Adams, C. I., Lead and zinc deposits of the Ozark region: Twenty-second Annual Rept., U. S. Geol. Survey, pt. 2, 1902, p. 121.

[b] George Steiger, analyst.

[c] L. G. Eakins, analyst. See Clarke, F. W., Analyses of rocks from the laboratory of the United States Geological Survey, 1880-1903: U. S. Geol. Survey Bull. 228, 1904, p. 297.

If, as is believed by Smith and Siebenthal,[a] the black chert represents a part of the limestone that was replaced by silica and other material when the ores were deposited, the analyses show that the replacement has been all but complete. Field observations and microscopic study of much black chert, as well as the analyses themselves, indicate that the material chosen for chemical study was pure jasperoid. Material ranging from almost unaltered limestone to typical black chert is abundantly available in the district.

Comparison of these analyses of the black chert with those of the older gray chert shows that the two types of rock differ mainly in the silica content, the older chert containing 3 to 5 per cent more silica than the black rock. The black chert, on the other hand, contains appreciably more iron, alumina, lime, and organic matter than does the older gray chert.

ORIGIN OF JASPEROID.

The investigations herein recorded throw some light on the origin of the jasperoid and led the writer to agree with the hypothesis offered by Smith and Siebenthal,[a] whose conclusions are as follows:

> It [the jasperoid] occurs locally in lenticular forms such as characterize the occurrence of limestone in chert, and the manner of its occurrence in the sheet ground suggests the replacement of sheets and lenses of limestone. More definite evidence is found in the fact that all stages of the process of change from unaltered limestone to jasperoid have been observed, both megascopically and microscopically. Corroborative facts are the occurrence here and there of fossils, particularly crinoid stems, in typical jasperoid as well as the presence of stylolites in a somewhat calcareous jasperoid at the Jack Johnson mine near Chitwood.

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[a] Smith, W. S. T., and Siebenthal, C. E., Geol. Atlas U. S., Joplin district folio (No. 148), U. S. Geol. Survey. 1907, p. 14.

The general process of replacement, as shown by the microscope, is as follows: First, a few scattered crystals of quartz appear in the limestone; with increase in the proportion of quartz the limestone decreases until in its later stages the rock consists chiefly of a granular aggregate of quartz, with scattered, ragged grains of calcite, mere remnants of the former limestone. Finally, even these disappear.

This explanation of the origin of jasperoid accounts for the scarcity of limestone in the ore deposits. In the sheet ground, limestone is rarely found, jasperoid occupying those positions in which limestone would normally be looked for. In the breccia, limestone blocks are sometimes found, but the chert breccia of many mines appears to be wholly free from limestone, although dolomite and chert may border it on one side and limestone with chert on the other. The replacement of limestone by jasperoid would not only account for the apparent absence of much or most of the limestone to be expected in the chert breccias, especially in those mines which are practically free from dolomite, as the Oronogo mines and those in the hard sheet ground, but would also explain the fractured beds of chert sometimes seen suspended in a matrix of jasperoid.

Most of the geologists who have studied the Joplin ore deposits believe that the silica forming the jasperoid was deposited originally in colloidal form. The first writer to suggest that it was deposited as colloidal silica and later changed to the crystalline condition is Bain,[a] who studied the district in 1900. The premises on which he bases his hypothesis were stated by him substantially, as follows: Crystals of sphalerite completely inclosed in jasperoid possess well-defined crystal outlines with no indication that they were deposited in cavities; and the microscope does not reveal any such concentric laminations in the matrix immediately surrounding the crystals as commonly surround crystals that have in their growth crowded aside the matrix.

In 1915 Siebenthal,[b] after an extended study of the ore deposits of the district, reviewed the work of Bain and concurred with him in his conclusions in regard to the original colloidal conditions of the silica of the jasperoid. The writer believes not only that the silica of the jasperoid was deposited in the colloidal condition but also that the silica of the older chert was to a large extent similarly deposited. Of course the deposition took place long before the period of mineralization and the development of the jasperoid.

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[a] Bain, H. F., Van Hise, C. R., and Adams, C. I., work quoted, pp. 106, 108.

[b] Siebenthal, C. E., The origin of the zinc and lead deposits of the Joplin region: U. S. Geol. Survey Bull. 606, 1915, pp. 181-183.

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INDEX.

A.

Page.
Agricola, on diseases of miners 81, 82
Australia, miners' consumption in 65
  mines, rock dust in air of 50
  See also Bendigo, Australia.

B.

Bain, H. F., on origin of chert 105
  on origin of jasperoid 108
Beattie, —, investigations of 84
Beiger fluid for eliminating rock dust, tests with 93
Bendigo, Australia mines, miners' phthisis in 94
Blasting, as source of rock dust 20, 21, 30, 48, 57, 92
  in Joplin district, explosives for 13
  rock dust from, elimination of 48, 59
    proportion of 38, 39
Blowing of holes, as source of rock dust 21, 44, 57
  rock dust from, proportion of 33, 36, 37
  rules prohibiting, need of 53, 59
"Bowlder popping" as source of rock dust 20, 21, 57
  definition of 15
  objections to 72
  rules against, need of 53, 59, 80
Breast hole, rock dust from 33–41, 45, 46
    view of 46
Bucket, dumping, rock dust from, proportion of 34, 35
Buckley, E. R., on classes of chert 100
Buehler, H. A., on classes of chert 100
Burch, I. L., acknowledgments to 3
  work of 51, 52
Bureau of Mines, investigations of 3, 62, 97, 98
Burrell, G. A., acknowledgments to 3
Butte, Mont., copper mines of, investigations in 1, 2

C.

Carbide lamps, danger from, investigation of 73
Carbon dioxide in mine air 17
Carbon monoxide in mine air 17
Carborundum dust, effect on lungs 84
Carterville, Mo., educational work at 63
  mining plant near, views of 12
  ore deposits, rocks associated with 8
  prospecting methods near, view of 10
  sheet-ground mines near, map showing 6
Chalcedony in chert, properties of 101, 102
Change houses, improvements in 74, 75
  maintenance of, legislation regarding 56
  views of 10, 76
Chert, classes of 99, 100
  gray, analyses of 102
    characteristics of 100, 101
    geologic occurrence of 99
    microscopic view of 102
    minerals associated with 102
    origin of 103, 104
  See also Jasperoid.

Page.
Children, prevention of tuberculosis among, need of 77, 79
Coal dust, effect on lungs 83, 84, 95
Coal miners, pulmonary diseases among 83, 84
Compressor air, as cause of disease, investigation of 74
Consumption. *See* Miners' consumption; Tuberculosis.
Cornwall, metal mines in, rock dust in 51, 60
    sanitation of, recommendations on 60
  miners in, death rate of 59, 60, 84
    lung diseases among 65, 84
Costs of mining and milling in Joplin district 18, 19
Cumpston, J. H. L., on pulmonary diseases among Australian miners 94

D.

De Fenton, J., cited 81, 82
Diembroek, —, cited 81
Dispensary in Joplin district, need of 77–80
Drill cuttings, tests of, results of 25
Drill holes, method of placing, determination 10
    figure showing 12
  removal of cuttings from 13
Drills in Joplin district, types of 8
  water-injection, advantage of 96
Drilling, as source of rock dust 20, 30, 57
  in Joplin district, methods of 8, 9, 12
    regulations regarding 93
  results of tests of 47
  rock dust from, abatement of, recommendations for 53
    factors affecting 45, 46, 47
    proportion of 33–43
    results of tests of 92
    variations in 45
    view of 46
Drinking devices, sanitary, in mines, legislation on 56
Duenweg, Mo., dry-and-change house near, view of 10
  sheet-ground mines near, map showing 6
Dust sampler, details of 22, 23
  errors in, factors affecting 23, 24
  operation of 24
  testing for leaks in 24
  views of 22
"Dusty mines," classification of, factors determining 50

E.

Educational work in Joplin district, methods of 54
  scope of 63
  value of 79
Emery dust, injury to lungs from 84
England, air in mines of, rock dust in 50, 51
Explosives, fumes of, as cause of phthisis 87, 88
  used in Joplin district 13

F. Page.

- Fans in dusty mines, value of 71
- Fieldner, A. C., acknowledgments to 4
- Flint. *See* Chert.
- Flint dust, injury to lungs from 84
- Fumay, France, slate miners of, silicosis among 83

G.

- Ganister, miners of, death rate among 84
- Germany, industrial dusts in, dangers from 95
  - pulmonary tuberculosis in 95
- Glass grinders, death rate of 95
- Gold mines in South Africa, mortality in 84, 85

H.

- Haldane, J. S., cited 50, 97
  - investigations of 84
- Hamaide, —, cited 83
- Hammermen, silicosis among 91
- Harlan, C. M., acknowledgments to 3
  - work of 51, 52
- Hartz mines, Germany, consumption in 95
- Heading work, breaking ground in, figure showing 11
  - definition of 10
  - description of 10, 11
  - drilling in 12
- Higgins, Edwin, investigations of 1
- Hippocrates, cited 81
- Hirt, Ludwig, on pulmonary tuberculosis in Germany 95
- Hoisting in sheet-ground mines, methods of 15
- Holmes, W. W., acknowledgments to 3
  - work of 51, 52
- Hoover, H. C., cited 81, 82
- Hoover, L. H., cited 81, 82
- Hotchkiss, S. C., investigations of 1, 97, 98
- Hovey, E. O., cited 101

I.

- Insurance, sickness, need of 78, 79
  - value of 80
- Iron-ore miners, pulmonary diseases among 83

J.

- Jasperoid, analyses of 107
  - properties of 105
  - geologic occurrence of 99, 105, 107, 108
  - microscopic view of 102
  - origin of 107, 108
  - structure of 106
- Joplin district, Mo., development methods in, description of 8, 9
  - view of 10
  - educational work at 63
  - investigations in, results of 1
  - mines of, production in 1914 4
  - ore deposits in 7, 8

L.

- Laney, F. B., investigations of 98
- Lanza, A. J., investigations of 31, 98
  - work of 1, 54
- Lead, concentrate, production of, in Joplin district 4
- Lead mines, inspection of, legislation on 55
- Lead poisoning, investigations of 73
- Leggi, —, investigations of 84
- Loehniss, G. E., cited 81
  - lungs, dust particles in, protection against 89, 90
  - size of 32, 58
  - structure of 89
  - effect of rock dust on 83, 90, 91
- Lung diseases of miners, classes of 81, 82
  - in Joplin district, prevalence of 79
  - remedies for 79
  - *See also* Miners consumption; tuberculosis.

M.

- Martin, J. S., cited 97
  - investigations of 84
- Metal mines, sanitary condition of, investigations of 97, 98
  - *See also* Gold mines; Lead mines· Zinc mines.
- Milling in Joplin district, cost of 18, 19
- Mine air, impurities in, as factor in miners' phthisis 87
  - sampling of, conditions governing 51
  - results of tests of 34–43, 92
- Mine inspection, legislation regarding 55, 56
- Mine inspectors, cooperation of 98
  - duties of, legislation regarding 55, 56
- Mine operators, cooperation of 54, 98
  - organization of 52
- Mineral dust, danger from breathing, early records of 83
  - *See also* Rock dust.
- Miners, cooperation of 1
  - methods of obtaining 54
  - physical examination of 62, 63
  - results of 57, 68
  - *See also* Rock-drill miners.
- Miners' consumption, causes of 65, 84
  - definition of 88
  - miners affected with 67, 68, 69
  - nature of 65
  - prevalence of 85, 86
  - prevention of, recommendations for 79, 80
  - stages of 70, 90
  - symptoms of 65, 85, 86
  - treatment for 78
  - tuberculosis infection in, dangers of 65, 77, 78
  - proportion of 67–70
  - types of 87
- Miners' diseases, early records of 81, 82
  - *See also* diseases named.
- Miners' houses, heating of 76
  - poor, causes of 75
  - sanitary conditions in 76
  - ventilation of 77
  - views of 76, 78
  - water supply for 76
- Miners' Phthisis. *See* Miners' consumption.
- Mining by stope and heading, definition of 10
  - description of 11, 12
  - drilling in 12
  - figure showing 11
- Mining methods in Joplin district 9–16
  - efficiency of 19
- Missouri, cooperative work with 52
  - mining laws of 55–57

N. Page.

National association for the prevention of tuberculosis, action by ... 97
New Zealand, rock dust prevention in mines of, regulations regarding ... 61
Nitrogen in mine air, proportion of ... 17

O.

Oliver, Thomas, on effect of rock dust on lungs ... 83, 85, 87
  on occupational diseases ... 83
Oronogo, Mo.; sheet-ground mines near, map showing ... 6
Overwork as cause of tuberculosis ... 74, 79
Oxygen in mine air, proportion of ... 17

P.

Pillars in sheet-ground mines, arrangement of, figure showing ... 10
  precautions in ... 14
Pliny the Elder, cited ... 81
Poisoning, metallic, in mines, investigations of ... 78
Porto Rico, Mo., sheet-ground mines near, map showing ... 6
Potters, diseases of ... 83
Privies, sanitary, in mines, need of ... 72, 73
Prospecting in Joplin district, method of ... 8, 9
  view of ... 10
Prosperity, Mo., sheet-ground mines near, map showing ... 6

Q.

Quartz dust, injury to lungs from ... 84

R.

Rand, South Africa, gold mines of, drilling in, regulations regarding ... 61, 93
  improvements in ... 94
  mortality in ... 84, 85
  rock dust in ... 50, 51, 92
  miners' phthisis in, investigations of ... 65, 85–88
  proportion of ... 91
  rock drill miners in, fatality rate of ... 87
Respirators, early use of ... 81
  objections to ... 71, 93, 96, 97
Respiratory system, physiology of ... 89
Rice, G. S., acknowledgments to ... 3
Robertshaw, —, investigations of ... 84
Rock-drill miners, fatality rate of ... 87
  miners' consumption among, prevalence of ... 86, 91
Rock dust, in mine air, abatement of, legislation on ... 53, 55
  methods for ... 58, 59, 71, 72, 97
  causes of ... 20
  composition of ... 57, 58
  early records of ... 82
  proportion of ... 58
  protection against, measures for ... 89, 90
  size of particles of ... 30–32, 58, 92
  suspension of, determination of ... 28, 30, 31
  factors governing ... 31, 58
  microscopic examination of, results of ... 26
  relation of, to lung diseases ... 64, 67, 83
  samples of, description of ... 26, 27
  views of ... 26, 28

Rock dust—Continued. Page.
  sampling of ... 22, 88, 92
  conditions governing ... 34–43
  results of ... 34–43
Roof and pillar trimming, as source of rock dust ... 20, 21, 57
  rock dust from, proportion of ... 33
Roof hole, drilling of, rock dust from ... 36–39, 45, 46

S.

Sampling of mine air. *See* Dust sampler.
Sandstone dust, effect on lungs ... 84
Sanitation in mines, efforts for ... 52
  need of ... 67
  recommendations on ... 53, 54, 60, 92
Schools, medical inspection in, need of ... 80
Sheet ground in Joplin district, character of ... 8
Sheet-ground mines, arrangement of pillars in ... 14
  costs of mining in ... 18
  investigations in, scope of ... 3
  mining methods in, classes of ... 10
  relation of siliceous dust to ... 20
  plan of workings in, figure showing ... 10
  rock dust in, factors causing ... 57
Sheet-ground mining district, map showing ... 5, 6
Shoveler, work of ... 14, 15
  need of regulating ... 80
Shoveling, as source of rock dust ... 20, 21, 30, 49, 57
  rock dust from, abatement of ... 49
  proportion of ... 33–43
Siebenthal, C. E., cited ... 6, 105, 106
  on origin of jasperoid ... 107, 108
Silica dust, injury to lungs by ... 84
Silica in gray chert, proportion of ... 102, 103
Siliceous dust. *See* Rock dust.
Silicosis, early records of ... 81
Silicosis. *See also* Miners' consumption.
Smith, W. S. T., cited ... 105, 106
  on origin of jasperoid ... 107
Smithsonite, production of, in Joplin district ... 4
Sommerfeld, Theodor, on pulmonary tuberculosis in Germany ... 95
South Africa. *See* Rand, South Africa.
Southwestern Missouri Mine Safety and Sanitation Association, organization of ... 52
Sputum, sampling of, difficulties in ... 68
"Squibbing," as source of siliceous dust ... 13, 20, 47, 57
  during underground work, elimination of, need of ... 48, 53, 59, 80
  objections to ... 13, 48, 72
  rock dust from, factors affecting ... 47, 48
  proportion of ... 33, 40–43
Stoke's law, formula for ... 28
Stonemasons, liability to lung diseases ... 83
Stope hole, drilling of, rock dust produced by ... 33, 38, 39, 45
  with water, view of ... 46
  wet, blowing of, rock dust from ... 36, 37
"Stoper" drill, use of ... 96
Sugar, granulated, for filtering mine air ... 22
Summons, Walter, on miners' consumption ... 66, 94

T.

Temperature in sheet-ground mines ... 18
Thiessen, Reinhardt, acknowledgments to ... 4
  on rock dust from lungs ... 32
Thomas, R. A., investigations of ... 84

Page.

- Timbermen, silicosis among, 91
- Trammers, silicosis among, 91
- Tramming as source of siliceous dust, 20, 21, 57
  - in sheet-ground mines, method of, 15
- Tuberculosis, pulmonary, causes of, 67
  - with miners' phthisis, dangers of, 65, 66, 85, 86, 87, 91
    - frequency of, 89
  - investigations of, 64, 65, 67, 68, 69
  - sanitarium for, need of, 77

U.

- Ulrich, E. O., on origin of chert, 105
- Union of South Africa, commissions appointed by, 87, 88
  - *See also*, Rand, South Africa.

V.

- Ventilation in metal mines, better need for, 53, 58
    - need of, 82
  - in sheet-ground mines, 16
    - effect of, on rock dust, 44
  - of miners' houses, 77

W.

- Wash house, maintenance of, legislation on, 56, 57
  - provision for, legislation regarding, 56, 57
- Water, drinking, in mines, improvement in, 72
- Water drill, view of, 46
- Water for drilling, abatement of rock dust, by, 46, 49, 50, 93
    - equipment for, 59
      - efficiency of, 71
      - maintenance of, legislation regarding, 55
      - precautions in use of, 72
    - need of, 58
    - regulations regarding, 93
    - rules requiring, need of, 53
  - for miners' houses, 76
  - in sheet-ground mines, amount of, 16
- Webb City, Mo., educational work at, 63
  - housing conditions near, 75
  - sheet-ground mines near, map showing, 6
  - physical examination of miners at, 62, 64
- Witwatersrand, South Africa. *See* Rand, South Africa.
- Workmen's compensation laws, establishment of, 79
  - value of, 80

Z.

- Zinc poisoning, investigations of, 73
- Zinc concentrate, production of, in Joplin district, 4
- Zinc mines, inspection of, legislation regarding, 55
  - tailing heaps of, views of, 12